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Fac-simile of Medal Awarded to The Mineral Industry
by the
Société d'Encouragement pour l'Industrie Nationale de France,
in recognition of
Its Services to the World's Industry and Commerce.

THE MINERAL INDUSTRY,

ITS

STATISTICS, TECHNOLOGY AND TRADE,

IN THE

UNITED STATES AND OTHER COUNTRIES

TO THE END OF

1898.

EDITED BY

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American Statistical Association, etc.

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SUPPLEMENT TO VOLUMES I. TO VI.

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CONTENTS OF VOLUME VII.

[Articles marked with an asterisk (*) are illustrated.]

INTRODUCTION.

ALUMINUM AND ALUM.

AMMONIA AND AMMONIUM SULPHATE.

ANTIMONY.

ARSENIC.

ASBESTOS.

ASPHALTUM.

BARYTES.

BISMUTH.

BORAX.

BROMINE.

CALCIUM CARBIDE.

CARBORUNDUM.

CEMENT.

CHROMIUM AND CHROME ORE.

CLAY.

GOLD AND SILVER.

GRAPHITE.

GYPSUM.

IRON AND STEEL.

MAGNESIUM AND MAGNESITE.

MANGANESE.

MICA.

MOLYBDENUM.

MONAZITE.

NICKEL AND COBALT.

OCHER AND OXIDE OF IRON PIGMENTS.

PETROLEUM.

SALT.

SILICA.

SLATE.

SODA AND SODIUM.

STONE.

SULPHUR AND PYRITES.

TUNGSTEN.

ZINC AND CADMIUM.

PROGRESS IN ORE DRESSING IN 1898.

THE PROGRESS OF METALLOGRAPHY IN 1898.

THE MINING STOCK EXCHANGES IN 1898.

MINERAL STATISTICS OF FOREIGN COUNTRIES.

THE UNITED STATES.

THE BUYERS' MANUAL.

THE PROFESSIONAL DIRECTORY.

TABLES FOR CONVERTING UNITED STATES WEIGHTS AND MEASURES TO METRIC.

LINEAR.					CAPACITY.							
Inches to Millimeters.	Feet to Meters.	Yards to Meters.	Miles to Kilometers.		Drams to Cubic Centimeters.	Ounces to Milliliters.	Quarts to Liters.	Gallons to Liters.	Cubic Inches to Cubic Centimeters.	Cubic Feet to Cubic Meters.	Cubic Yards to Cubic Meters.	Bushels to Hectoliters.
25.4000	0.304801	0.914402	1.60935	= 1 =	3.70	29.57	0.94636	3.78544	16.387	0.02832	0.765	0.35242
50.8001	0.609601	1.828804	3.21869	= 2 =	7.39	59.15	1.89272	7.57088	32.774	0.05663	1.529	0.70485
76.2001	0.914402	2.743205	4.82804	= 3 =	11.09	88.72	2.83908	11.35632	49.161	0.08495	2.294	1.05727
101.6002	1.219202	3.657607	6.43739	= 4 =	14.79	118.30	3.78544	15.14176	65.549	0.11327	3.058	1.40969
127.0002	1.524003	4.572009	8.04674	= 5 =	18.48	147.87	4.73180	18.92720	81.936	0.14158	3.823	1.76211
152.4003	1.828804	5.486411	9.65608	= 6 =	22.18	177.44	5.67816	22.71264	98.323	0.16990	4.587	2.11454
177.8003	2.133604	6.400813	11.26543	= 7 =	25.88	207.02	6.62452	26.49808	114.710	0.19822	5.352	2.46696
203.2004	2.438405	7.315215	12.87478	= 8 =	29.57	236.59	7.57088	30.28352	131.097	0.22654	6.116	2.81938
228.6004	2.743205	8.229616	14.48412	= 9 =	33.28	266.16	8.51724	34.06896	147.484	0.25485	6.881	3.17181

SQUARE.					WEIGHT.			
Square Inches to Square Centimeters.	Square Feet to Square Decimeters.	Square Yards to Square Meters.	Acres to Hectaers.		Grains to Milligrams.	Avoirdupois Ounces to Grams.	Avoirdupois Pounds to Kilograms.	Troy Ounces to Grams.
6.452	9.290	0.836	0.4047	=1=	64.7989	28.3495	0.45359	31.10348
12.903	18.581	1.672	0.8094	=2=	129.5978	56.6991	0.90719	62.20696
19.355	27.871	2.508	1.2141	=3=	194.3968	85.0486	1.36078	93.31044
25.807	37.161	3.344	1.6187	=4=	259.1957	113.3981	1.81437	124.41392
32.258	46.452	4.181	2.0234	=5=	323.9946	141.7476	2.26796	155.51740
38.710	55.742	5.017	2.4281	=6=	388.7935	170.0972	2.72156	186.62089
45.161	65.032	5.853	2.8328	=7=	453.5924	198.4467	3.17515	217.72437
51.613	74.323	6.689	3.2375	=8=	518.3914	226.7962	3.62874	248.82785
58.065	83.613	7.525	3.6422	=9=	583.1903	255.1457	4.08233	279.93133

1 chain = 20.1169 meters.
1 square mile = 259 hectaers.
1 fathom = 1.829 meters.
1 nautical mile = 1853.27 meters.
1 foot=0.304801 meter, 9.4840158 log.
1 avoir. pound = 453.5924277 gram.
15432.35639 grains = 1 kilogram.

TABLES FOR CONVERTING METRIC TO UNITED STATES WEIGHTS AND MEASURES.

LINEAR.					CAPACITY.							
Meters to Inches.	Meters to Feet.	Meters to Yards.	Kilometers to Miles.		Milliliters or Cubic Centiliters to Fluid Drams.	Centiliters to Fluid Ounces.	Liters to Quarts.	Dekaliters to Gallons.	Hektoliters to Bushels.	Cubic Centimeters to Cubic Inches.	Cubic Meters to Cubic Feet.	Cubic Merets to Cubic Yards.
39.3700	3.28083	1.093611	0.62137	= 1 =	0.27	0.338	1.0567	2.6417	2.8375	0.0610	35.314	1.308
78.7400	6.56167	2.187222	1.24274	= 2 =	0.54	0.676	2.1134	5.2834	5.6750	0.1220	70.629	2.616
118.1100	9.84250	3.280833	1.86411	= 3 =	0.81	1.014	3.1700	7.9251	8.5125	0.1831	105.943	3.924
157.4800	13.12333	4.374444	2.48548	= 4 =	1.08	1.352	4.2267	10.5668	11.3500	0.2441	141.258	5.232
196.8500	16.40417	5.468056	3.10685	= 5 =	1.35	1.691	5.2834	13.2085	14.1875	0.3051	176.572	6.540
236.2200	19.68500	6.561667	3.72822	= 6 =	1.62	2.029	6.3401	15.8502	17.0250	0.3661	211.887	7.848
275.5900	22.96583	7.655278	4.34959	= 7 =	1.89	2.368	7.3968	18.4919	19.8625	0.4272	247.201	9.156
314.9600	26.24667	8.748889	4.97096	= 8 =	2.16	2.706	8.4534	21.1336	22.7000	0.4882	282.516	10.464
354.3300	29.52750	9.842500	5.59233	= 9 =	2.43	3.043	9.5101	23.7753	25.5375	0.5492	317.830	11.771

SQUARE.					WEIGHT.			
Square Centimeters to Square Inches.	Square Meters to Square Feet.	Square Meters to Square Yards.	Hectares to Acres.		Kilograms to Grains.	Hectograms to Ounces Avoirdupois.	Kilograms to Pounds Avoirdupois.	Grams to Ounces Troy.
0.1550	10.764	1.196	2.471	= 1 =	15432.36	3.5274	2.20462	0.03215
0.3100	21.528	2.392	4.942	= 2 =	30864.71	7.0548	4.40924	0.06430
0.4650	32.292	3.588	7.413	= 3 =	46297.07	10.5822	6.61386	0.09645
0.6200	43.055	4.784	9.884	= 4 =	61729.43	14.1096	8.81849	0.12860
0.7750	53.819	5.980	12.355	= 5 =	77161.78	17.6370	11.02311	0.16075
0.9350	64.583	7.176	14.826	= 6 =	92594.14	21.1644	13.22773	0.19290
1.0850	75.347	8.372	17.297	= 7 =	108026 49	24 6918	15.43235	0.22505
1.2400	86.111	9.568	19.768	= 8 =	123458.85	28.2192	17.63697	0.25721
1.3950	96.874	10.764	22.2	= 9 =	138891.21	31.7466	19.84159	0.28936

The only material standard of customary length authorized by the U. S. Government is the Troughton scale, whose length at 59°.62 Fahr. conforms to the British standard. The yard in use in the United States is therefore equal to the British yard.

The only authorized material standard of customary weight is the Troy pound of the Mint. It is of brass of unknown density, and therefore not suitable for a standard of mass. It was derived from the British standard Troy pound of 1758 by direct comparison. The British avoirdupois pound was also derived from the latter, and contains 7,000 grains Troy.

The grain Troy is therefore the same as the grain avoirdupois, and the pound avoirdupois in use in the United States is equal to the British pound avoirdupois.

The British gallon = 4·54346 liters.

The British bushel = 36·3477 liters.

By the concurrent action of the principal Governments of the world an International Bureau of Weights and Measures has been established near Paris. Under the direction of the International Committee, two ingots were cast of pure platinum-iridium in the proportion of 9 parts of the former to 1 of the latter metal. From one of these a certain number of kilograms were prepared, from the other a definite number of meter bars. These standards of weight and length were intercompared, without preference, and certain ones were selected as International prototype standards. The others were distributed by lot to the different Governments and are called National prototype standards.

The metric system was legalized in the United States in 1866.

The International Standard Meter is derived from the Metre des Archives, and its length is defined by the distance between two lines at 0° Centigrade, on a platinum-iridium bar deposited at the International Bureau of Weights and Measures.

The International Standard Kilogram is a mass of platinum-iridium deposited at the same place, and its weight *in vacuo* is the same as that of the Kilogramme des Archives.

The liter is equal to a cubic decimeter of water, and it is measured by the quantity of distilled water which, at its maximum density, will counterpoise the standard kilogram in a vacuum, the volume of such a quantity of water being, as nearly as has been ascertained, equal to a cubic decimeter.

Long ton: 2240 lb. avoirdupois	=1016	kilogram.	Barrel of petroleum	= 42 gal.	= 1·59	hectoliter.
Short ton: 2000 " "	= 907·2	"	" " salt	= 280 lb.	=127	kilogram.
Pound avoirdupois	= 453·6	grams.	" " lime	= 200 "	=90·720	"
Flask of mercury=76½ lb. avoir.	= 34·700	kilogram.	" " natural cement,	= 300 "	=136·080	"
Troy ounce	= 31·104	grams.	" " Portland cement,	= 400 "	=181·440	"
Gallon	= 3·785	litres	Gold coining value per oz. Troy	$20·6718	=$0·6646	per gram.
			Silver " " " " Troy	$1·2929	=$0·04157	" "

OFFICIAL UNITED STATES VALUES OF FOREIGN COINS, APRIL 1, 1899.

Country.	Standard.	Unit.	Value in U. S. Gold.	Coins.
			cts.	
Argentina.	Both	Peso	96·5	Gold: argentine ($4·824) and ½ argentine. Silver: peso and divisions.
Austria-Hungary	Gold	Crown	20·3	Gold: former system—4 florins ($1·929), 8 florins ($3·858), ducat ($2·287), and 4 ducats ($9·149). Silver: 1 and 2 florins. Present system—Gold: 20 crowns ($4·052) and 10 crowns ($2·026).
Belgium	Both	Franc	19·3	Gold: 10 and 20 francs. Silver: 5 francs.
Bolivia	Silver	Boliviano	43·4	Silver: boliviano and divisions.
Brazil	Gold	Milreis	54·6	Gold: 5, 10, and 20 milreis. Silver: ½, 1, and 2 milreis.
Canada	Gold	Dollar	100·0	
Central America				
Costa Rica	Gold	Colon	46·5	Gold: 2, 5, 10, and 20 colons ($9·307). Silver: 5, 10, 25, and 50 centimos.
British Honduras	Gold	Dollar	100·0	
Guatemala Honduras Nicaragua Salvador	Silver	Peso	43·4	Silver: peso and divisions.
Chile	Gold	Peso	36·5	Gold: escudo ($1·825), doubloon ($3·650), and condor ($7·300). Silver: peso and divisions.
China	Silver	Tael*	71·4	
Colombia	Silver	Peso	43·4	Gold: condor ($9·647) and double condor. Silver: peso.
Cuba	Both	Peso	92·6	Gold: doubloon ($5·017). Silver: peso.
Denmark	Gold	Crown	26·8	Gold: 10 and 20 crowns.
Ecuador	Silver	Sucre	43·4	Gold: condor ($9·647) and double condor. Silver: sucre and divisions.
Egypt	Gold	Pound	494·3	Gold: pound (100 piasters), 5, 10, 20, and 50 piasters. Silver: 1, 2, 5, 10 and 20 piasters.
Finland	Gold	Mark	19·3	Gold: 20 marks ($3·859), 10 marks ($1·93).
France	Both	Franc	19·3	Gold: 5, 10, 20, 50, and 100 francs. Silver: 5 francs.
German Empire	Gold	Mark	23·8	Gold: 5, 10, and 20 marks.
Great Britain	Gold	Pound	486·65	Gold: sovereign (pound sterling) and ½ sovereign.
Greece	Both	Drachma	19·3	Gold: 5, 10, 20, 50 and 100 drachmas. Silver: 5 drachmas.
Haiti	Both	Gourde	96·5	Silver: gourde.
India	Silver	Rupee	20·6	Gold: mohur ($7·105). Silver: rupee and divisions.
Italy	Both	Lira	19·3	Gold: 5, 10, 20, 50 and 100 lire. Silver: 5 lire.
Japan	Gold	Yen	49·8	Gold: 5, 10, and 20 yen. Silver: 10, 20, and 50 sen.
Liberia	Gold	Dollar	100·0	
Mexico	Silver	Dollar	47·2	Gold: dollar ($0·983), 2½, 5, 10, and 20 dollars. Silver: dollar (or peso) and divisions.
Netherlands	Both	Florin	40·2	Gold: 10 florins. Silver: ½, 1, and 2½ florins.
Newfoundland	Gold	Dollar	101·4	Gold: 2 dollars ($2·027))
Norway	Gold	Crown	26·8	Gold: 10 and 20 crowns.
Peru	Silver	Sol	43·4	Silver: sol and divisions.
Portugal	Gold	Milreis	108·0	Gold: 1, 2, 5, and 10 milreis.
Russia	Gold	Ruble	51·5	Gold: imperial 15 rubles ($7·718) and ½ imperial 7½ rubles ($3·859). Silver: ¼, ½, and 1 ruble.
Spain	Both	Peseta	19·3	Gold: 25 pesetas. Silver: 5 pesetas.
Sweden	Gold	Crown	26·8	Gold: 10 and 20 crowns.
Switzerland	Both	Franc	19·3	Gold: 5, 10, 20, 50 and 100 francs. Silver: 5 francs.
Turkey	Gold	Piaster	04·4	Gold: 25, 50, 100, 250, and 500 piasters.
Uruguay	Gold	Peso	103·4	Gold: peso. Silver: peso and divisions.
Venezuela	Both	Bolivar	19·3	Gold: 5, 10, 20, 50, and 100 bolivars. Silver: 5 bolivars.

*Haikwan (Customs).

RICHARD P. ROTHWELL.

prepared himself by a post-graduate course in chemistry and mining engineering, he went to Utah in 1880, and has since confined himself to the metallurgy of lead and copper at various Western smelting works as foreman, chemist and superintendent, having been connected with the Germania Lead Works at Salt Lake City, the Standard Smelting and Refining Co. at Durango, Colo., the Viola Mining and Smelting Co. at Nicholia, Idaho, the Buena Vista Mining and Smelting Co. at Buena Vista, Colo., and elsewhere. He has made a number of contributions to the *Engineering and Mining Journal* and the *Transactions of the American Institute of Mining Engineers*, all on subjects relating to lead and copper smelting, and has written for this volume the paper on "Present Practice in Silver-Lead Smelting at Denver and Pueblo."

BAIN, H. FOSTER, graduated from Moore's Hill College in Indiana in 1890. He studied geology and chemistry at Johns Hopkins from 1891 to 1893, and was a fellow of the University of Chicago in 1896 and 1897, receiving from the latter the degree of Ph.D. Since 1893 Dr. Bain has been Assistant State Geologist of Iowa. He has also served as lecturer on economic geology at the State University of Iowa and at the University of Chicago. He contributes to this volume the article on the manufacture of paving brick.

BLAKE, WILLIAM PHIPPS, was born in New York in 1826, and graduated at the Sheffield Scientific School of Yale University in 1852. In 1853 he was geologist of the Union Pacific Railway Co.'s exploring expedition to California, in connection with which he wrote several reports. In 1859–60 he was editor of the *Mining Magazine*, and from 1861 to 1863 was mining engineer for the Japanese government, and with his associate, Raphael Pumpelly, organized, under the auspices of that government, the first school of science in Japan. Returning to America, he accompanied a Russian scientific expedition in Alaska, and described a series of glaciers not before known. In 1863 he was engaged in explorations in California and Nevada, and became professor of geology in the College of California and geologist to the State Board of Agriculture. Since that time Prof. Blake has been engaged in mining in various parts of the United States. At present he holds the chair of mining in the University of Arizona at Tucson. He has been connected prominently with most of the world's expositions during the last three decades, having been either a special agent or commissioner at Paris in 1867, at Vienna in 1873, at Philadelphia in 1876, at Paris in 1878, and at Chicago in 1893; and in connection with these offices made numerous reports, and in 1878 received from the French government the cross of the Legion of Honor. Prof. Blake has been a voluminous contributor to mining and metallurgical literature and is the author of several treatises. To the present volume he contributes the notes on the occurrence of tungsten in Arizona.

CLAREMONT, LEOPOLD, born in London, was articled at an early age to Richard Cook Nockhold, precious-stone merchant, of London, and during his apprenticeship devoted himself to the study of the sciences which pertain to the business, so that by the end of his apprenticeship he had become an expert mineralogist as well as a practical gem cutter. For a while he acted as curator of a collection of rare gems exhibited at the Imperial Institute, and was afterward appointed manager of the gem-cutting establishment of Streeter & Co., Ltd., Glasshouse Street, London, which business he finally purchased and is now carrying on at the same address. Mr. Claremont has gained a high reputation as a gem expert through the application of scientific methods to this industry. He has been a contributor to numerous technical periodicals, and for this volume has written the paper on "The Identification of Gems."

CLEMMER, JOEL G., was educated at the Central High School of Philadelphia, and after studying medicine for a short time entered the employ of the Pennsylvania Salt Manufacturing Co., with which he remained fourteen years. During the last six years of his connection with this company he acted as assistant to the superintendent of the works, built its electrolytic copper plant, and was engaged in technical investigations concerning the various processes in use at Natrona. On closing his connection with this company Mr. Clemmer established himself at Philadelphia as consulting electro-metallurgist. To this volume he has contributed the paper on the "Manufacture of Aluminum Sulphate."

OME OF THE CONTRIBUTORS TO THE MINERAL INDUSTRY.

L. S. AUSTIN.

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W. P. BLAKE.

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G. P. GRIMSLEY.

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H. O. HOFMAN.

Cole, L. A., who contributes to this volume some notes on the white lead industry in the United States in 1898, is well known as president of the National Lead Co.

Dawson, George M., Director of the Geological Survey of Canada, was educated at McGill University, Montreal, and at the Royal School of Mines, London, graduating as an associate of that school in 1872. In 1873 and 1874, as naturalist and geologist to the British contingent of the international commission engaged in the delimitation of the boundary between Canada and the United States, he traversed and examined the Western country from the Lake of the Woods to the Rocky Mountains, publishing a detailed report on the geology and resources of the 49th parallel in 1875. In the same year he was appointed to the Geological Survey of Canada, with which he has since been identified. His field work in this connection has been chiefly in British Columbia and the Northwest Territories, upon which he has published numerous reports and papers, both officially and in scientific and technical periodicals. Several long explorations have been undertaken by Dr. Dawson, including the survey of the Queen Charlotte Islands in 1878, a traverse from the Pacific Coast to Manitoba in 1879, and the Yukon expedition of 1887, all resulting in the production of maps and reports embodying much new information, both geographical and geological. In 1891 he was appointed one of H. M. Bering Sea Commissioners, and in this connection visited the western coasts of Alaska and parts of the eastern Asiatic coast, with the intervening islands. In the following year he was a member of the joint commission appointed to deal with the fur-seal question, which held its sessions at Washington. In January, 1895, he was appointed Director of the Geological Survey, in succession to Dr. A. R. C. Selwyn. His original geological work has been largely connected with the discovery and development of the economic resources of the western part of the Dominion of Canada. He is an original Fellow and past President of the Royal Society of Canada, a Fellow of the Royal Society of England, and has been awarded the Bigsby medal of the Geological Society of London and the Patron's or Victoria medal of the Royal Geographical Society. He has written for this volume concerning the coal deposits of the Canadian Rocky Mountain region.

Douglass, James, is a Canadian, who has made his home in the United States since 1875. His first experience in mining and metallurgy was acquired in trying to unravel the complicated affairs of an unsuccessful Canadian mining enterprise. He came to the States in order to take charge of copper works established in Phœnixville, Pa., for the utilization of local copper ores, whose supply, however, proved deficient; but he is best known through his connection with the copper industry of Arizona, with which he has been intimately associated almost since its initiation. He is president of the American Institute of Mining Engineers; is also president of the Copper Queen Consolidated Mining Co. and of other Arizona concerns, and is also president of the Arizona and Southern Railroad Co. Such original work as he has done was chiefly in connection with the late eminent chemist, Dr. T. Sterry Hunt, in the field of the hydrometallurgy of copper. Mr. Douglass contributes to this volume the notes on the copper industry in Arizona.

Falding, F. J., was born in England and educated at Amersham Hall, London, and at the Bergakademie at Freiberg in Saxony, but did not graduate from the latter. After emigrating to America he spent a year and a half in the Bank of Montreal and about three years in the office of the Attorney-General of Canada. In 1878–79 he made a study of the Canadian apatite deposits, and in 1880 returned to Europe and studied the manufacture of sulphuric acid and fertilizers in England and Germany. In 1881 he returned to Canada and unwatered the Capelton pyrites mines, now worked by G. H. Nichols & Co. From 1882 to 1886 he practiced as a mining engineer, with headquarters in New York, making a specialty of pyrites and phosphate mining, and made several visits to Europe to study mines and chemical and metallurgical works there. In 1888 he entered the employ of the Grasselli Chemical Co. as engineer in charge of its mines, and in 1890 became chief engineer for that company with charge of all its works. During this time he designed the company's new works at East Chicago, Ill. In 1895 he established himself in New York as a consulting chemical engineer, since which time he has constructed and rebuilt many sulphuric acid plants in various parts of the United States. He contributes to this volume the paper on the manufacture of chamber sulphuric acid.

FERRY, CHARLES, was born in 1868 at Peterborough, N. H. He received his early education at Williston Seminary, Easthampton, Mass., and graduated at the Worcester Polytechnic Institute, Worcester, Mass., in 1888. For two years after leaving Worcester he was assistant to Dr. T. M. Drown, who was then director of the chemical department of the Massachusetts Institute of Technology in Boston. There he gathered much valuable professional experience and acquired an interest in scientific research, which finally became concentrated upon refractory materials. For seven years Mr. Ferry has been chemist of the Ostrander Fire Brick Co., the main office and laboratories of which are located at Troy, N. Y. In the interests of this company Mr. Ferry's practical and theoretical investigations have been along the lines of fire clays and their proper application to various classes of work. He has written for this volume the paper on the principles of fire brick manufacture.

FRASCH, H. A., was born and educated in Germany, coming in 1876 to the United States, where he first was engaged in the sugar business in Louisiana, remaining there until 1892, when he joined his brother, who was then erecting ammonia soda works at Bay City, Mich. In 1886 he accepted the technical management of a cement works at Kansas City and established a mineral paint factory. From 1888 to 1891 he was engaged in the development of the salt and soda ash works at Cleveland, Ohio, and then took charge of the department of technical research of the Grasselli Chemical Co. at Cleveland. In this capacity he discovered the presence of valuable organic acids in petroleum refuse and devised methods for their separation and utilization in the production of dye stuffs. While with the Grasselli Chemical Co. he also constructed a plant for the manufacture of artificial asphalt from petroleum refuse and an apparatus for the production of aqua ammonia direct from gas liquor. He left the Grasselli Co. in 1896 in order to construct an asphaltum refining works in Texas. Lately Mr. Frasch has been established as consulting chemical engineer in New York. He contributes to this volume the papers on asphaltum refining, recovery of ammonia from gas liquor, and lime and lime burning.

GILCHRIST, PETER S., was born in 1861 at Manchester, England, and received an engineering training with Wren & Hopkins, of Manchester. In 1881 he came to the United States to introduce the Spence furnace for burning pyrites fines for the Sulphur Mines Co., of Virginia. Upon his return to England he was appointed manager of a large chemical works in Yorkshire, where he remained several years. He then came to the United States again and constructed many acid and fertilizer plants, principally in the South, where he is now located as a chemical engineer. In 1893 he patented with an associate a device for reducing by the use of pipe columns in connection with chambers the amount of chamber space necessary to the manufacture of sulphuric acid, this system being now in use in many parts of the United States. He contributes to this volume the paper on the manufacture of acid phosphate of lime.

GORBY, S. S., was born in Ohio in 1847 and graduated from the State University of Missouri in 1867. He was first engaged in engineering work in the West, and gradually became interested in mining and made professional visits to Colorado, Idaho, Montana, Washington and British Columbia in the employ of various mining companies. Later he was connected with the development of phosphate mines in Florida, gold and mica mines in North Carolina and marble deposits in Tennessee, Alabama, Georgia, Missouri and Arkansas, together with the clay resources of these States. He was for three years Assistant State Geologist of Indiana, under Prof. Maurice Thompson, whom he succeeded. Since leaving that position Mr. Gorby has devoted his attention especially to the development of deposits of onyx marble in Kentucky and elsewhere.

GRESLEY, WILLIAM STUKELEY, was born in 1852 at Ashby de la Zouch, England, and was educated at Repton School and the University of Edinburgh. From 1863 to 1873 he was articled to John Thomas Woodhouse, civil and mining engineer of Derby and London. He is now engaged in practice as consulting mining engineer at Erie, Pa., being employed by the Scott interests in Pennsylvania, Illinois and elsewhere. In connection with the General Electric Co. he planned the pioneer central power station electric coal-mining plant in the United States, which was installed at Scott Haven, Pa., and has since been largely added to. Mr. Gresley is a Fellow of the Geological Society and a member of numerous technical societies. He is the author of *Glossary of Terms Used in Coal Mining* (1883); and *Different Methods of Working Coal in Various Countries* (1893), and has contributed numerous papers on coal mining to technical periodicals in England and America. He has written the paper on "Coal Mining Methods and their Economic Bearing" for the present volume.

GRIMSLEY, G. PERRY, graduated from the Ohio State University in 1890 and from Johns Hopkins with the degree of Ph.D. in 1894. He was then connected with the Ohio Geological Survey for two years, being engaged in work on coal, clay and building stones. In Maryland he made a study of the granite of the northeastern part of the State. The last three years he has been assistant geologist on the University Geological Survey of Kansas, employed especially in studying the gypsum deposits, and has also been professor of geology and natural history at Washburn College, at Topeka. He has written for this volume the paper on the "Technology of Gypsum."

GROHMANN, E., who contributes notes on the mineral industry of Greece to the present volume, is a resident of the island of Seriphos, Greece, where he is engaged in mining.

HECKSCHER, AUGUST, was born at Hamburg, Germany, and educated in Germany and Switzerland. He was actively connected with the anthracite coal mining industry in the United States from 1868 to 1882. Since 1881 he has been engaged in the zinc mining and manufacturing industry in New Jersey, Pennsylvania, Illinois, Wisconsin and Missouri. At present he is general manager of the New Jersey Zinc Co.

HOFMAN, H. O., was born in 1852 at Heidelberg, Germany. He studied at the Bergakademie at Clausthal, where he graduated in 1877 in mining engineering and metallurgy. He was then appointed chemist and assistant at the smelting and refining works at Lautenthal in the Harz. In 1881 he came to the United States and was employed successively at Mine La Motte in Missouri, at the Argentine smelting and refining works of the Consolidated Kansas City Smelting and Refining Co., and as metallurgist of the Delaware Lead Co. in Philadelphia. When the last named works were closed he went to Colorado, and after running the Rico smeltery for a short time went to Park City, Utah, to study the amalgamation and lixiviation of silver ores at the Ontario mill. After a short time spent in charge of a smeltery in Mexico he was appointed assistant to Prof. Richards at the Massachusetts Institute of Technology in Boston; from there he went to the School of Mines of South Dakota as professor of metallurgy and assaying, where he remained until called back to the Massachusetts Institute of Technology to the professorship of metallurgy, which he now holds. Dr. Hofman has made numerous contributions to technical literature, his most important work being the admirable treatise on *The Metallurgy of Lead*. For his paper on the "Dry Assay of Tin Ores" the degree of Ph.D. was conferred on him by the University of Ohio. For the present volume he has reviewed progress in metallurgy of lead, continuing the series of articles begun in the first volume.

HOFMANN, OTTOKAR, was born in 1843 at Ruskberg in Austria-Hungary. He was educated at the Realschule and Polytechnic Institute in Vienna, and the Bergakademie at Freiberg, graduating from the last in 1866. In 1867 he came to the United States, first entering the assay office of Guido Küstel in San Francisco as a partner. In 1868 he erected lixiviation works at La Dura, Sonora, Mexico, which was the first introduction of the silver leaching process in North America. Its success led to his building works at La Trinidad and San Marcial, Sonora, and Trinufo, Lower California. While at La Dura he substituted calcium polysulphide for sodium polysulphide as a precipitant. Returning to the United States Mr. Hofmann served as metallurgist in various amalgamation and chlorination mills in Nevada and California. In 1878 he introduced a successful leaching process of his own device for gold-silver ore at Monitor, Cal., and in 1880 erected the works of the Silver King Co. at Pinal, Ariz. Since that time Mr. Hofmann has been occupied almost exclusively in hydrometallurgy. He has been engaged successively at St. Louis, Mo. (1882–83), Cusihuiriachic, Chihuahua (1885), Yedras, Sinaloa (1886–87), Cusihuiriachic (1887), Parral, Chihuahua (1888). In 1890 he was appointed managing director of the North Mexican Mining and Milling Co., operating at Cusihuiriachic, which position he held until the spring of 1894. While at "Cusi" he substituted trough-lixiviation for tank-lixiviation. In 1894 he was engaged at Sombrerete and at Parral. In 1895 Mr. Hofmann went to Argentine, Kan., to take charge of the leaching department, in which the Hunt & Douglas process was employed. In 1897 he discovered a process by which he could free in one operation copper sulphate solutions from iron, arsenic, antimony and other impurities, producing a very fine grade of blue vitriol direct from any kind of copper matte. On account of this discovery the company changed the works arranged for the Hunt & Douglas process into blue vitriol works, and enlarged them. Mr. Hofmann is still in charge of these works. Mr. Hofmann has made many valuable contributions to metallurgical literature, chiefly on his own specialty, through the *Engineering and Mining Journal* and the *Transactions of the American Institute of Mining Engineers.*

HOOVER, HERBERT C., is a graduate of the department of mining in the Stanford University, California, and has been connected with the Geological Survey of Arkansas, and the U. S. Geological Survey; also with various mines in California, New Mexico, Colorado and elsewhere. In 1897 he went to Western Australia as engineer for Bewick, Moreing & Co., and became manager of the Sons of Gwalia, Ltd., and consulting engineer for various companies in that region.

HOPKINS, T. C., was born in 1861 in Pennsylvania, and graduated from De Pauw University, Greencastle, Ind., 1887, taking the degree of M.S. in 1890. He then spent one year at Stanford University and two years at the University of Chicago. He was assistant geologist of the Geological Survey of Kansas for three years, and on the Geological Survey of Indiana for part of two years. He is now teaching geology at the Pennsylvania State College. His papers on feldspar and kaolin in the present volume are based on his investigations in the field last summer. Prof. Hopkins has been a contributor to the State geological reports of Arkansas, Indiana and Pennsylvania, to previous volumes of THE MINERAL INDUSTRY, and to various technical journals.

HOWE, HENRY MARION, was for a number of years lecturer at the Massachusetts Institute of Technology in Boston, and is now professor of metallurgy at the Columbia School of Mines, and is one of the most eminent authorities on the metallurgy of iron and steel. His close and careful studies and his thorough and scientific experimental work have been generally recognized, and his monumental book on *The Metallurgy of Steel* is received everywhere as the highest authority on the subject. Mr. Howe has been president of the American Institute of Mining Engineers. In 1895 the British Iron and Steel Institution voted him the Bessemer medal, which is the highest honor it can bestow. In 1895, also, the Franklin Institute of Philadelphia awarded him the Elliott Cresson medal, and the Verein Zur Beförderung des Gewerbfleisses of Berlin its great gold medal, for distinguished scientific services. Mr. Howe has written for this volume a paper on the "Progress in Iron and Steel Metallurgy in 1898."

INGALLS, WALTER RENTON, a well-known mining engineer and metallurgist, was born at Lynn, Mass., in 1865, and graduated from the Massachusetts Institute of Technology. in 1886. In 1886–90 he was engaged in mining at Leadville and elsewhere in Colorado. In 1890–92 he was assistant editor of the *Engineering and Mining Journal*, resigning that position to go to Mexico to open tin mines in the State of Durango for the Pittsburg & Mexican Tin Mining Co. In 1893 and 1894 he visited professionally various mining districts in the United States, Canada, Belgium, Germany and Poland, devoting himself

SOME OF THE CONTRIBUTORS TO THE MINERAL INDUSTRY.


HENRY MARION HOWE.


T. C. HOPKINS.


EDWARD KELLER.


E. D. LEVAT.


F. H. LEWIS.


J. OBALSKI.


HENRY LOUIS

SOME OF THE CONTRIBUTORS TO THE MINERAL INDUSTRY.

ALBERT SAUVEUR.

ROBERT H. RICHARDS.

C. E. SIEBENTHAL.

R. B. SYMINGTON.

ROBERT M. THOMPSON.

CHARLES L. WHITTLE.

TITUS ULKE.

especially to the metallurgy of zinc. During a part of 1894 he had charge of the operations of the Illinois Phosphate Co. in Florida, and later in the year became connected with the Gold and Silver Extraction Co. of America, Ltd., as metallurgist. In 1895 he was manager of cyanide reduction works at Cripple Creek, Colo., and in 1896 of copper-matte smelting works in Durango, Mexico, returning to New York in 1897. For the present volume he has written on the progress in the metallurgy of gold and silver and zinc, besides numerous unsigned articles. He has been assistant editor of THE MINERAL INDUSTRY, Vols. V., VI. and VII.

KELLER, EDWARD, was born at Sauk City, Wis., in 1857, and studied chemistry and the allied sciences at Stuttgart in Germany and Zurich in Switzerland, taking the degree of Ph.D. at the latter university. Ill health compelled him to go to the Rocky Mountains, and while there he took up metallurgical chemistry, and subsequently studied at the Colorado School of Mines and at Freiberg in Saxony. He then spent four years at Butte, Mont., in connection with the smelting works of W. A. Clark and the Parrot Silver and Copper Co. Since 1893 he has been metallurgical representative of the Anaconda Copper Mining Co. at the Baltimore Copper Works, where half of the former company's output is refined electrolytically and its whole output is brought into marketable shape. Dr. Keller contributes to this volume the paper on "The Electrolysis and Refining of Copper," a subject in which he has gained much distinction on account of his scientific investigation of it.

KNAPP, S. A., was born in Tuolumne County, Cal., in 1854, but since 1875 has lived in Esmeralda County, Nev., where he has been engaged in mercantile business and in mining, either as owner, manager, or superintendent, his experience covering the operation of gold and silver mines and mills and the exploitation of natural soda deposits. He contributes to this volume the paper on the "Occurrence and Recovery of Sodium Carbonate in the Great Basin."

LEVAT, DAVID, born in 1855, at Montpellier, France, studied in the École Polytechnique of Paris, graduating as a mining engineer in 1880. After four years' experience in the copper mines of Montecatini in Tuscany and in Greece and Algeria, he was appointed in 1885 general manager of the "Société le Nickel" and remained till the end of 1889 at the head of that company's works. Since 1894 M. Levat has been specially engaged in placer gold mining in eastern Siberia and in French Guiana, and has introduced in both countries several improvements, excavators and dredges of his own design. In 1898 M. Levat discovered the phosphate deposits in the Pyrenees, France, which promise to be of great importance. As consulting engineer of various companies M. Levat has visited and reported upon mines in many parts of the world, especially New Caledonia, Australia, California, Algeria, Tunis, Spain, Siberia, and the Guianas.

LEWIS, FREDERICK H., studied civil engineering at the University of Pennsylvania, graduating in 1878. He then became heliotroper on the United States Coast Survey, serving during the summer of 1878, and for three years afterward was assistant engineer of the construction department of the Pennsylvania Railroad Co.'s lines west of Pittsburg. From 1882 to 1885 he was superintendent of bridges and buildings of the Northern Pacific Railway, being situated at St. Paul, Minn., and was also in charge of the location of the company's terminal lines between St. Paul and Minneapolis. In 1885 and 1886 he was resident engineer of the South Pennsylvania Railroad, at Sideling Hill tunnel, Fulton County, Pa. From 1886 to 1893 he was eastern manager of the Pittsburg Testing Laboratory at Philadelphia. Since 1893 he has been practicing as consulting engineer at Philadelphia, and has also been consulting engineer for the firm of Booth, Garrett & Blair, in their department of physical tests and inspection. He contributes to this volume the paper on "The Manufacture of Hydraulic Cement in the United States."

LOUIS, HENRY, was born in London in 1855. He first attended school in Bavaria, then in London, and in 1873 entered the Royal School of Mines, the associateship of which he obtained in 1876. He was then engaged for some time in Dr. Percy's private laboratory, making some of the researches for Dr. Percy's treatise on *Silver*. Thence he was sent by Sir William Siemens to Nova Scotia, for the Steel Company of Canada. Later he went to South America, where he was employed in gold mining in Colombia and Ecuador; from thence to the Gold Coast and next to South Africa, where he spent over three years in gold and diamond mining. He then spent three years managing and inspecting mines and exploring in the far East, including the Malay Peninsula (gold and tin), Siam, Tonkin and Bengal. Upon his return to Europe he devoted himself to consulting work and the management of an iron mine in Spain. In 1895 he was appointed professor of mining in the Durham College of Science, Newcastle-upon-Tyne, and in 1896 received from the University of Durham the honorary degree of M.A. Mr. Louis is the author of a *Handbook on Gold Milling* (1894), and a revised and enlarged edition of Phillips' *Treatise on Ore Deposits* (1896), and a translation of Schnabel's *Handbook of Metallurgy* (1898). He contributes to this volume the notes on the metallurgy of tin.

OBALSKI, J., was born in France in 1852, and studied at the École des Mines at Paris; after graduation he occupied several positions in connection with the mining industry in France and Spain, and visited various mining districts in those countries and Belgium and Portugal. In 1881 he was called by the government of the Province of Quebec to fill the position of mining engineer and inspector of mines, which post he still occupies. He has contributed to the present volume the notes on asbestos, chrome ore, and mica mining in Quebec.

RICHARDS, ROBERT HALLOWELL, professor of mining and metallurgy at the Massachusetts Institute of Technology, was born in 1844 at Gardiner, Me. He graduated in 1868 from the Massachusetts Institute, being a member of its first class, and became assistant in chemistry in the corps of instruction, passing successively to the posts of instructor, assistant professor of chemistry, professor of mineralogy and assaying, professor of mining engineering, and in 1884 to his present professorship of mining and metallurgy. Under his administration the mining and metallurgical laboratory, which was the first of its kind in an educational institution, has been developed to a high degree of excellence, and has been a model for similar laboratories in other colleges. In addition to his professional duties Prof. Richards has been actively engaged as a consulting engineer in mining and metallurgical work, and has been the inventor of several ingenious devices, which have found extended use in practice. He has contributed many valuable papers to the technical press and to the transactions of various scientific societies. His most recent work on the principles of ore dressing is the admiration of the entire body of engineers engaged in that branch of metallurgy. To the several volumes of THE MINERAL INDUSTRY he has contributed the comprehensive reviews of "Progress in Ore Dressing."

ROPES, LEVERETT S., was born at Ishpeming, Mich., in 1868, and studied at the Michigan Mining School at Houghton, Mich. He has had experience in mining and surveying in northern Michigan in connection with his father, Julius Ropes, and the United States Geological Survey and the Michigan Geological Survey. In 1893 he was assistant to Prof. Denton in mining and mine surveying school work, but resigned this place to accept a commission to develop a corundum property in North Carolina. Since that time he has been engaged in corundum mining, gold mining and engineering construction work in North Carolina. Mr. Ropes contributes to this volume notes on the present condition of the corundum industry of North Carolina and Georgia.

SAUVEUR, ALBERT, studied four years at the École des Mines at Liege, Belgium, and then at the Massachusetts Institute of Technology, where he was graduated in 1889. After one year in the chemical laboratory of the Pennsylvania Steel Co., at Steelton, Pa., he became connected with the South Chicago works of the Illinois Steel Co., with which he remained six years, most of the time in charge of the experimental laboratory. Since February, 1897, he has been proprietor and manager of the Boston Testing Laboratory at Boston, and is now lecturer at the Massachusetts Institute of Technology and editor of *The Metallographist*. Mr. Sauveur is a member of numerous scientific societies, and has been a contributor to numerous technical publications. He has a world-wide reputation as an expert in his specialty, the investigation of the microstructure of metals and its bearing upon their technology. Mr. Sauveur contributes to this volume a paper upon that subject.

SIEBENTHAL, CLAUDE E., was born at Vevay, Ind., in 1869, and studied at Indiana University and Stanford University, 1889–93. He became a fellow in geology at the University of Chicago in 1897, where he is now located. Mr. Siebenthal was assistant on the Geological Survey of Arkansas from 1889 to 1893, and has been connected with the Geological Survey of Indiana since 1896. In connection with Prof. T. C. Hopkins he prepared the treatise on the Bedford oölitic limestone for the twenty-first report of the State Geologist of Indiana, published in 1896. He has written on the same subject for the present volume of THE MINERAL INDUSTRY.

SJOSTEDT, ERNST A., was graduated in 1876 from the School of Mines at Stockholm, Sweden, whence he went first to Le Creusot, France, and thence to the works of the Bethlehem Steel Co. in Pennsylvania, where he occupied the position of assistant chemist. He next carried on experiments for the roasting of iron ores at Katahdin, Me., and then was for five years in charge of charcoal iron smelting furnaces at Shelby, Ala., and Cherokee, Ga. Returning to Maine he rebuilt the Katahdin furnaces, and remained as superintendent for five years. He then organized the Pictou Charcoal Co. in Nova Scotia, for which he acted as manager during the subsequent six years. In 1897 Mr. Sjostedt established himself in Montreal, making a specialty of the design of roasting furnaces and gas producers and the utilization of peat and mill refuse. Mr. Sjostedt has been a contributor to numerous technical publications, and for this volume has written the paper on the utilization of peat.

SYMINGTON, R. B., is a native of Scotland. After serving an apprenticeship as civil and mining engineer he studied under Macquorn Rankine in the University of Glasgow and at the Glasgow School of Mines. He has been engaged in his profession on the Pacific coast for the last 24 years. Mr. Symington contributes to this volume a paper on the metallurgy of quicksilver in California, a subject in which he has had long professional experience.

THOMPSON, ROBERT M., who contributes to this volume certain notes on the nickel industry of the United States in 1898, is well known as president of the Orford Copper Co., which is now the most important nickel smelter in the world. Under Mr. Thompson's direction the new process of separating nickel from copper matte by smelting with an alkaline flux was developed at the Orford works; this process has been of great importance in the economical production of nickel.

ULKE, TITUS, was born in 1866, at Washington, D. C. In 1889 he graduated from the Royal School of Mines at Freiberg, Saxony, as metallurgical engineer. After spending some time in visiting the various mines and metallurgical works in Europe, Mr. Ulke returned to this country and was engaged as chemist to the Harney Peak Tin Co., in South Dakota. In 1891 he became assayer for the United Smelting Co., and afterward was engaged by the Anaconda Mining Co. as chemist at its electrolytic copper refining works. In 1893 Mr. Ulke acted as metallurgist to the Mines and Mining Department of the Chicago Columbian Fair, and was later employed at the Guggenheim works at Perth Amboy, N. J. As triangulator for the U. S. Geological Survey in 1897, he had charge of a party to survey the Montana timber reserves, from which he returned to continue work in his specialty—the electrolytic production of copper, nickel, gold and silver. Soon after the declaration of war with Spain, Mr. Ulke was appointed resident Inspector of Ordnance, U. S. A., at the works of the Morgan Engineering Co., Alliance, Ohio, which position he still holds. Mr. Ulke has contributed the article on "Manufacture of Nickel-Steel in the United States" to the present volume.

WALKER, ARTHUR L., graduated from the Columbia School of Mines in 1883, going thence to Globe, Ariz., as assayer for the Old Dominion Copper Mining Co., of which company he was assistant superintendent in 1884–5. In 1886 he was engaged in New York in connection with the magnetic concentration of iron ores, in 1887 was mechanical engineer for the Silver King Mining Co., at Silver King, Ariz., and from 1888 to 1893 was superintendent of the Old Dominion Copper Mining Co. From 1893 to 1898 Mr. Walker was manager of the plant of the Baltimore Copper Smelting and Rolling Co., at Baltimore, Md., which position he resigned in February, 1898, to take charge of the copper department of the works of the Guggenheim Smelting Co., at Perth Amboy, N. J. To this volume Mr. Walker contributes notes on the new casting apparatus of his invention, which are incorporated in Dr. Keller's paper on the "Electrolysis and Refining of Copper."

WHITTLE, CHARLES L., studied at the Lawrence Scientific School of Harvard University, and upon completion of his course spent six years with the Archæan and Atlantic coast divisions of the U. S. Geological Survey, working mainly in Vermont and western Massachusetts. After the disruption of the Archæan division of the survey in 1892 he gave a course in assaying at Harvard University, and assisted in teaching mineralogy and petrography. Mr. Whittle has devoted much time to economic geology, and since leaving Harvard has been established professionally at Boston as a geologist and mining engineer. Mr. Whittle contributes to this volume papers on the clay and stone industries of Massachusetts, in both of which subjects he is well known as an expert.

INTRODUCTION.

THE total value of the mineral production of the United States in 1898 was $709,816,750, against $648,804,899 in 1897. Of the production in 1898, $314,255,620 was the value of metals against $272,178,392 in the previous year, and $433,659,141 ores and minerals, against $407,913,912 in 1897. The values given for ores and minerals include $38,098,011 in 1898 and $31,287,405 in 1897 for bauxite, manganese and iron ores, which were used for making aluminum, ferromanganese and pig iron. These duplications were deducted in the aggregate values stated above. Eighteen secondary products for which statistics were collected, derived from some of the ores included in these totals had an aggregate value of $49,432,829 in 1898 against $41,718,420 in 1897. There was also a production of copper, lead, silver and gold from foreign ores and bullion, valued at $58,948,125 in 1898, against $47,127,174 in 1897.

In the preparation of the statistics for this volume, the figures previously reported for 1897 have been revised in the light of later and more minute investigation, in accordance with our practice, wherefore it is important for students to observe the caution to use always the figures in the latest volume of THE MINERAL INDUSTRY. There are no statistical reports of this nature which are absolutely correct, owing to the practical impossibility of obtaining accurate reports from all the producers in some extensive and greatly subdivided industries, the absence of records on the part of many producers which prevents them from making returns, the unwillingness of a few to give their figures, and confusion as to the stage in which many products are to be reported. The last difficulty is especially likely to lead to errors in values, some producers estimating the worth of their product at the pit's mouth, and others reporting it in a more or less advanced state of completion, including thus not only the cost of carriage, but also the cost of manipulation. These difficulties appear not only in our own statistics, but also in the statistics reported by various governments. In our own work, however, we make a practice of going backward and correcting figures previously reported, whenever mistakes are discovered by subsequent investigation. In estimating values we are disposed, therefore, to use actual market prices rather than the values reported by the producers themselves, which are apt to be misleading for the reasons mentioned above.

We have made great use of the reports of several State geological surveys, especially the geological surveys of West Virginia, Alabama, Kansas, Iowa, and Montana, and the State mining bureaus of California and Colorado. We have generally credited these figures to the proper sources in the subsequent pages, but this acknowledgment may stand for any unintentional oversights.

PRODUCTION OF ORES AND MINERALS IN THE UNITED STATES. (FIRST PRODUCTS.)

Number.	Products.	Customary Measures.	1897.				1898.			
			Quantity.		Value at Place of Production. *a*		Quantity.		Value at Place of Production. *a*	
			Customary Measures.	Metric Tons.	Totals.	Per M. Ton.	Customary Measures.	Metric Tons.	Totals.	Per M. Ton.
1	Asbestos	Sh. T.	840	762	$12,950	$16·90	885	803	$13,425	$16·72
2	Asphaltum	Sh. T.	27,397	24,854	486,620	19·58	25,690	23,306	482,175	20·69
3	Asphaltic limestone	Sh. T.	2,390	2,168	11,450	5·28	14,099	12,791	70,495	5·51
4	Bitumin's sandstone.	Sh. T.	47,470	43,065	136,173	3·16	49,536	44,938	146,275	3·26
5	Barytes	Sh. T.	26,430	23,977	105,720	4·41	28,247	25,626	112,988	4·41
6	Bauxite	L. T.	20,590	20,919	51,475	2·46	26,791	27,220	66,978	2·46
7	Borate of calcium. *c.*	Sh. T.	19,400	17,600	388,000	*n*22·05	15,335	13,911	306,700	*n*22·05
8	Bromine	Lb.	487,149	221	136,402	617·19	486,978	221	136,354	617·19
9	Cement, nat. hydraul.	*g*Bbls	7,890,573	1,073,742	3,976,050	3·70	8,161,078	1,110,552	3,819,995	3·44
10	Cement, Portland	*h*Bbls	2,430,903	441,060	3,724,905	8·45	3,584,586	650,383	6,168,106	9·48
11	Chrome ore	L. T.	150	152	1,550	10·17	100	102	1,000	9·84
12	Clay products	......	......	......	56,487,527	......	......	......	58,470,543	......
13	Coal, anthracite	Sh. T.	52,645,133	47,759,351	85,857,717	1·80	52,848,605	47,943,940	81,445,937	1·70
14	Coal, bituminous	Sh. T.	148,155,567	134,405,849	120,013,372	0·89	165,208,025	149,875,737	128,419,354	0·86
15	Coal, cannel	Sh. T.	56,511	51,266	153,145	2·99	49,889	45,259	134,700	2·98
16	Cobalt oxide	Lb.	19,300	*k*8,754	30,880	*k*3·53	9,640	*k*4,373	15,424	*k*3·53
17	Copper sulphate. *f.*	Lb.	16,731,247	7,589	645,826	*m*85·10	28,061,501	12,729	956,897	*m*75·18
18	Corundum	Sh. T.	293	266	19,810	74·47	786	713	63,630	89·24
19	Emery	Sh. T.	1,900	1,724	92,000	53·36	2,956	2,682	143,800	53·62
20	Feldspar	L. T.	21,901	22,251	111,392	5·01	21,350	21,692	107,147	4·94
21	Fluorspar	Sh. T.	4,739	4,299	36,264	8·44	12,145	11,018	86,985	7·89
22	Fullers earth	Sh. T.	17,195	15,599	92,398	5·92	15,553	14,110	87,365	6·19
23	Garnet	Sh. T.	2,261	2,051	66,353	32·35	2,882	2,615	82,930	31·71
24	Grahamite	Sh. T.	1,756	1,593	52,680	*n*33·07	2,675	2,427	80,250	*n*33·07
25	Graphite, crystalline.	Lb.	993,138	*k*450,484	*m* 44,691	*k*0·10	1,647,679	*k*747,382	*m* 148,291	*k*0·20
26	Graphite, amorphous	Sh. T.	1,200	1,089	11,400	10·47	1,200	1,089	11,400	10·47
27	Gypsum	Sh. T.	300,369	272,493	889,177	3·26	348,686	316,326	989,589	3·13
28	Iron ore	L. T.	18,235,406	18,527,772	30,941,379	1·67	20,655,865	20,986,359	37,593,674	1·79
29	Magnesite	Sh. T.	*c*1,907	1,730	*c*7,628	4·41	*c*2,355	2,136	*c*9,420	4·41
30	Manganese ore. *i.*	L. T.	158,600	161,147	328,176	2·04	217,782	221,279	481,127	2·17
31	Mica, scrap	Sh. T.	2,882	2,615	*o*28,820	11·02	3,529	3,201	*o*39,837	12·44
32	Mica, sheet	Lb.	118,852	*k*53,911	83,298	*k*1·55	109,968	*k*49,881	91,432	*k*1·83
33	Molybdenite	Sh. T.	*Nil.*	*Nil.*	*Nil.*	*Nil.*	8	7	400	57·14
34	Monazite	Lb.	40,000	18	2,000	110·23	*d*150,000	68	*e*7,500	110·23
35	Natural gas	......	......	......	*e*10,000,000	......	......	......	*e*10,500,000	...
36	Ocher. *p.*	Sh. T.	42,590	38,637	468,490	*n*12·13	41,950	38,057	461,450	*n*12·13
37	Petroleum, crude	*y*Bbls	57,124,783	7,992,046	39,372,111	4·93	51,774,465	7,243,509	42,100,522	5·81
38	Phosphate rock	L. T.	1,007,367	1,023,485	3,022,101	2·95	1,257,645	1,277,717	4,355,025	3·41
39	Precious stones	......	......	......	*d*101,000	......	......	......	*e*125,000	......
40	Pyrites	L. T.	133,368	135,502	404,699	2·99	191,160	194,219	589,329	3·03
41	Salt. *q*	Bbls.	15,822,923	2,009,625	3,898,556	1·94	18,756,394	2,382,197	4,753,664	2·00
42	Silica, brick	M.	20,233	......	323,561	......	33,449	......	590,254	......
43	Diatom. earth	Sh. T.	3,000	2,722	30,400	11·17	1,392	1,263	11,002	8·71
44	Flint	Sh. T.	28,407	25,771	55,817	2·17	35,593	32,290	73,313	2·27
45	Sand, etc	L. T.	*e*750,000	762,000	*e*1,125,000	1·48	*e*800,000	812,800	*e*1,178,560	1·45
46	Pumice	Sh. T.	1,700	1,542	8,500	*n*5·51	144	131	720	*n*5·51
47	Grindstones	Sh. T.	36,502	33,114	342,186	10·33	38,859	35,253	438,675	12·44
48	Whetstones	......	......	......	97,229	......	......	......	107,990	......
49	Tripoli	Sh. T.	1,631	1,480	5,475	3·70	1,865	1,692	3,367	1·99
50	Slate, roofing	Sq'es.	932,124	281,872	2,829,526	*j*3·04	1,136,632	343,715	2,958,496	*j*2·60
51	Manufactures	......	......	......	595,105	......	......	......	528,856	......
52	Pigment	Sh. T.	6,857	6,221	57,863	9·30	7,886	7,154	70,671	9·88
53	Soapstone	Sh. T.	16,904	15,335	169,040	*n*11·02	18,862	17,111	158,635	9·27
54	Soda, natural. *r.*	Sh. T.	6,350	5,761	95,400	16·56	7,150	6,486	94,650	14·59
55	Stone, for building	......	......	......	*e*30,000,000	......	......	......	*e*32,500,000	......
56	Stone, limestone (flux)	L. T.	4,247,688	4,315,651	1,868,983	0·43	5,275,819	5,360,232	2,304,900	0·43
57	Stone, lithographic	Sh. T.	*Nil.*	*Nil.*	*Nil.*	*Nil.*	112	102	*n*2,240	*n*21·96
58	Strontium sulphate	Sh. T.	40	36	200	*n*5·55	2	2	10	*n*5·55
59	Sulphur	L. T.	1,690	1,717	34,814	*m*20·28	2,726	2,770	59,754	*m*21·57
60	Sulphuric acid. *f. l.*	Sh. T.	42,352	38,421	699,655	*m*18·21	47,558	43,144	970,183	*m*22·49
61	Talc, common	Sh. T.	10,164	9,221	90,908	9·86	9,112	8,266	78,645	9·51
62	Talc, fibrous	Sh. T.	52,836	47,933	283,685	5·92	54,807	49,721	285,759	5·75
63	Tungsten ore	L. T.	(*b*)	(*b*)	(*b*)	(*b*)	78·865	80·13	17,398	217·12
64	Uranium ore	Sh. T.	17	15	9,010	600·67	33	30	16,500	550·00
65	Wulfenite	Sh. T.	*Nil.*	*Nil.*	*Nil.*	*Nil.*	12	11	144	13·09
66	Zinc sulphate	Sh. T.	*Nil.*	*Nil.*	*Nil.*	*Nil.*	145	132	4,640	35·15
67	Zinc ore, exported	Sh. T.	9,251	8,393	211,350	25·18	11,782	10,688	299,870	28·06
68	Zinc white. *s.*	Sh. T.	26,262	23,825	1,686,020	70·77	32,747	29,708	2,226,796	74·96
69	Est. prod. unspecified		......	......	5,000,000	......	......	......	5,000,000	......
	Totals		......	......	407,913,912	......	......	......	433,659,141	......

PRODUCTION OF METALS IN THE UNITED STATES.

Number.	Products.	Customary Measures.	1897.				1898.			
			Quantity.		Value at Place of Production. *a*		Quantity.		Value at Place of Production. *a*	
			Custom'ry Measures.	Metric Tons.	Totals.	Per M. Ton.	Custom'y Measures.	Metric Tons.	Totals.	Per M. Ton.
					$	$			$	$
70	Aluminum........	Lb....	4,000,000	*k*1,814,388	*m*1,400,000	*k*0·77	5,200,000	*k*2,358,705	*m*1,690,000	*k*0·72
71	Antimony.........	Lb....	1,500,000	680	107,250	157·72	2,000,000	907	165,000	181·92
72	Copper...........	Lb....	501,370,295	227,420	*t*55,351,281	243·40	535,900,232	243,083	*t*63,129,047	259·70
73	Ferromang'nese. *u*	L. T..	173,695	176,474	*v*7,989,970	45·28	213,769	217,189	*v*10,474,681	48·23
74	Ferromolybden'm	Lb....	*Nil.*	*Nil.*	*Nil.*	....	2,100	....	1,050	1,102·30
75	Gold.............	*w*Oz..	2,864,576	*k* 89,092·4	59,210,786	*k*664·60	3,148,642	*k*97,933	65,082,430	*k*664·60
76	Iron, pig..........	L. T..	9,478,985	9,630,649	*v*91,009,633	9·45	11,560,165	11,745,128	*v*110,168,372	9·38
77	Iridium..........	*w*Oz..	20·25	....	606	....	8·5	....	255	....
78	Lead..............	Sh. T.	197,718	179,369	*m*14,156,609	78·93	228,475	207,271	*m*17,272,710	83·33
79	Molybdenum.....	Lb....	*Nil.*	*Nil.*	*Nil.*	....	9,550	*k*4,332	11,937	*k*2·76
80	Nickel............	Lb....	33,700	*k*15,286·2	*m*11,667	*k*0·76	11,145	*k*5,055·3	*m*3,845	*k*0·76
81	Platinum..........	*w*Oz..	200	*k* 6·2	2,468	396·78	300	*k* 9·3	3,837	*k*411·25
82	Quicksilver *x*......	Fl'sks	26,079	905	910,418	1,005·98	30,493	1,058	1,109,945	1,049·09
83	Silver.............	Oz....	56,457,292	*k*1,756,004·0	33,755,815	*mk*19·22	58,763,127	*k*1,827,723	34,670,245	*mk*18.97
84	Tungsten	Lb....	*Nil.*	*Nil.*	*Nil.*	....	33,200	*k*15,059	43,160	*k*2·87
85	Zinc..............	Sh. T.	100,387	91,070	8,271,889	*m*90·83	114,104	103,514	10,429,106	*m*100·75
	Totals	......	....	....	272,178,392	....	....	....	314,255,620	....

SECONDARY MINERAL AND CHEMICAL PRODUCTS OF THE UNITED STATES.

Number.	Products.	Customary Measures.	1897.				1898.			
			Quantity.		Value at Place of Production. *a*		Quantity.		Value at Place of Production. *a*	
			Customary Measures.	Metric Tons.	Totals.	Per M. Ton.	Custom'ry Measures.	Metric Tons.	Totals.	Per M. Ton.
					$	$			$	$
86	Alum.................	Sh. T.	15,456	14,022	463,680	*m*33·06	18,791	17,047	563,730	*m*33·07
87	Aluminum sulphate.	Sh. T.	46,355	42,053	1,158,875	*m*27·56	56,663	51,404	1,416,675	*m*27·56
88	Ammonium sulphate	Sh. T.	3,111	2,822	136,884	*m*48·51	3,589	3,256	179,450	*m*55·11
89	Calcium carbide.....	Sh. T.	1,925	1,746	134,750	77·18	(*b*)	(*b*)	(*b*)	(*b*)
90	Carborundum........	Lb...	1,242,929	564	153,812	272·72	1,594,152	723	151,444	209·47
91	Cement, slag........	*h*Bbls	40,000	7,258	60,000	8·27	157,662	28,606	235,721	8·24
92	Coke	Sh. T.	12,768,891	11,583,862	23,203,920	2·00	15,897,797	14,422,387	30,505,563	2·12
93	Copperas............	Sh. T.	15,702	14,245	78,660	5·52	11,285	10,238	58,105	5·68
94	Copper sulphate.....	Lb...	29,397,575	13,335	1,134,746	*m*85·10	27,057,860	12,273	922,673	*m*75·18
95	Crushed steel........	Sh. T.	324	294	51,824	176·27	330	299	46,200	154·52
96	Graphite, artificial..	Lb...	162,382	*k*73,655	10,149	*k*0·14	185,647	*k*84,209	11,603	*k*0·14
97	Lead, white. *z*.......	Sh. T.	105,804	95,988	9,522,360	99·20	93,172	84,525	9,391,738	111·11
98	Lead, red............	Sh. T.	7,798	7,074	744,709	105·27	9,160	8,310	916,000	139·59
99	Lead, orange miner'	Sh. T.	477	433	76,320	176·26	541	491	108,200	220·37
100	Litharge..............	Sh. T.	8,591	7,794	773,190	99·20	7,460	6,768	710,192	104·93
101	Mineral wool.........	Sh. T.	5,617	5,096	61,494	12·07	6,560	5,951	70,314	11·82
102	Soda, manufact'ed. *r*	M. T..	....	277,072	3,912,257	14·12	....	340,622	4,080,651	11·98
103	Venetian red.........	Sh. T.	2,196	1,996	40,790	20·48	3,270	2,967	64,570	21·76
	Totals.............	......	....	....	41,718,420	....	....	....	49,432,829	....

METALS PRODUCED FROM FOREIGN ORES AND BULLION. (*aa*)

Metals.	Customary Measure.	1897.			1898.		
		Quantities.		Values.	Quantities.		Values.
		Customany Measure.	Kgm.		Customary Measure.	Kgm.	
Copper	Pounds....	26,938,254	12,219,112	$2,973,983	36,055,352	16,354,600	$4,247,320
Gold..........	Troy ozs..	584,983	18,195	12,091,599	1,065,552	33,142	22,024,960
Lead..........	Short tons	92,117	83,568,000	6,595,577	89,209	80,929,874	6,744,200
Nickel........	Pounds....	4,099,390	1,859,000	1,419,209	7,127,784	3,233,142	2,459,085
Silver.........	Troy ozs..	40,218,776	1,250,934	24,046,806	39,784,000	1,237,560	23,472,560
Total values	....	....	....	$47,127,174	....	....	$58,948,125

In using the statistics in the foregoing tables reference should also be made to the detailed tables under the respective captions further on in this volume, where many explanatory notes as to the statistics will be found. The following notes refer to the four preceding tables: (*a*) Except where otherwise specified. (*b*) Not enumerated. (*c*) Amount or value of crude mineral. (*d*) Partly estimated. (*e*) Estimated. (*f*) Amount recovered as a by-product. (*g*) Barrels of 300 lb. (*h*) Barrels of 400 lb. (*i*) Includes manganiferous iron ore;

We have made great use of the reports of several State geological surveys, especially the geological surveys of West Virginia, Alabama, Kansas, Iowa, and Montana, and the State mining bureaus of California and Colorado. We have generally credited these figures to the proper sources in the subsequent pages, but this acknowledgment may stand for any unintentional oversights.

PRODUCTION OF ORES AND MINERALS IN THE UNITED STATES. (FIRST PRODUCTS.)

Number.	Products.	Customary Measures.	1897.				1898.			
			Quantity.		Value at Place of Production. *a*		Quantity.		Value at Place of Production. *a*	
			Customary Measures.	Metric Tons.	Totals.	Per M. Ton.	Customary Measures.	Metric Tons.	Totals.	Per M. Ton.
1	Asbestos.............	Sh. T.	840	762	$12,950	$16·99	885	803	$13,425	$16·72
2	Asphaltum	Sh. T.	27,397	24,854	486,620	19·58	25,690	23,306	482,175	20·69
3	Asphaltic limestone..	Sh. T.	2,390	2,168	11,450	5·28	14,099	12,791	70,495	5·51
4	Bitumin's sandstone.	Sh. T.	47,470	43,065	136,173	3·16	49,536	44,938	146,275	3·26
5	Barytes..............	Sh. T.	26,430	23,977	105,720	4·41	28,247	25,626	112,988	4·41
6	Bauxite..............	L. T..	20,590	20,919	51,475	2·46	26,791	27,220	66,978	2·46
7	Borate of calcium. *c*.	Sh. T.	19,400	17,600	388,000	*n*22·05	15,335	13,911	306,700	*n*22·05
8	Bromine.............	Lb...	487,149	221	136,402	617·19	486,978	221	136,354	617·19
9	Cement, nat. hydraul.	*g*Bbls	7,890,573	1,073,742	3,976,050	3·70	8,161,078	1,110,552	3,819,995	3·44
10	Cement, Portland ...	*h*Bbls	2,430,903	441,060	3,724,905	8·45	3,584,586	650,383	6,168,106	9·48
11	Chrome ore..........	L. T..	150	152	1,550	10·17	100	102	1,000	9·84
12	Clay products.......	......	....	....	56,487,527	....	....	....	58,470,543	....
13	Coal, anthracite......	Sh. T.	52,645,133	47,759,351	85,857,717	1·80	52,848,605	47,943,940	81,445,937	1·70
14	Coal, bituminous	Sh. T.	148,155,567	134,405,849	120,013,372	0·89	165,208,025	149,875,737	128,419,354	0·86
15	Coal, cannel.........	Sh. T.	56,511	51,266	153,145	2·99	49,889	45,259	134,700	2·98
16	Cobalt oxide.........	Lb...	19,300	*k* 8,754	30,880	*k* 3·53	9,640	*k* 4,373	15,424	*k* 3·53
17	Copper sulphate. *f*..	Lb...	16,731,247	7,589	645,826	*m*85·10	28,061,501	12,729	956,897	*m*75·18
18	Corundum...........	Sh. T.	293	266	19,810	74·47	786	713	63,630	89·24
19	Emery...............	Sh. T.	1,900	1,724	92,000	53·36	2,956	2,682	143,800	53·62
20	Feldspar.............	L. T..	21,901	22,251	111,392	5·01	21,350	21,692	107,147	4·94
21	Fluorspar............	Sh. T.	4,739	4,299	36,264	8·44	12,145	11,018	86,985	7·89
22	Fullers earth.........	Sh. T.	17,195	15,599	92,398	5·92	15,553	14,110	87,365	6·19
23	Garnet...............	Sh. T.	2,261	2,051	66,353	32·35	2,882	2,615	82,930	31·71
24	Grahamite...........	Sh. T.	1,756	1,593	52,680	*n*33·07	2,675	2,427	80,250	*n*33·07
25	Graphite, crystalline.	Lb...	993,138	*k* 450,484	*m* 44,691	*k* 0·10	1,647,679	*k* 747,382	*m* 148,291	*k* 0·20
26	Graphite, amorphous	Sh. T.	1,200	1,089	11,400	10·47	1,200	1,089	11,400	10·47
27	Gypsum..............	Sh. T.	300,369	272,493	889,177	3·26	348,686	316,326	989,589	3·13
28	Iron ore..............	L. T..	18,235,406	18,527,772	30,941,379	1·67	20,655,865	20,986,359	37,593,674	1·79
29	Magnesite...........	Sh. T.	*c*1,907	1,730	*c* 7,628	4·41	*c* 2,355	2,136	*c* 9,420	4·41
30	Manganese ore. *i*....	L. T..	158,600	161,147	328,176	2·04	217,782	221,279	481,127	2·17
31	Mica, scrap..........	Sh. T.	2,882	2,615	*o* 28,820	11·02	3,529	3,201	*o* 39,837	12·44
32	Mica, sheet..........	Lb...	118,852	*k* 53,911	83,298	*k* 1·55	109,968	*k* 49,881	91,432	*k* 1·83
33	Molybdenite..........	Sh. T.	*Nil.*	*Nil.*	*Nil.*	*Nil.*	8	7	400	57·14
34	Monazite.............	Lb...	40,000	18	2,000	110·23	*d* 150,000	68	*e* 7,500	110·23
35	Natural gas..........	......	....	....	*e*10,000,000	....	....	....	*e*10,500,000	...
36	Ocher. *p*.............	Sh. T.	42,590	38,637	468,490	*n*12·13	41,950	38,057	461,450	*n*12·13
37	Petroleum, crude....	*y*Bbls	57,124,783	7,992,046	39,372,111	4·93	51,774,465	7,243,509	42,100,522	5·81
38	Phosphate rock......	L. T..	1,007,367	1,023,485	3,022,101	2·95	1,257,645	1,277,717	4,355,025	3·41
39	Precious stones......	......	....	....	*d* 101,000	....	....	....	*e*125,000	....
40	Pyrites	L. T..	133,368	135,502	404,699	2·99	191,160	194,219	589,329	3·03
41	Salt. *q*	Bbls .	15,822,923	2,009,625	3,898,556	1·94	18,756,394	2,382,197	4,753,664	2·00
42	Silica, brick..........	M.....	20,233	....	323,561	....	33,449	....	590,254	....
43	Diatom. earth......	Sh. T.	3,000	2,722	30,400	11·17	1,392	1,263	11,002	8·71
44	Flint......	Sh. T.	28,407	25,771	55,817	2·17	35,593	32,290	73,313	2·27
45	Sand, etc....	L. T..	*e* 750,000	762,000	*e* 1,125,000	1·48	*e* 800,000	812,800	*e*1,178,560	1·45
46	Pumice.............	Sh. T.	1,700	1,542	8,500	*n* 5·51	144	131	720	*n* 5·51
47	Grindstones........	Sh. T.	36,502	33,114	342,186	10·33	38,859	35,253	438,675	12·44
48	Whetstones........	......	....	....	97,229	....	....	....	107,990	....
49	Tripoli..............	Sh. T.	1,631	1,480	5,475	3·70	1,865	1,692	3,367	1·99
50	Slate, roofing.........	Sq'es.	932,124	281,872	2,829,526	*j* 3·04	1,136,632	343,715	2,958,496	*j* 2·60
51	Manufactures......	......	....	....	595,105	....	....	....	528,856	....
52	Pigment............	Sh. T.	6,857	6,221	57,863	9·30	7,886	7,154	70,671	9·88
53	Soapstone............	Sh. T.	16,904	15,335	169,040	*n*11·02	18,862	17,111	158,635	9·27
54	Soda, natural. *r*......	Sh. T.	6,350	5,761	95,400	16·56	7,150	6,486	94,650	14·59
55	Stone, for building..	......	....	....	*e*30,000,000	....	....	....	*e*32,500,000	....
56	Stone, limestone (flux)	L. T..	4,247,688	4,315,651	1,868,983	0·43	5,275,819	5,360,232	2,304,900	0·43
57	Stone, lithographic..	Sh. T.	*Nil.*	*Nil.*	*Nil.*	*Nil.*	112	102	*n* 2,240	*n* 21·96
58	Strontium sulphate..	Sh. T.	40	36	200	*n* 5·55	2	2	10	*n* 5·55
59	Sulphur..............	L. T..	1,690	1,717	34,814	*m*20·28	2,726	2,770	59,754	*m*21·57
60	Sulphuric acid. *f. l.*..	Sh. T.	42,352	38,421	699,655	*m*18·21	47,558	43,144	970,183	*m*22·49
61	Talc, common........	Sh. T.	10,164	9,221	90,908	9·86	9,112	8,266	78,645	9·51
62	Talc, fibrous.........	Sh. T.	52,836	47,933	283,685	5·92	54,807	49,721	285,759	5·75
63	Tungsten ore.........	L. T..	(*b*)	(*b*)	(*b*)	(*b*)	78·865	80·13	17,398	217·12
64	Uranium ore.........	Sh. T.	17	15	9,010	600·67	33	30	16,500	550·00
65	Wulfenite............	Sh. T.	*Nil.*	*Nil.*	*Nil.*	*Nil.*	12	11	144	13·09
66	Zinc sulphate........	Sh. T.	*Nil.*	*Nil.*	*Nil.*	*Nil.*	145	132	4,640	35·15
67	Zinc ore, exported...	Sh. T.	9,251	8,393	211,350	25·18	11,782	10,688	299,870	28·06
68	Zinc white. *s*.........	Sh. T.	26,262	23,825	1,686,020	70·77	32,747	29,708	2,226,796	74·96
69	Est. prod. unspecified		....	....	5,000,000	....	....	....	5,000,000	....
	Totals..............		....	....	407,913,912	....	....	....	433,659,141	....

PRODUCTION OF METALS IN THE UNITED STATES.

Number.	Products.	Customary Measures.	1897.				1898.			
			Quantity.		Value at Place of Production. *a*		Quantity.		Value at Place of Production. *a*	
			Custom'ry Measures.	Metric Tons.	Totals.	Per M. Ton.	Custom'y Measures.	Metric Tons.	Totals.	Per M. Ton.
					$	$			$	$
70	Aluminum........	Lb....	4,000,000	*k*1,814,388	*m*1,400,000	*k*0·77	5,200,000	*k*2,358,705	*m*1,690,000	*k*0·72
71	Antimony.........	Lb....	1,500,000	680	107,250	157·72	2,000,000	907	165,000	181·92
72	Copper....	Lb....	501,370,295	227,420	*t*55,351,281	243·40	535,900,232	243,083	*t*63,129,047	259·70
73	Ferromang'nese. *u*	L. T..	173,695	176,474	*v*7,989,970	45·28	213,769	217,189	*v*10,474,681	48·23
74	Ferromolybden'm	Lb....	*Nil.*	*Nil.*	*Nil.*	....	2,100	....	1,050	1,102·30
75	Gold......	*w*Oz.,	2,864,576	*k* 89,092·4	59,210,786	*k*664·60	3,148,642	*k*97,933	65,082,430	*k*664·60
76	Iron, pig..........	L. T..	9,478,985	9,630,649	*v*91,009,633	9·45	11,560,165	11,745,128	*v*110,168,372	9·38
77	Iridium...........	*w*Oz..	20·25	....	606	....	8·5	....	255	....
78	Lead.....	Sh. T.	197,718	179,369	*m*14,156,609	78·93	228,475	207,271	*m*17,272,710	83·33
79	Molybdenum.....	Lb....	*Nil.*	*Nil.*	*Nil.*	....	9,550	*k*4,332	11,937	*k*2·76
80	Nickel............	Lb....	33,700	*k*15,286·2	*m*11,667	*k*0·76	11,145	*k*5,055·3	*m*3,845	*k*0·76
81	Platinum.........	*w*Oz..	200	*k* 6·2	2,468	396·78	300	*k*9·3	3,837	*k*411·25
82	Quicksilver *x*......	Fl'sks	26,079	905	910,418	1,005·98	30,493	1,058	1,109,945	1,049·09
83	Silver.............	Oz....	56,457,292	*k*1,756,004·0	33,755,815	*mk*19·22	58,763,127	*k*1,827,723	34,670,245	*mk*18.97
84	Tungsten	Lb....	*Nil.*	*Nil.*	*Nil.*	....	33,200	*k*15,059	43,160	*k*2·87
85	Zinc..............	Sh. T.	100,387	91,070	8,271,889	*m*90·83	114,104	103,514	10,429,106	*m*100·75
	Totals	......	....	....	272,178,392	....	....	....	314,255,620	....

SECONDARY MINERAL AND CHEMICAL PRODUCTS OF THE UNITED STATES.

Number.	Products.	Customary Measures.	1897.				1898.			
			Quantity.		Value at Place of Production. *a*		Quantity.		Value at Place of Production. *a*	
			Customary Measures.	Metric Tons.	Totals.	Per M. Ton.	Custom'ry Measures.	Metric Tons.	Totals.	Per M. Ton.
					$	$			$	$
86	Alum.................	Sh. T.	15,456	14,022	463,680	*m*33·06	18,791	17,047	563,730	*m*33·07
87	Aluminum sulphate.	Sh. T.	46,355	42,053	1,158,875	*m*27·56	56,663	51,404	1,416,675	*m*27·56
88	Ammonium sulphate	Sh. T.	3,111	2,822	136,884	*m*48·51	3,589	3,256	179,450	*m*55·11
89	Calcium carbide.....	Sh. T.	1,925	1,746	134,750	77·18	(*b*)	(*b*)	(*b*)	(*b*)
90	Carborundum........	Lb...	1,242,929	564	153,812	272·72	1,594,152	723	151,444	209·47
91	Cement, slag.........	*h*Bbls	40,000	7,258	60,000	8·27	157,662	28,606	235,721	8·24
92	Coke	Sh. T.	12,768,891	11,583,862	23,203,920	2·00	15,897,797	14,422,387	30,505,563	2·12
93	Copperas............	Sh. T.	15,702	14,245	78,660	5·52	11,285	10,238	58,105	5·68
94	Copper sulphate.....	Lb...	29,397,575	13,335	1,134,746	*m*85·10	27,057,860	12,273	922,673	*m*75·18
95	Crushed steel........	Sh. T.	324	294	51,824	176·27	330	299	46,200	154·52
96	Graphite, artificial..	Lb...	162,382	*k*73,655	10,149	*k*0·14	185,647	*k*84,209	11,603	*k*0·14
97	Lead, white. *z*.......	Sh. T.	105,804	95,988	9,522,360	99·20	93,172	84,525	9,391,738	111·11
98	Lead, red...........	Sh. T.	7,798	7,074	744,709	105·27	9,160	8,310	916,000	139·59
99	Lead, orange miner'	Sh. T.	477	433	76,320	176·26	541	491	108,200	220·37
100	Litharge............	Sh. T.	8,591	7,794	773,190	99·20	7,460	6,768	710,192	104·93
101	Mineral wool........	Sh. T.	5,617	5,096	61,494	12·07	6,560	5,951	70,314	11·82
102	Soda, manufact'ed. *r*	M. T..	....	277,072	3,912,257	14·12	....	340,622	4,080,651	11·98
103	Venetian red.........	Sh. T.	2,196	1,996	40,790	20·48	3,270	2,967	64,570	21·76
	Totals...........	......	....	....	41,718,420	....	....	....	49,432,829	....

METALS PRODUCED FROM FOREIGN ORES AND BULLION. (*aa*)

Metals.	Customary Measure.	1897.			1898.		
		Quantities.		Values.	Quantities.		Values.
		Customany Measure.	Kgm.		Customary Measure.	Kgm.	
Copper	Pounds....	26,938,254	12,219,112	$2,973,983	36,055,352	16,354,600	$4,247,320
Gold.........	Troy ozs..	584,983	18,195	12,091,599	1,065,552	33,142	22,024,960
Lead.........	Short tons	92,117	83,568,000	6,595,577	89,209	80,929,874	6,744,200
Nickel........	Pounds....	4,099,390	1,859,000	1,419,209	7,127,784	3,233,142	2,459,085
Silver........	Troy ozs..	40,218,776	1,250,934	24,046,806	39,784,000	1,237,560	23,472,560
Total values	....	....	....	$47,127,174	....	....	$58,948,125

In using the statistics in the foregoing tables reference should also be made to the detailed tables under the respective captions further on in this volume, where many explanatory notes as to the statistics will be found. The following notes refer to the four preceding tables: (*a*) Except where otherwise specified. (*b*) Not enumerated. (*c*) Amount or value of crude mineral. (*d*) Partly estimated. (*e*) Estimated. (*f*) Amount recovered as a by-product. (*g*) Barrels of 300 lb. (*h*) Barrels of 400 lb. (*i*) Includes manganiferous iron ore:

Barytes.—The production in 1898 was 28,247 short tons ($112,988), against 26,430 ($105,720) in 1897. Of the production in 1898 Missouri furnished 14,968 tons, the remainder being obtained in Virginia and North Carolina.

Bauxite.—The production in 1898 was 26,791 long tons ($66,978), against 20,590 ($51,475) in 1897. Of the production in 1898 Alabama furnished 13,848 tons, the remainder being mined in Georgia.

Borate of Calcium.—The production in 1898 was 15,335 short tons, against 19,400 in 1897. Most of this product is colemanite, mined in California.

Bromine.—The production in 1898 was 486,978 lb. ($136,354), against 487,149 ($136,402) in 1897. Of the production in 1898 liquid bromine constituted 356,978 lb., against 356,949 in 1897, the remainder of the output being the bromine equivalent of potassium bromine produced in Michigan.

Calcium Carbide.—The production in 1897 was 1,925 short tons ($134,750). It was probably larger in 1898, but statistics could not be obtained.

Carborundum.—The production reported by the sole producer was 1,594,152 lb. ($151,444) in 1898, against 1,242,929 ($153,812) in 1897. The decline in value was due to the increased proportion of the cheaper grades marketed.

Cement.—The total production of Portland cement in 1898 was 3,584,586 bbl. of 400 lb., valued at $6,168,106, against 2,430,903 ($3,724,905) in 1897. The largest part of the increase was due to the Lehigh district of Pennsylvania and New Jersey, but New York, Ohio and other States also showed important gains. There was remarkable prosperity in this industry on account of the increased demand and higher prices. The production of natural rock cement in 1898 was 8,161,078 bbl. of 300 lb., valued at $3,819,995, against 7,890,573 ($3,976,050) in 1897. The increase in the production of natural rock cement was due chiefly to the Indiana-Kentucky district, where the remarkably low prices brought about by excessive competition among the producers undoubtedly stimulated consumption. Many other parts of the United States showed a falling off, owing to the growing tendency to substitute Portland cement for natural rock. The production of slag cement in 1898 was 157,662 bbl. of 400 lb., valued at $235,721, against 40,000 bbl. ($60,000) in 1897.

Chrome Ore.—The production in 1898 was 100 long tons ($1,000) against 150 ($1,550) in the previous year. Chrome mining in California ceased in 1898 and the entire output was derived from Pennsylvania and Maryland.

Clay.—The value of brick and other clay products made in the United States in 1898 was $58,470,543 against $56,487,527 in the previous year.

Coal and Coke.—The total production of coal in the United States in 1898 was 218,106,519 short tons ($209,999,991) against 200,857,211 ($206,024,234) in 1897. The production of anthracite, all of it from Pennsylvania with the exception of an insignificant amount from Colorado, was 52,848,605 short tons ($81,445,937) in 1898 against 52,645,133 ($85,857,717) in 1897. Kentucky produced 49,889 short tons ($134,700) of cannel coal against 56,511 ($153,145) in 1897. The remainder of the output each year was bituminous coal, of which Pennsylvania and the Central States are the largest producers, their relative rank in 1898 having been Pennsylvania, Illinois, West Virginia and

Ohio in the order named. There was an increase in the output of most of the important coal-producing States in 1898, Illinois and Iowa being the only important ones which showed a decrease. The total production of coke in 1898 was 15,897,797 short tons ($30,505,563) against 12,768,891 ($23,203,920). Pennsylvania furnished about two-thirds of the output each year.

Coal Tar.—By-product coke oven plants turned out 16,021 short tons in 1898 against 13,304 in 1897. There are no statistics as to the large production of this substance by illuminating gas works.

Cobalt Oxide.—The production of cobalt oxide in 1898 was 9,640 lb. against 19,300 in 1897. The value was the same each year, namely $1·60 per lb.

Copperas.—The production in 1898 was 11,285 short tons ($58,105) against 15,702 ($78,660) in 1897. The decrease was due to discontinuance of this manufacture by two important producers. The chief feature of the year in this industry was the organization of the American Steel & Wire Co., which controls all the wire and rod mills recovering copperas as a by-product. The above statistics do not include copperas converted into Venetian and Indian reds at the works of original production.

Copper Sulphate.—The production in 1898 was 55,119,361 lb., against 46,128,822 in 1897. The amount recovered as a by-product, chiefly by gold and silver refiners, was 28,061,501 lb. in 1898 and 16,731,247 in 1897. The remainder of the output each year was made from metallic copper previously reported in the production of that metal. The average value of copper sulphate at New York per 100 lb. was $3·41 in 1898 against $3·86 in 1897.

Corundum and Emery.—The production of corundum in 1898, all of it from North Carolina and Georgia, was 786 short tons ($63,630) against 293 ($19,810) in 1897. The production of emery, most of it from Massachusetts, was 2,956 short tons ($143,800) against 1,900 ($92,000) in 1897. The resources of the United States in both corundum and emery of the highest quality are large and an increasing production is to be looked for. The production of steel emery or crushed steel in 1898 was 660,000 lb. ($46,200) against 647,800 ($51,824) in 1897, the entire make each year being supplied by the Pittsburg Crushed Steel Co.

Feldspar.—The production in 1898 was 21,350 long tons ($107,147) against 21,901 ($111,392) in 1897, Pennsylvania being the chief producer each year.

Fluorspar.—The production in 1898 was 12,145 short tons ($86,985) against 4,739 ($36,264) in 1897, Illinois and Kentucky furnishing the entire output.

Fullers Earth.—The output in 1898 was 15,553 short tons ($87,365) against 17,195 ($92,398). The most part each year was dug in the vicinity of Quincy, Fla. The new deposits near Ocala, Fla., did not become productive.

Garnet. The production in 1898 was 2,882 short tons ($82,930) against 2,261 ($66,353) in 1897, the output each year being furnished by New York, Pennsylvania and Connecticut. The domestic resources of this mineral are large, but the demand for it is limited.

Graphite.—The production of crystalline graphite in 1898 was 1,647,679 lb. ($148,291) against 993,138 ($44,691) in 1897. The proportionately large increase in value was due to the increased demand for crucibles for the manufacture of projectiles during the Spanish war. The larger part of the product in 1898 was obtained from Ticonderoga, N. Y., but a considerable quantity was mined in Pennsylvania, where some of the old mines were reopened. The production of amorphous graphite in 1898 was 1,200 tons ($11,400), these figures being unchanged from the previous year. One company produced 185,647 lb. ($11,603) of artificial graphite against 162,382 ($10,149) in 1897.

Gypsum.—The production increased from 300,369 short tons in 1897 to 348,686 in 1898. These figures represent the amount of crude rock quarried. The more part of the production is marketed as stucco or plaster of paris.

Iron Ore.—The production in 1898 was 20,655,865 long tons against 18,235,406 in 1897, these figures being exclusive of the production of manganiferous iron ore, reported separately under manganese. The increase in the production was due chiefly to the Lake Superior ranges, the Southern States showing only a comparatively small gain.

Lead White, Red Lead and Litharge.—The production of white lead in 1898 was 93,172 short tons ($9,391,738) against 105,804 ($9,522,360) in 1897; of red lead 9,160 ($916,000) against 7,798 ($744,709); of litharge 7,460 ($710,192) against 8,591 ($773,190); of orange mineral 541 ($108,200) against 477 ($76,320). The most of these products is obtained by the corrosion of pig lead, but a small part of the white lead product is made directly from ores.

Limestone for Iron Flux.—Iron smelters consumed 5,275,819 long tons in 1898 against 4,247,688 in 1897, the increase being caused by the greater make of pig iron.

Lithographic Stone.—Utah produced 112 short tons in 1898, this substance being reported for the first time in the United States.

Magnesite.—California produced 2,355 short tons ($9,420) in 1898 against 1,907 ($7,628) in the previous year.

Manganese Ore.—The production of manganese ore, including manganiferous iron ore, was 217,782 long tons in 1898 against 158,600 in 1897. Included in these figures is 142,318 tons of manganiferous iron ore from Michigan and Wisconsin in 1898 against 80,261 in 1897, and 47,470 tons of franklinite residuum from New Jersey in 1898 against 50,000 in 1897.

Mica.—The production of sheet mica in 1898 was 109,968 lb. ($91,432) against 118,852 ($83,298) in 1897. The production of scrap mica was 3,529 short tons ($39,837) against 2,882 ($28,820), the values of scrap mica being reckoned for the product before grinding. Practically the entire production of mica is made in New Hampshire and North Carolina.

Mineral Wool.—The production in 1898 was 6,560 short tons ($70,314) against 5,617 ($61,494) in 1897. A part of this product was made from slag and a part by the fusion of natural rock, the latter being the more valuable.

Molybdenite.—This mineral was produced in the United States in 1898, for the first time, to the amount of 8 short tons valued nominally at $50 per ton.

Monazite.—North Carolina produced 150,000 lb. ($7,500) against 40,000 ($2,000) in 1897. Notwithstanding the increase no important future is to be looked for in this industry on account of the vast supplies of cheaper mineral available in Brazil.

Ocher and Oxide of Iron Pigments.—The production of ocher, umber, sienna and natural oxide of iron ground pigment, the last being known commonly as "metallic paint," was 41,950 short tons ($461,450) in 1898 against 42,590 ($468,490) in 1897. The values are reckoned nominally at $11 per ton each year. These products merge into one another so that no satisfactory division of the statistics is possible. Pennsylvania is the largest producer.

Petroleum.—The total output in 1898 was 51,774,465 bbl. ($42,100,522) against 57,124,783 ($39,372,111) in 1897. The great falling off was due to New York and Pennsylvania and the Lima field of Ohio and Indiana. West Virginia, California and Texas made increased outputs. The above values are calculated for crude oil at the wells. The average was considerably higher in 1898 than in the previous year.

Phosphate Rock.—The production in 1898 was 1,257,645 long tons ($4,355,025) against 1,007,367 ($3,022,101) in 1897. The increase was due especially to Tennessee and the land rock deposits of South Carolina. Phosphate mining was generally prosperous in 1898 on account of the higher prices.

Salt.—The domestic output of salt increased from 15,822,923 bbl. in 1897 to 18,756,394 in 1898, Kansas, Michigan and New York each contributing largely to the increase.

Silica.—The production of vein and dike quartz in 1898 was 35,593 short tons (73,313) against 28,407 ($55,817) in 1897. This material was used chiefly in pottery, for packing acid towers, and for grinding for various purposes. The production of grindstones, which are made out of quartzite, or a very hard sandstone, amounted to 38,859 short tons ($438,675) in 1898 against 36,502 ($342,186) in 1897. These were produced entirely in Ohio and Michigan. The production of oilstones, scythestones and whetstones in 1898 was valued at $107,990 against $97,229 in 1897. Utah produced 144 short tons of pumice stone, valued nominally at $720, against 1,700 ($8,500) in 1897. There was a production of 1,865 short tons of tripoli, valued at $3,367, against 1,631 ($5,475) in 1897, most of this product being obtained in Missouri. The production of diatomaceous earth in the United States in 1898 was 1,392 short tons ($11,002) against 3,000 ($30,400) in the previous year.

Slate.—The production of roofing slate in 1898 was 1,136,632 squares ($2,958,496) against 932,124 ($2,829,526) in 1897. Both Pennsylvania and Vermont made large increases, but this, together with the fierce competition among the producers, reduced the average price of the product. The production of slate manufactures, chiefly blackboards and structural material, was valued at $528,856 in 1898, against $595,105 in 1897. School slates are not included. The production of slate pigment, including Baraga graphite and various kinds of mineral black, was 7,886 short tons, against 6,857 in the previous year, the values of the products after grinding being respectively $70,671 and $57,863.

Soda.—The production of soda and soda products from salt, reduced to a common basis of 58% soda ash, was 340,622 metric tons in 1898, against 277,072 in 1897. The average value of 58% ash at the works was $11·98 per metric ton in 1898 against $14·12 in 1897. California and Nevada produced 7,150 short tons ($94,650) of natural soda, basis 58% in 1898, against 6,350 short tons ($95,400) in 1897.

Strontium Sulphate.—Ohio produced 2 tons in 1898 against 40 tons in 1897, the mineral being got for experimental purposes only.

Sulphur and Pyrites.—Louisiana, Nevada and Utah produced 2,726 long tons of sulphur in 1898 against 1,690 in 1897. The average price of Sicilian seconds at New York was $21·92 per ton, against $20·60 in the previous year. The domestic production of pyrites in 1898 was 191,160 long tons, worth an average of $3·08 per ton, against 133,368, averaging $3·04 per ton in 1897. Virginia was the largest producer each year, Massachusetts ranking second. Important new mines were opened in 1898 in Virginia, New York and California. Sulphuric acid was recovered as a by-product in roasting blende and pyrites, chiefly the former, to the extent of 47,558 short tons against 42,352 in 1897, the figures being reduced to a common basis of 66° B. Concentrated acid of the latter strength averaged $20.40 per 2,000 lb. at New York in 1898 against $16·52 in 1897.

Talc and Soapstone.—The production of soapstone for slabs and other manufactured articles in 1898 was 18,862 short tons ($158,635), against 16,904 ($169,040) in 1897. The decrease in value was due to a largely increased production of very low grade stone. The production of common talc, mostly ground to powder, was 9,112 short tons ($78,645), against 10,164 ($90,908) in 1897. This was produced chiefly in North Carolina, Pennsylvania and Vermont. The production of fibrous talc, all of it from St. Lawrence County, N. Y., was 54,807 short tons ($285,759), against 52,836 ($283,685) in 1897.

Tungsten Ore.—There was a production in 1898 of 78·865 long tons of tungsten ore, all of it from South Dakota, Arizona and New Mexico, which was valued nominally at $17,398 at the mines. Connecticut produced about 400 tons of low grade ore, which was held at the mine awaiting dressing.

Uranium Ore.—Colorado produced 33 short tons, valued at $500 per ton, in 1898, against 17 short tons, valued at $530 per ton, in 1897.

Venetian Red.—The production of Venetian and Indian reds in 1898 was 3,270 short tons ($64,570), against 2,196 ($40,790) in 1897. These figures include only the output at works where the original copperas was made, and do not include any that may have been made by second hands.

Wulfenite.—Arizona shipped 12 short tons of this mineral, valued at $12 per ton, in 1898, but this was probably in part the accumulation of previous years.

Zinc Ore.—The amount of zinc ore of domestic origin exported from the United States in 1898 was 11,782 short tons (299,870), against 9,251 ($211,350) in the previous year. Most of the ore exported was mined in New Jersey.

Zinc Sulphate.—There was a production in 1898, by one concern, of 145 short tons, valued at $32 per ton, recovered as a by-product from mixed ores.

Zinc White.—The production in 1898 was 32,747 short tons, against 26,262 in 1897. The average value at the works was $68 per ton in 1898 and $64 20 in 1897. Nearly all the American production of zinc white is made directly from ores, most of it from ores mined in New Jersey.

ALUMINUM AND ALUM.

In previous volumes of The Mineral Industry it has been the custom to treat alum, aluminum, bauxite, corundum, emery and cryolite under separate captions, but since bauxite and cryolite are the raw materials from which both alum and aluminum are made, and since the industries are so completely interwoven that a rational separation is impossible, and since corundum, now used only as an abrasive, may be used some day as a source of the metal, it seems best to consider the entire subject, from the production of the crude minerals to the manufacture of the first products, under the general caption, "Aluminum and Alum." In this view aluminum, alum, and the aluminum minerals—bauxite and cryolite, corundum and emery—bear the same relation to one another as do copper, copper sulphate or bluestone, and copper ores.

I. Bauxite.

The domestic production of bauxite in 1898, as in 1897, was derived entirely from Georgia and Alabama, where there were four producers, the American Bauxite Co. having been succeeded by the Dixie Bauxite Co. The production in Alabama in 1898, reported by the Alabama Geological Survey, was 13,848 tons of 2,240 lb. The production in Georgia is arrived at by difference between the Alabama production and the total of the reports received directly from the producers.

PRODUCTION, IMPORTS, EXPORTS, AND CONSUMPTION OF BAUXITE IN THE UNITED STATES.

Year.	Production.					Imports.		Exports.		Consumption.	
	Alabama	Georgia.	Total.								
	Lg. Tons.	Lg. Tons.	Lg. Tons.	Value.	Per Ton.	Lg. Tons.	Value.	Lg. Tons.	Value	Lg. Tons	Value.
1894	8,687	2,045	10,732	$42,928	$4·00	1,028	$6,661	....	....	11,760	$49,589
1895	14,000	4,800	18,800	56,400	3·00	5,797	34,782	....	....	24,597	91,182
1896	9.796	7,300	17,096	42,740	2·50	2,119	10,477	....	....	19,215	53.217
1897	13,083	7,507	20,590	51,475	2·50	2,645	10,515	2,537	5,074	20,708	56.916
1898	13,848	12.943	26,791	66,978	2·50	1,201	4,238	1,000	2,000	26,992	69.216

Several new discoveries of bauxite were made in 1898, the most important being the Wilson & Shaw mines, 3 miles from Rock Run Furnace, Ala., and the Estes mine in the same vicinity. The former, which is known also as the Klondike, has been equipped with a mining plant. Other new discoveries were the Miller and Saylor mines, 1 mile east of Silver Creek, Ga., and the Reynolds mine, 3 miles east of Cunningham, on the Southern Railway. The Arkansas bauxite deposits remained unproductive, but the incorporation of a company called the Arkansas Alumina Works, to manufacture aluminum hydrate and sodium aluminate at Mablevale, Ark., was reported. This company, which owns bauxite mines at Bryant and Mablevale, is putting up experimental works at Mablevale of an estimated capacity of 1,500 lbs. of product per day. It is aimed to determine if this concentration of the raw material on the spot will make it possible to work successfully the Arkansas bauxites.

Preparation for Market.—Up to 1895 but little attention was devoted to the preparation of bauxite for market. In that year a mechanical drying furnace was introduced at Hermitage, Ga., by the Republic Mining and Manufacturing Co., and since then all the producers have equipped themselves with facilities for artificially drying their ore. The latest type of drying furnace introduced in this business is an iron cylinder 30 ft. × 30 in., inclined 8 to 12 in. in its length, with scoops attached to the lining, which serve to raise the ore and shower it through the flames. In general the ore is about 14 minutes in making the passage through the cylinders.

Low grade bauxite is concentrated or washed for market by the Dixie Bauxite Co. and the Georgia Bauxite Co., which employ the regular log washers used extensively in the South for washing iron ores and phosphate rock. Clay is the chief impurity to be eliminated. Some of the producers have tried to separate this by screening, but this process has been found inexpedient.

Market Conditions.—Although the production showed such a material increase, prices remained the same, notwithstanding the optimistic expectations of some of the producers. However, in view of the large reserves of this mineral in Georgia and Alabama, the cheapness with which it can be mined, and the present demand for it, there is no reason to look for any important increase in price so long as existing conditions continue. The energies of the producers should be devoted, therefore, to economies in the cost of production, and some of them have been doing this with the result that their mineral marketed realized for them somewhat larger profits in 1898 than in the previous year.

The principal markets for bauxite in the United States are Pittsburg, Pa., and Niagara Falls, N. Y., where it is used for the manufacture of aluminum; and Buffalo and Brooklyn, N. Y., Everett and Woburn, Mass., Philadelphia and Natrona, Pa., and Bayonne and Camden, N. J., where it is used for the manufacture of alum and aluminum sulphate. In general the bauxite mines of Georgia and Alabama are situated from 2 to 8 miles distant from the nearest railway shipping point, to which the ore is carried in wagons. The freight rate from point of shipment to point of consumption ranges from $3·75 to $5·15 per 2,240 lb. The price for the mineral remained the same in 1898 as in 1897, *i.e.*, $7 per ton delivered at the works of the consumer, which means $3·25 to $1·85 to the producer. One of the largest producers in 1898 realized an average of $2·80 per ton, not including cartage or railway freight.

Export Trade.—The exportation of bauxite, which was begun in 1897, fell off heavily in 1898, the failure being attributed to the high ocean freights which prevailed. It is worthy of attention, however, that the foothold which has been gained in the foreign market has been retained.

BAUXITE IN FOREIGN COUNTRIES.

The world's supply of bauxite is derived chiefly from France, Ireland and the United States, the production statistics being given in the following table:

PRODUCTION OF BAUXITE IN THE PRINCIPAL COUNTRIES, IN METRIC TONS.

Year.	1891.	1892.	1893.	1894.	1895.	1896.	1897.
France	22,260	23,298	33,923	26,032	17,958	33,820	41,740
United Kingdom	10,939	7,439	8,880	8,097	10,574	7,365	13,449
United States	3,962	9,957	11,204	10,904	19,101	17,370	20,919
Total	37,161	40,694	54,007	45,033	47,633	58,555	76,108

Austria.—Bauxite is found in Austria at several places, having been first noted in the valley of a branch of the river Save, in the northeast part of Carniola.* This deposit is about 4 m. in thickness, and occurs at the junction of Triassic and Jurassic limestones. The best samples show 64% Al_2O_3, the yellowish show 58% Al_2O_3 and from 8 to 9% Fe_2O_3, while the dark yellow or brown-colored mineral has from 10 to 30% Fe_2O_3, and from 12 to 15% SiO_2. Analyses of a number of samples taken in 1895 show from 67·6 to 51·4% Al_2O_3, from 0·7 to 19·3% Fe_2O_3, 5·9 to 14·4% SiO_2, traces of manganese, lime, magnesia, and titanic acid, and from 23·1 to 12·2% water. Another deposit of bauxite is known in Styria, on the northern slope of the Dobrol Mountain, near Prichova. This deposit is superficial, forming patches thickly interspersed through the clays which rest on a base of Mesozoic limestone. Analyses show 58·6% Al_2O_3, 18·7% Fe_2O_3, and 11% SiO_2. No considerable quantities have been mined from either of these deposits.

France.—According to G. L. Bourgerel, the bauxites occurring in the south of France are employed for various purposes. The red is used for the manufacture of aluminum, and the white for the preparation of aluminum sulphate, intermediate varieties being mixed with plastic clay and made into fire bricks. The red variety contains 62 to 50% of alumina, 24 to 28% of ferric oxide, and 1 to 7% of silica; while the white contains 74 to 65% of alumina, 0·25 to 3% of ferric oxide, and 12 to 18% of silica. The bauxite bed, which is very thick, occurs in limestone. Its formation is believed to be due to an interaction between quicklime and hot concentrated solutions of aluminum and iron chlorides.

Ireland.—The British Aluminum Co. was obliged to give up the use of Irish bauxite for aluminum manufacture on account of its high tenor in silica. The production of bauxite in Great Britain fell off to 12,600 metric tons in 1898.

TECHNOLOGY OF BAUXITE.

Use in Paper-making.—Franz Mueller, of Hugohuette, patented in Austria a process for the recovery of by-products from the waste lyes of wood pulp mills by the use of bauxite.† The waste lyes and wash waters are evaporated in the

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* R. Helmhacker, *Engineering and Mining Journal*, Oct. 15, 1898.

† Kuhlow's *German Trade Review and Exporter*, No. 681; originally from *Papier Zeitung*, 1898.

ordinary manner. At any time during the evaporation pulverized bauxite is added in sufficient quantity to convert all its alumina into aluminate by the action of the alkali contained in the waste lye. If the lye contains sulphur compounds a corresponding amount of limestone or caustic lime should also be added. The waste lyes contain caustic and carbonated alkalies and alkali sulphides, with other alkali sulphur salts. On calcining the mixture these salts combine with the bauxite, according to the following equations:

(*I.*) $6NaOH+Al_2(OH)_6=Al_2(ONa)_6+6H_2O$.
(*II.*) $3Na_2CO_3+Al_2(OH)_6=Al_2(ONa)_6+3H_2O+3CO_2$.
(*III.*) $Na_2SO_4+4C=Na_2S+4CO$. (*IV.*) $Na_2SO_3+3C=Na_2S+3CO$.

There is much more than sufficient carbon contained in the organic matter in the lye to account for the following reaction:

(*V.*) $3Na_2S+3CaCO_3+Al_2(OH_6)=Al_2(ONa)_6+3CaS+3H_2O+3CO_2$.

After evaporating to dryness the mass is calcined and lixiviated in the ordinary manner, and the clear liquid containing the alkali aluminate in solution is treated in either of the two following ways:

(*a*) For soda pulp, CO_2 is passed into it, whereby aluminum hydrate is precipitated.

(*VI.*) $Al_2(ONa)_6+3CO_2+3H_2O=Al_2(OH)_6+Na_2CO_3$.

The solution of alkali carbonate is causticized with lime and the resulting lye used for making cellulose. The aluminum hydrate is separated by a filter press. The CO_2 may be obtained from a lime kiln, and the burnt lime used for causticising.

(*b*) For sulphite pulp, SO_2 is passed into the solution, precipitating aluminum hydrate as follows:

(*VII.*) $Al_2(ONa)_6+3SO_2+3H_2O=Al_2(OH)_6+3Na_2SO_3$.

These are separated as under *a*, and the recovered Na_2SO_3 is again used for boiling cellulose. The pure aluminum hydrate obtained may be used for making aluminum salts, or it may be treated with hot sulphuric acid and used for sizing paper. To counteract the inevitable loss of alkali a sufficient quantity of sodium sulphate or bisulphate and lime is added to the condensed lye before the bauxite.

*Commercial Analysis of Bauxite.**—W. B. Phillips and D. Hancock state that the formula given by Roscoe and Schorlemmer for bauxite, namely $(AlFe)_2O(OH)_4$, does not correspond with the composition of the American mineral, which appears to be a mixture of aluminum trihydrate, clay, and possibly a lower hydrate of aluminum. In determining the value of the mineral for alum manufacture it is usual to estimate the amount of available alumina by evaporating with sulphuric acid until acid fumes appear. Really a discrimination ought to be made between (1) the trihydrate, which is readily soluble and of greater value to the alum maker, and (2) the other aluminum compounds, which are only soluble with difficulty. Messrs. Phillips and Hancock propose

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* *Journal American Chemical Society*, 1898, 20(3), pp. 209-225.

to term the alumina soluble in sulphuric acid of 50° B. (sp. gr.=1·53) after one hour's treatment at 100° C. as "free alumina," that dissolved on evaporation of the acid to fumes as "available alumina," and the difference between them as "combined alumina;" and they describe experiments to prove that practically the whole of the aluminum trihydrate is extracted by the former treatment.

The following method of analysis is recommended: The sample is prepared by grinding to pass a 100-mesh sieve.

Moisture.—Two grams are heated to constant weight at 100° C.

Available Alumina.—Two grams are mixed with 10 c.c. of cold sulphuric acid (50° B.), and gradually heated until fumes appear, when the basin is covered with a watch-glass and the heating continued for 10 minutes. The liquor is then filtered, the residue washed, and the filtrate made up to 200 c.c., of which 50 c.c. are taken for the determination of alumina, ferric oxide, and titanium dioxide, and a second 50 c.c. for the determination of the latter two.

The first portion is diluted to 300 c.c., acidified with hydrochloric acid, and precipitated boiling with ammonia. The precipitate, dried and weighed, gives the alumina, ferric oxide, and titanium dioxide.

The second portion of the original solution is neutralized with ammonia, the addition of the latter being continued until a precipitate forms. This is redissolved in sulphuric acid, and the liquid is made up to 300 to 400 c.c. and boiled for an hour. Any iron present is reduced by means of sulphur dioxide, care being taken that the liquid smells of the gas throughout the boiling process. In this way a precipitate is obtained practically free from iron. The liquid is filtered while hot, and the precipitate dried, ignited, and weighed as titanium dioxide.

Ferric Oxide.—This is determined in the filtrate from the titanium dioxide by reduction with zinc and titration with permanganate.

Free Alumina.—Two grams of the sample are placed in an Erlenmeyer flask with 10 c.c. of sulphuric acid (50° B.). The flask, which is fitted with a perforated stopper, is kept in the hot-water bath for an hour, with constant shaking. The liquid is filtered and the alumina precipitated as before. A little iron is also precipitated, but this is separately determined and deducted. The titanium dioxide does not pass into solution.

Combined Alumina.—This is reckoned as the difference between the available and free alumina.

It is not advisable to determine silica in the insoluble residue by means of hydrofluoric acid, since some of the titanium dioxide is also volatilized. Titanium dioxide is always present in bauxite, and may amount to 4%.

II. CORUNDUM AND EMERY.

The only large producer of corundum in the United States in 1898 was the Hampden Emery and Corundum Co., of Chester, Mass., and Cullasaja, N. C., which owns a large amount of property in the South, but in 1898 operated only the Corundum Hill mine in Macon County, N. C. In the early part of the year corundum mining in the South was practically at a standstill, the Hampden Emery and Corundum Co. being the only concern that was putting any mineral on the market. In the second half of the year there was a large increase in the output, and there is not unlikely to be a further expansion in 1899, sufficient to satisfy the foreign trade as well as the domestic. The Isbell Corundum Co., of Clay County, N. C., which was engaged in the construction of a concentrating mill, described further on, put several sample lots of mineral on the market in 1899, but it is said to be inferior to Chester emery, which can be bought at 3 to 3·25c. per lb. in keg lots. The National Abrasive Co. commenced operations on an extensive scale and produced some mineral, mostly crystals of first class quality. The Lucas Corundum Mining and Manufacturing Co., of Franklin, Macon County, N. C., worked its mine only to July, producing about 100 tons of crude mineral, which was delivered at the mill and worked into grain, but it was found that the product could not be sold in competition with foreign emery except at a loss, and consequently it was not marketed. This production is included in our statistics.

PRODUCTION OF CORUNDUM AND EMERY IN THE UNITED STATES. (*a*)

Substance.	1895.			1896.			1897.			1898.		
	Tons. (*b*)	Value.	Per Ton.	Tons. (*b*)	Value.	Per Ton.	Tons. (*b*)	Value.	Per Ton.	Tons. (*b*)	Value.	Per Ton.
Corundum	385	$25,989	$67·50	250	$17,000	$68·00	293	$19,810	$67·61	786	$63,630	$80·96
Emery............	1,700	*c* 81,600	48·00	1,850	*c* 89,000	48·11	1,900	*c* 92,000	48·42	2,956	*c* 143,800	48·65
Steel emery.......	336	53,864	160·30	326	50,624	155·29	324	51,824	159·95	330	46,200	140·00
Total............	2,421	$161,453		2,426	$156,624		2,517	$163,634		4,072	$253,630	

(*a*) The statistics of production given in the above table have been changed considerably from previous volumes of THE MINERAL INDUSTRY owing to more precise returns, not only for 1898, but also for previous years. The change in the valuation of the products is due to basing this upon the price at the mine instead of at New York, as has been the case previously. However, in this industry the value figures are of slight significance owing to the great range between the different grades of these minerals. Consequently the difference in the annual average does not indicate change in market quotations so much as it does change in the proportion of different grades of mineral in the total. (*b*) 2,000 lb. (*c*) More or less nominal.

IMPORTS OF EMERY INTO THE UNITED STATES.

Year.	Grains.		Ore or Rock.		Other M'f'es.	Total Value.	Year.	Grains.		Ore or Rock.		Other M'f'es.	Total Value.
	Pounds	Value.	Long Tons.	Value.	Value.			Pounds	Value.	Long Tons.	Value.	Value.	
1891..	90,658	$3,729	2,530	$67,573	....	$71,302	1895..	678,761	$25,066	6,803	$80,386	$27,586	$133,038
1892..	566,448	22,586	5.280	95,625	$2,412	120,623	1896..	751,464	26,520	6.289	119.667	1.971	148,158
1893..	516,953	20,073	5,066	103.875	3,819	127,767	1897..	520,095	20.022	5,209	107.649	2,211	129,882
1894..	597,713	18,645	2,804	51.487	1,847	71,973	1898.	577,655	23,320	5,547	106,269	3,810	133,399

C. N. Jenks, a leading authority in the American corundum business, states that there are 10,000 tons of the highest grade of corundum in sight in the Corundum Hill mine, while the Buck Creek mine has large deposits, covering over 300 acres, and the Laurel Hill mine in Raybun County, Ga., has many thousands of tons of the best quality of corundum readily accessible. These mines are the property of the Hampden Emery and Corundum Co., which for various reasons has been holding its large supply in reserve. The great obstacle to the development of the corundum industry in the United States and abroad is the high price at which this mineral is held, namely, 7 to 10c. per lb., which is out of proportion to the price of other abrasives. It is conceded by the trade that the best corundum will do twice the work of the best emery, but when emery can be bought as at present at 3 to 3·25c. per lb. (best Chester, itself the best in the world) it is obvious that corundum will have to be offered at a price not exceeding 6c. per lb. in order to command a ready sale. The expense of exploring new corundum deposits is so great, and the chance of obtaining a considerable quantity is so small (most of the valuable deposits being controlled by the companies now in the field), that there is little inducement for outsiders to engage in the industry, while many of the mines are so remote from the railway, and the expense of transportation is so great that the cost of production must, for a considerable time at least, continue to be high. First class corundum realized from $80 to $125 per 2,000 lb. at the mines in 1898 and about $150 in the market.

The production of emery in the United States in 1898 was confined to Chester, Mass., and Peekskill, N. Y., the Tanite Emery Wheel Co. and the Crescent Emery Mills operating at the latter place. Their operations were conducted on a small scale and their product is inferior in every respect to the Chester. This grade of emery can be purchased for 2 to 2·5c. per lb., while Chester emery fetches 3 to 3·50c.

It has been demonstrated that a clean separation can be effected between the iron and magnetite which constitutes the Chester emery, but the quality of the mineral thus obtained is very inferior to the best grade of North Carolina and Georgia corundum. However, a process of preparation based on this separation has been introduced by the Hampden Emery and Corundum Co. at Chester, Mass., with the result of making a product which is 50% more efficient than the best Naxos emery, which is itself more efficient than theTurkish. Chester emery is consequently rapidly taking the place of the foreign mineral, and is now used in nearly every emery wheel factory in the United States and Canada. The deposit of emery at Chester, Mass., shows an immense amount of high grade mineral in sight.

With its great resources in emery and corundum, consequently, we may look to see the United States assume a commanding position in the market for abrasive material. The only rivals of these products, namely, garnet, steel, etc., are employed generally for inferior purposes. Reference to the production of an artificial corundum will be found under the caption, "Chromium and Chrome Ore," elsewhere in this volume. This material has not yet appeared in the market. C. M. Pielsticker, of Wembley, England, has patented a substitute for emery, which consists of New Zealand iron sand, from which the silica is removed, after which it is mixed with carbon and heated in a crucible to produce fine particles of steel, the whole being then quenched in coal oil, or oil and water, to increase the hardness. This quality may be further increased by adding a compound of phosphorus to the mixture in the crucible. The hard, fine particles, containing 5% or more of carbon and some phosphorus, are mixed with a binding material and made into wheels in the usual manner.*

The production of steel emery, or crushed steel, in the United States in 1898 was 660,000 lb. ($46,200) against 647,800 ($51,824) in 1897, the entire make

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* English Patent No. 284, Jan. 5, 1898.

each year being supplied by one company, the Pittsburg Crushed Steel Co. An imported steel emery, known as "Krushite," is used to some extent.

Corundum Mining in North Carolina and Georgia.—Leverett S. Ropes contributes the following notes concerning the industry in these States in 1898:

"Corundum was produced in Western North Carolina and North Georgia in 1898 by the Hampden Emery and Corundum Co. (operating only one of its mines, namely, the Corundum Hill), the National Abrasive Co., of Waynesville, N. C. (mining a mixture of garnet and corundum on Sugar Loaf Mountain; sapphire corundum at its Reed mine on Wantaga Creek, in Macon County; and corundum at the Sheffield mine on Cowee Creek, in the same county), and a few smaller producers. The principal supply of the second named company was the garnet-corundum mixtures from Sugar Loaf, the Reed mine where the ore bodies have been opened up to considerable depth, having been operated for only a short period. At the Sheffield mine considerable difficulty is experienced in separating the corundum from the gangue, the latter being a dense, hard hornblende, troublesome to crush, and too near the specific gravity of corundum to be separated with any degree of ease.

"Garnet and mixtures of corundum and garnet, 1:5, ranged in price from $25 to $80 per 2,000 lb., while 80% corundum and upward brought $80 to $125, and some small lots for special purposes realized higher prices.

"At Corundum Hill about 20 men were employed in the mine and mill. The ore from the mine is treated in the same primitive way as heretofore, the 'chasers' with wooden rolls (a crude type of the modern Chilean mill) and long tom wash boxes being used. The mineral is cleaned to a greater or less degree of purity, depending on the nature of the material coming from various parts of the mine, sized through a 12-mesh screen, and shipped to Chester, Mass., where it is finished for the market. The maximum capacity of the mill is 6 to 8 tons per week with 8 to 10 men, about 7 to 8 tons of crude ore being handled in 10 hours. The Laurel Creek and Buck Creek mines of this company have been idle since the fall of 1893.

"In North Georgia, along the Track Rock and Young Harris belts, no work was done in 1898 except some exploration work on one or two properties near Young Harris, which gives promise of developing into good mining.

"The American Corundum Co.'s mill, at Ellijay, in Macon County, N. C., was idle, the company being involved with the Foster propositions mentioned further on. The National Abrasive Co. secured options on a tract of land lying on the divide of the Blue Ridge Mountains and along the western slope in Clay County, N. C., and was engaged in testing the ores. These deposits are radically different from those of the old mines from which all corundum heretofore mined has come.

"All the older known corundum deposits are in the chrysolite and peridotite series of rocks associated with serpentine, talc, chlorite, vermiculite, asbestos, and kindred minerals, quartz, tourmaline, spinel, garnet, magnetite, chromite, and staurolite, being of frequent occurrence in the veins. In the Blue Ridge region the corundum occurs (sometimes associated also with garnet, but this only as a marginal occurrence) in a stratum or strata of mica schist in which the percentage of corundum runs as high as 40. It is estimated by those interested in developing the deposits that a supply of 10 to 15% ore can be furnished the mill, and that the outcrop indications are such as to insure a reliable output of this grade.

"It is in this range, 1 mile east, that what is locally known as the Foster mine was explored and partially developed about two years ago. Surveys were made for water power development, many locations being available, and affording from 50 h.p. upward, under heads varying from 20 ft. to 1,000 ft. Plans and specifications for a mill and concentrating plant were completed, when work was suspended on account of financial difficulties involving certain of the persons interested in the undertaking, and the property was sold to satisfy judgments. This mine, together with a large area covering the lode, with the exception of that held under option by the National Abrasive Co., and what is known as the Heaton mine (about 120 acres), is now controlled by a company of Northern and Southern men, which contemplates erecting an 80-ton mill in 1899. The distance to the railroads and wretched wagon roads are the greatest drawbacks to development of this region.

"Parallel with the Blue Ridge formation, about 5 or 6 miles northwest, is the Buck Creek formation, consisting of heavy beds of chrysolite and hornblendic rocks, with which are associated massive corundum, crystals and sand, with some gem material. In this belt are found the Isbel, Herbert, and 'old' Buck Creek mines. The first, at the foot of the Chunky Gal Mountain, formerly known as the Shooting Creek corundum mine, is situated on the headwaters of Shooting Creek, 30 miles from the railroad. Considerable corundum has been produced here, but little of it has gone into the general market. The mill was not operated in 1898. Two years ago, in connection with Mr. Fred Isbel, I arranged a scheme for washing the ore, adopting jigs and Frue vanners, which were used for some time, proving highly satisfactory, furnishing an almost pure mineral. Working on this line, a new and much larger mill has been designed, using two double two-compartment and four double three-compartment Harz jigs, Frue vanners, rolls, elevators, rubbers (cones), conveyors, etc. The problem of crushing the ore was a difficult one, the gangue consisting of decomposed hornblende with included masses, or balls, of undecomposed rock, chrysolite, etc. The decomposed rock is tougher than fire clay, and will not 'wash,' instead packing in the rolls and crushers, clogging them even under a strong current of water. The E. P. Allis Co., which furnished the plant, has supplied a heavy dog-toothed roll, the teeth heavily backed, and those of one roll meshing between those of the opposite. Two sets of plain rolls will assist in crushing down and grading, while the cones will be used to scour the corundum. This is accomplished by keeping the grains in motion, so that all foreign matter is removed by abrasion of the particles upon each other. This plant complete will cost about $20,000, and will be put in operation in 1899.

"The interests of the Herbert estate, on Buck Creek, about 3 or 4 miles northeast of the Isbel mines, are being investigated by New York parties. A ton of corundum has been shipped to be tested as to quality and system of dressing. Nothing is being done at the old Buck Creek mines, they being too far from the mills to be worked economically.

"The occurrence of corundum in a great variety of forms and rocks, of many degrees of hardness, and of varying specific gravities, makes the problem of concentrating and cleaning a difficult one, and one demanding individual consideration in each case. Careful study of the mechanical nature of the gangue rock with reference to crushing and liberating the mineral, a proper arrangement of washers and recrushing machinery to prevent an excess of 'powders,' and, on the other hand, to produce the largest amount of marketable 'grains,' is absolutely necessary for successful, economical treatment of the ores as they come from the mines. The corundum ores of the Blue Ridge mica schist belt offer the most easily treated material found in the two States. They require powerful crushers, but the difference in the specific gravity of the gangue and corundum and the micaceous nature of the former greatly facilitate concentration, and the corundum separating freely from the inclosing gangue, but little subsequent cleaning is necessary. A thin film of graphite is frequently noticed coating the crystals, and this seems to have prevented strong adhesion.

"The area, about 8 miles square, covering the Blue Ridge and Buck Creek formations, promises to become a large producer of fine abrasives in the near future. Within this area, moreover, can be produced a large amount of corundum which will not meet the requirements of the abrasive market. This can be mined and concentrated cheaply to a product containing 95% of aluminum oxide. A market for this material for some other purpose ought to be found. Cheap water power is available in the near vicinity.

"The outlook is altogether favorable for the opening of several new corundum mines which should show a production in 1899. Systematic mining and milling practice and careful study of the geology and mineralogy of corundum ores are gradually bringing the industry to a satisfactory basis, and the close of 1898 witnessed more development work being done and more money being judiciously invested than any year in the history of the industry."

Canada.—Explorations were continued in the new corundum field of Ontario with the result of proving the existence of even greater resources in this mineral than was supposed at first. Exploitation of the deposits has not yet begun, but it is expected that this will be undertaken during 1899. With this in view the government has drawn up regulations for the disposition of the corundum-bearing lands. Hereafter the mineral and mining rights in such lands can be acquired only under the leasehold system—the rental for the first year being 60c., and for subsequent years 15c. per acre. Instead of allowing speculators to take up and hold lands with a view to selling out their interests to miners and capitalists at a large profit, it is proposed that the advantage of acquiring lands upon the lowest terms shall go to the miner and manufacturer direct; and in the case of parties who will undertake to conduct mining and milling operations on the largest and completest scale, and who can furnish satisfactory assurances that they possess the requisite capital for the proposed operations (including separation of the ore from its gangue, milling for abrasive uses, manufacture of abrasive goods, and the production of aluminum), the government may concede a preference in the selection of mineral lands. It is also provided that the government shall have power to require that all corundum mined from lands leased under the regulations shall undergo certain processes of treatment and milling at works to be erected in the province to prepare it for market; and may further require, from time to time, as circumstances appear to warrant, that works be established in the province for the manufacture of all commercial products for which the mineral is economically adapted.

The occurrence of corundum in Ontario was described briefly in THE MINERAL INDUSTRY, Vol. VI. For further particulars, especially those of an historical, geological, and mineralogical nature, together with results of tests for the concentration of the mineral, reference should be made to the report of the Bureau of Mines of Ontario, Vol. VII., Third Part, 1898, and a paper by Archibald Blue on "Corundum in Ontario."[*] Prof. Courtenay De Kalb, of the School of Mining at Kingston, Ont., has been making experiments with respect to the concentration of the Ontario corundum with extremely satisfactory results as to the production of a high grade mineral free from the magnetite with which it is associated in the deposits. We have had reports from the trade, however, to the effect that the Ontario corundum is inferior in quality as an abrasive. The Canadian authorities contemplate not only a market for it for this purpose, but also expect it to form the basis of a home aluminum industry.

India.—Outside of the United States, India probably has the largest resources in corundum, which is said to be very freely distributed among the crystalline rocks of the southern part of the Empire.[†] The mines in the districts of Salem, and South Canara, in the Madras Presidency, are considered to have great possibilities, although not worked at present. The total production of the Madras Presidency is reported at 3 tons of 2,240 lb. in 1894, 13 in 1895, 20 in 1896, and 203 in 1897. Mysore produced 38·6 in 1895, 18 in 1896, and 12·5 in 1897. An immense deposit is said to exist near Pipra and Kodopani in South Rewah in Madras.

III. CRYOLITE.

The importations of cryolite into the United States, which came entirely from Greenland, as in previous years, are given in the following table. These importations were made by the Pennsylvania Salt Manufacturing Co., of Natrona, Pa., which is the possessor of the exclusive privilege to import this mineral into North and South America. The remainder of the output of the Ivigtut mines is shipped to Copenhagen. According to a report furnished by the Danish government the total production in Greenland in 1895 was 12,287 metric tons and 6,058 in 1896.

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[*] American Institute of Mining Engineers, Buffalo meeting, October, 1898.

[†] David Hooper, *Review of Mineral Production in India*, 1897. For particulars as to the mineralogical and geological occurrence of corundum in India reference should be made to the *Agricultural Ledger No. 16*, of 1896, and in the *Records of the Geological Survey*, Vol. XXIX., Part 2, pp. 39-50, and Vol. XXX., Part 3, pp. 118-122.

IMPORTS OF CRYOLITE INTO THE UNITED STATES.

Year.	Long Tons.	Value.	Year.	Long Tons.	Value.	Year.	Long Tons.	Value.	Year.	Long Tons.	Value.
1887..	10,328	$138,068	1890..	7,129	$95,405	1893..	9,574	$126,688	1896..	3,009	$40,056
1888..	7,388	98,830	1891..	8,298	76,350	1894..	10,684	142,494	1897..	10,115	135,114
1889..	8,603	115,158	1892..	7,241	96,932	1895..	9,425	125,368	1898..	6,201	88,501

IV. ALUMINUM.

The production of aluminum in the United States in 1898 showed a large increase, consumption having expanded in all the old lines, while many new uses were developed. The more part of the increase in consumption was due to the employment of aluminum for electrical conductors and in the utilization of the metal to replace zinc and brass for many purposes.

PRODUCTION, IMPORTS, AND CONSUMPTION OF ALUMINUM IN THE UNITED STATES.

Year.	Production.			Imports.		Consumption.	
	Pounds.	Value.	Per Lb.	Pounds.	Value.	Pounds.	Value.
1893	312,000	$202,800	$0.65	7,816	$4,683	319,816	$207,483
1894	817,600	490,560	0.60	5,303	2,524	822,903	493,084
1895	900,000	495,000	0.55	25,294	7,814	925,294	502,814
1896	1,300,000	520,000	0.40	698	591	1,300,698	520,591
1897	4,000,000	1,400,000	0.35	*Nil.*	*Nil.*	(a)	(a)
1898	5,200,000	1,690,000	0.325	60	30	(a)	(a)

(a) The exports from the United States not being reported it is no longer possible to compute the domestic consumption.

These is prospect that the domestic production of aluminum will show another large increase in 1898. Arrangements are now being made to double the capacity of the plant at Niagara Falls, with the intention of putting the extension in operation by May, and in the meanwhile efforts are being made to increase the output with the existing plant and the installation of some additional machinery, which has been on hand since the company discontinued the manufacture of aluminum with steam power at New Kensington, Pa.

The statistics of aluminum production in Europe are not authoritative, several of the important companies being unwilling to make public their figures. The *Metalgesellschaft*, of Frankfurt-am-Main, gives the following statistics for Europe in its last annual report, to which we have added our own figures for the United States, and those of C. Le Neve Foster for England in 1897.

ALUMINUM: WORLD'S PRODUCTION AND COMMERCE; IN KILOGRAMS.

	Germany.	Switzerland. (a)		England.	France.			United States.(b)		Total Production.
	Imports.	Production.	Exports.	Production.	Production.	Imports.	Exports.	Production.	Imports.	
1889..	....	....	....	34,500	14,840	....	762	21,531	453	70,871
1890..	....	40,538	137,200	70,000	37,000	109	187	97,797	930	165,335
1891..	....	168,669	201,900	52,500	36,000	593	130	76,238	1,779	233,407
1892..	....	237,395	277,900	41,000	75,000	2,007	42,287	133,811	20	487,206
1893..	....	437,476	404,100	....	137,000	1,524	33,367	141,522	3,545	715,998
1894..	....	600,000	520,200	....	270,000	2,958	30,787	370,860	2,405	1,240,860
1895..	....	650,000	490,900	....	360,000	3,806	109,954	408,237	11,473	1,418,237
1896..	591,500	700,000	661,100	....	370,000	7,012	192,763	589,676	317	1,659,676
1897..	942,400	800,000	706,000	c310,000	470,000	6,300	224,000	1,814,388	*Nil.*	3,394,388

(a) The statistics of exports from Switzerland in 1890 and 1891 include aluminum alloys and manufactures of aluminum (b) The United States has been an exporter of aluminum for several years, but these exportations were not enumerated by the Bureau of Statistics of the Treasury Department until 1898, in which year they amounted to $239,997. (c) C. Le Neve Foster, British Mineral Statistics for 1897.

The production of aluminum in Europe increased in 1898, and not unlikely amounted to as much as 2,000 tons. The British works increased their production, as did also the works at Neuhausen, Switzerland, while the French works at La Praz and St. Michel produced about the same amount as in 1897. The electric power of the Swiss works was largely diverted to the production of calcium carbide, but with the completion of its new 10,000 h. p. water installation at Rheinfelden, the aluminum production will probably show a greater increase.

The production of aluminum continues to be very profitable. The Aluminum Industrie Actien Gesellschaft, of Neuhausen, Switzerland, reported gross profits of 1,131,141 fr. in 1897 against 1,041,992 in the previous year. The British Aluminium Co. realized a profit of £10,811 in 1897. The largest producer, the Pittsburg Reduction Co., of the United States, is a close corporation and does not publish its results, but it is known to have realized large profits in the business although these may have been consumed largely in litigation and the extension of its plant.

Price and Market Conditions.—The ruling prices for aluminum during 1898 were uniform and steady for ordinary uses of the metal. For electrical conductors, the price for aluminum in the form of bars, plates, sheet and wire was between 28c. and 29c. per lb. at the point of consumption, according to the size of the order and the point of delivery. Alfred E. Hunt, the President of the Pittsburg Reduction Co., expressed himself as follows in an article in the *Engineering and Mining Journal* of January 7, 1899.

"During the year 1898 aluminum was sold in the form of sheet at prices which made it 10% cheaper than brass sheet for a given purpose and 35% cheaper than copper sheet. The metal was not sold at as cheap a price as sheet zinc, but for many purposes where zinc has been utilized in the past special precautions have been necessary to protect the metal from corrosion by the substances with which it came in contact, and especially was this necessary as many of the zinc salts thus formed were very deleterious. Under these conditions, with the use of aluminum, the corrosion being much less in amount and the salts that are produced being non-poisonous, it is not necessary to take the precautions to protect the metal that have been necessary with zinc sheets. In this way, for many uses where zinc has been previously required, aluminum has been found more economical. To-day zinc sheets are not much cheaper than aluminum sheets. The tonnage of aluminum, which thus replaced zinc and similar metals in 1898, was fully equal to 500 tons. The amount of aluminum used for electrical conductors in 1898 was fully 650 tons."

The Pittsburg Reduction Co. had a Russian government order for 50,000 lb. of aluminum, and filled large orders for the German and French governments. It furnished 200,000 lb. of 0·25 in. aluminum wire to the Snoqualmie Falls Power Co. of Washington. In January, 1899, it secured an order for 1,540,000 lb. of sheet aluminum from Belgium.

In England also the aluminum business experienced an important development, and the works at Foyers, Larne and Milton, Staffordshire, were actively engaged during the year, although the Pittsburg Reduction Co. was a strong competitor in the business. In July a new company, Scottish Aluminium, was incorporated, in which the British Aluminium Co. is largely interested. This company bought two works in Greenock; one is being used as a foundry, and the other is being converted into rolling mills. The foundry was started at the beginning of December; the rolling mills will be ready some time this year. Aluminum cooking utensils, which were formerly imported from America under royalty to the British Aluminium Co., are now being manufactured in Great Britain.

Uses.—In the employment of aluminum in the arts the most important development in 1898 was, as previously pointed out, its adoption as an electrical conductor, for which 650 tons of 2,000 lb. were furnished by the Pittsburg Reduction Co. There was considerable discussion as to the economy of aluminum for this purpose.

Use of Aluminum as an Electrical Conductor.—Mr. A. E. Hunt made a comparison of the advantages of aluminum with those of copper and brass, taking copper at 14c. per lb. and aluminum at 29c., a cost ratio of 0·48.*

The density ratio of copper to aluminum is 3·332. Taking the electrical conductivity of copper at 100, the aluminum specially manufactured for electrical purposes has a conductivity of 63. On this basis the sectional area of an aluminum conductor of given resistance and length must be 1·6 times that of an equivalent copper conductor; but since the density ratio is 3·332, the aluminum conductor has a weight only 0·48 that of the copper of same length and resistance. As regards the cost of equivalent conductors, there is consequently little to choose. This considers only bare wires, and does not take into account the additional cost of insulation for the increased section for covered conductors. The tensile strength for a given sectional area is the same both for aluminum and copper; the equivalent electrical conductor of aluminum is thus 1·6 times stronger than copper of equal electrical resistance. It is assumed that for aerial lines the snow and ice load is practically independent of the gauge, but the advantage here is on the side of copper. As aluminum has less weight and greater strength than the equivalent copper conductor, aerial lines of aluminum can be allowed proportionately longer spans, with a corresponding diminution in the number of poles and insulators. However, if aluminum be drawn too severely through the dies, or if it be not properly annealed in the drawing operations, it is much more brittle than is represented by the above figures. Alloying has been adopted with some success as regards increase in tensile strength, but there appears to be a corresponding falling off in the conductivity. Aluminum is said to resist corrosion much better than does copper.

Kershaw criticised Hunt's statements, but without weakening them.† Zingler in reply to Kershaw pointed out that for equal conductivity aluminum has only half the weight of copper and about the same breaking strain, so that for equal conductivity the spans may be double those for copper, and with aluminum at 29c. per lb. the cost is nearly the same as for copper.‡ There is

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* *Electrical Review*, New York, Vol. XXXII., 1898, p. 118.

† *Electrical Review*, London, Sept. 2, 1898. ‡ *Idem.*, Sept. 9, 1898.

consequently in the use of aluminum a saving in carriage, labor, posts, insulators, and maintenance. The difficulty of the joints is an artificial one since there are plenty of good mechanical joints. However, some exception may be taken as to the last statement.

Edward F. Northrop tested the conductivity of some aluminum rods manufactured by the Pittsburg Reduction Co., finding the percentage to be respectively 61·45, 61·50, and 61·59 for three specimens of pure aluminum.* Aluminum alloyed with 0·75% copper had a conductivity of only 56·37%. The belief is expressed that these results are certainly accurate within 0·5% and are probably closer.

Use of Aluminum in Machinery.—Aluminum has found its way extensively into vibrating and reciprocating machines, both large and small. Here the advantage is entirely in reducing the weight. Oil cups on locomotive driving shafts which are likely to break at the shank on account of the high speed are now made of cast aluminum, with excellent results.

Use of Aluminum in Lithography.—The usefulness of aluminum as a substitute for lithographic stone, inaugurated in the United States several years ago, seems to have been established beyond doubt in 1898. At the atelier of Josef Scholz, at Mainz, Germany, thin sheets of aluminum weighing about 5 lb. are employed in place of the usual Solenhofen limestone slabs weighing 50 to 100 lb. The aluminum sheets are prepared by polishing with ground pumice stone, after which they are dried, then washed with a patented acid solution, and then dried again, after which they are ready for the imprint. Errors can be corrected easily since the ink can be washed off completely by use of a special acid, and the part thus cleaned can be prepared in the usual manner for a fresh imprint, or an aluminum sheet can be prepared to take the ink, and then the artist can draw directly upon it, giving a plate which can be printed from. If it be desired to retain the sharpness of the original, copies can be transferred to paper and these impressed in turn on other prepared aluminum sheets to be used for the actual printing.

The advantages claimed for the aluminum plates are: cost only one-third that of stones, freedom from liability to fracture under heavy or irregular pressure and greater durability in other respects, saving of space in storage (100 sheets occupying the room of a single stone) and saving in handling. The quality of the work done with them is excellent. The aluminum sheets are applicable to existing machinery, but of course much lighter presses can be used if specially designed, and if this be done rotary presses with curved aluminum sheets are employed. It has been found that to get the best results from aluminum plate printing, an ink especially adapted to the aluminum is necessary. In the United States aluminum plates and rotary presses are in use by several lithographic establishments. There are two processes of aluminum plate printing, the Mullaly and the Strecker-Scholz. There was some litigation between the owners of these processes, but at the end of 1898 a consolidation of interests was affected.

Aluminum Military Equipments.—Within two years it is expected that the

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* *Electrical World*, Dec. 3, 1898.

entire French army will be equipped with individual mess pans and one liter canteens of aluminum, and a mess pan and a stew pot for every four men. The metal furnished for these army equipments has averaged 99·5% pure and at present is costing 3 fr. per kg. It is estimated that the aluminum equipments will relieve each soldier of a kilogram of dead weight.

Aluminum in Pyrotechny.—Dr. E. Demole states that comparative experiments showed that aluminum burned as a flash light—in some cases at any rate—has advantages over magnesium. The mixture recommended is two parts of very finely powdered and dry potassium permanganate and one part of aluminum in a state of fine division.

A. Weiss suggests a mixture of 15 to 60 parts aluminum with 85 to 40 of potassium perchlorate (the latter proportion being most generally suitable for ordinary photographic purposes) as a substitute for magnesium in making flash-light photographic exposures.

Other Uses of Aluminum.—Messrs. David White & Co. built several aluminum boats in 1898, including one for the Klondike 18 ft. long, which weighed only 214 lb. A lot of plates are on order for doors in a torpedo boat, and it is expected that aluminum will be used soon for all cabin fittings, doors, bedsteads, tables, chairs, etc., in British naval construction.

Prof. Chatterton, of the Madras Engineering College, has been rather successful in his attempt to induce the natives of India to use domestic vessels made of aluminum instead of the heavier and more expensive copper and brass articles. The lightness, the color and the solidity of the cooking vessels attracted the favorable attention of all classes and a considerable number of men are now engaged in their manufacture in the native workshops.

A considerable increase took place in the use of aluminum paints, sometimes called improperly "aluminum bronze powder," which is really nothing but very finely divided aluminum. The United States Post Office Department has decided to have a number of the iron letter boxes in the larger cities painted with this paint, which, it is claimed, resists corrosion very effectively.

A new use for aluminum, developed in 1898, which promises to be of great importance, was in the preparation of metallic chromium by the process of Messrs. Goldschmidt and Vautin. For a description of this process the reader is referred to the caption "Chromium and Chrome Ore" elsewhere in this volume.

PROGRESS IN THE METALLURGY OF ALUMINUM.

The Peniakoff process reached the industrial stage at a works in Belgium, but is not considered by J. B. Kershaw* to be a promising procedure. In this process aluminum sulphide is obtained by reduction of the sulphate. In the United States experiments with an aluminum sulphide process have been made at Mount Vernon, N. Y., by Henry S. Blackmore and the Pure Aluminum and Chemical Co. This process† consists in the transformation of aluminum oxide into aluminum sulphide, by the liquefaction of the aluminum oxide by the

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* *Electrical Review*, London, Jan. 13, 1899.

† United States Patent No. 605,812 describes the latest developments.

action of heat and fused solvent fluxes, then exposing the liquefied oxide to the action of carbon bisulphide vapor.

Properties of Aluminum.—A. Ditte states that the opinion that aluminum resists the greater number of chemical reagents is incorrect.[*] When the metal is placed in dilute acid it dissolves rapidly at first, but a layer of gas is deposited on the surface which resists further action. This layer is found to adhere very firmly to the metal. By getting rid of the gas as fast as it is formed, either by working at reduced pressure or by some other means, it is found that the metal dissolves completely. In the case of a solution of common salt the following reactions take place:

$$(I.)\ Al_2+6NaCl+3H_2O=Al_2Cl_2+3Na_2O+3H_2+157{\cdot}3\text{ cals.};$$
$$(II.)\ Al_2Cl_6+3Na_2O=Al_2O_3+6NaCl+287\text{ cals.},$$

and the deposit of the layer of Al_2O_3 arrests further action. The same reactions occur when a plate of aluminum is placed in a solution of a salt of the alkaline earths. In fact, Ditte's experiments tend to show that aluminum is attacked in a more or less marked manner by all chemical agents. The protecting layer is not proof against an acid salt, or even common salt in the presence of such an acid as vinegar. This effect, however, does not act against the use of aluminum domestic utensils, since the aluminum compounds are not injurious, but the rapid wear of the metal makes it unsuitable for military purposes.

Aluminum as a Reducing Agent.—Léon Franck finds that aluminum powder contains small quantities of silica and iron, which must be carefully estimated when aluminum is used as a reducing agent.[†] About 0·8% silica and 0·34% iron are present in the purest forms of commercial aluminum powder.[‡] Phosphorus combines in various proportions with aluminum: Al_7P_3, obtained as a grayish black infusible mass; Al_5P_3 as a yellowish gray infusible mass; an unestimated compound forming little dark blue needles; and Al_3P and $2(AlP)$ as balls of metallic appearance. All these compounds are decomposed by water with evolution of PH_3. Aluminum decomposes phosphates at a high temperature, with formation of phosphorus. By the addition of some silicic acid the decomposition is almost quantitative and according to the equations:

$$(I.)\ 6NaPO_3+10Al+3SiO_2=3Na_2SiO_3+5Al_2O_3+6P,$$
$$(II.)\ 3Ca(PO_3)_2+10Al+3SiO_2=3CaSiO_3+5Al_2O_3+6P.$$

Aluminum decomposes CO_2, CO, and carbonates respectively as follows:

$$(III.)\ 3CO_2+4Al=2Al_2O_3+3C.\quad 3CO+2Al=Al_2O_3+3C.$$
$$(IV.)\ Na_2CO_3+2Al=Al_2O_3+C+NO_2,\ CaCO_3+2Al=Al_2O_3+C+Ca.$$

During all these reactions some aluminum carbide is formed. Aluminum is both a good and a sure reducing agent; for it decomposes metallic oxides, forming the metal and Al_2O_3; it decomposes sulphates forming sulphur and sulphides, and chlorides forming the metal.

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[*] *Comptes Rendus*, CXVII., No. 23, Dec. 5, 1898.

[†] *Stahl und Eisen*, Vol. XVIII., 1898, p. 410.

[‡] This is incorrect. The Pittsburg Reduction Co. is furnishing European consumers with powder assaying 99·60 to 99·70% Al, 0·35 to 0·25% Si, 0·05 to 0·01% Fe. American metal is used almost exclusively for this purpose on account of its superior purity.—EDITOR.

V. ALUM AND ALUMINUM SULPHATE.

Alum, using the word generally, comes into commerce both as a product derived from the simple lixiviation of alum-bearing earth and as a product prepared artificially by chemical agency, the latter being the more important.

ALUM, ARTIFICIAL.—The term "alum" is used commonly to designate not only the true crystallized alum represented by the formula $Al_2(SO_4)_3+18H_2O$, but also aluminum sulphate, which is now the more important of the two.

UNITED STATES PRODUCTION AND IMPORTS OF ALUM.

Year.	Production.								Imports. (a)		
	Alum.			Aluminum Sulphate.			Total Reckoned as Alum. Sh. Tons.	Total Value.	Short Tons.	Value.	Per Ton.
	Short Tons.	Value.	Per Ton.	Short Tons.	Value.	Per Ton.					
1896..	14,090	$442,700	$30·00	42,340	$1,056,000	$25·00	77,700	$1,478,700	2,763	$86,372	$31·26
1897..	15,456	463,680	30·00	46,355	1,158,875	25·00	79,600	1,622,555	2,373	78,883	33·24
1898..	18,791	563,730	30·00	56,663	1,416,675	25·00	97,302	1,980,405	(b) 893	16,187	18·13

(a) Includes alumina, alum, alum cake, aluminum sulphate, aluminous cake, and alum in crystals or ground.
(b) There was also imported in 1898, 1,205 short tons ($76,884) of alumina hydrate, or refined bauxite.

The statistics of the production of alum and aluminum sulphate given in the above table are computed from the consumption of bauxite and cryolite in the United States, and the production of aluminum, it being assumed that what is not used for aluminum is used for making the sulphates. The yield of both American and imported bauxite is well known, and consequently this is an accurate method of determining the production in so far as it is expressed in terms of crystallized alum. The division into crystallized alum and aluminum sulphate is estimated, and consequently is only approximate. However, since it is apt to be misleading to report the entire production as crystallized alum, of which really only a comparatively small amount is made, we began last year to report our statistics of this industry in the present form. Any apparent discrepancy is thus accounted for.

Market Conditions in 1898.—Crystallized alum was even less in demand in 1898 than in 1897, dyers having replaced it still further by other chemicals, and chiefly by aluminum sulphate, which is considerably cheaper, and consequently has now come to be employed for almost all purposes for which crystallized alum is not indispensable. The use of the sulphate has now become so common indeed that when the word "alum" is used the sulphate is understood, and not the true crystallized alum to which the term really belongs.

ALUM, NATURAL.—Numerous deposits of alum are known to exist in New Mexico and Arizona, and perhaps elsewhere in the Southwest, but owing to their remoteness from markets they have not been exploited to any considerable extent up to the present time. According to the last report of the governor of New Mexico, published by the Secretary of the Interior in October, 1898, a large deposit of alum has been opened on the Gila River, which is reported to be of great commercial value, and is now being worked to some extent.

THE MANUFACTURE OF ALUMINUM SULPHATE.

BY JOEL G. CLEMMER.

IN the manufacture of aluminum sulphate or "paper-maker's alum," a product practically free from silica, iron, and insoluble matter should be aimed at. The insoluble matter usually consists of alumina not decomposed in the process of manufacture, either through the carelessness of the maker, or because of overdried alumina. The silica can readily be separated from the alumina when the latter is still in combination with soda, as aluminate of soda, by an inexpensive process. Alumina produced by the "soda process" is of course free from iron, and consequently no trouble is experienced on this account with alumina made thereby.

As previously stated, the insoluble matter in alum consists usually of undissolved alumina. A competent alum-maker will do much to overcome this trouble, but there are also four other points under the control of the alumina producer which will go far to help him out. These are: (1) Instead of washing or otherwise entirely freeing the alumina from soda, it is best when it contains about 20%, of the latter. (2) Alumina, instead of being produced as fine as flour, should be sandy, that is to say, when rubbed between the fingers sandy alumina will not form a paste, like floury alumina, in the process of dissolving; hence will dissolve more quickly and perfectly in the acid than the latter. (3) Alumina, instead of being "overdried," should contain from 60 to 62% Al_2O_3. (4) If the alumina contains small particles of wood, which may happen if it be produced among a lot of wood-work, or if it be lumpy, it must be sifted before use.

The sulphuric acid, which is now almost universally made from pyrites, should be free from arsenic, iron, etc.; otherwise these would become part of the alum, which would be, to say the least, very undesirable. The acid is best prepared by the usual method of precipitating the impurities with sulphuretted hydrogen.

It is often the case that either the alumina or sulphuric acid, or both, is a little "off color," which will of course show itself, more or less, in the alum; and the consumer will naturally look upon such a product with a suspicion that it is not up to the standard in quality. The use of a small quantity of potassium chlorate in each charge of alum will prevent this off-color result, and give the alum a uniform snow-white color. The quantity to use is about one pint of saturated solution of potassium chlorate to each 100 lb. of alumina hydrate used. The alum should, of course, be made "porous," so that it will dissolve more quickly. The addition of about 5½ lb. of sodium bicarbonate to each 100 lb. aluminum hydrate will give the desired result.

Assuming that the alum manufacturer also produces the alumina and the sulphuric acid; and assuming that his plant has a minimum capacity of about 17 short tons of aluminum sulphate per day, the cost of producing 33,000 lb. ready for shipment will be, in actual practice, as follows: Labor, $13.50; packages, $44; alumina, 9,000 lb. assaying 62% Al_2O_3, equal to 5,580 lb.@3c., $167.40; sulphuric acid, 48 to 49° B., 30,000 lb.@$4 per 2,000 lb., $60; sodium bicarbonate, 500 lb.@1½c., $7.50; fuel, repairs, depreciation, etc., $16.15; total, $308.55, or $1.87 per ton of 2,000 lb.

Two copper kettles, each 6 ft. in diameter and 5 ft. deep, are sufficient to produce the above quantity. These kettles should each have an outlet pipe at the bottom, and have an outside steam jacket for supplying additional heat to facilitate dissolving the alumina. The kettles may be set and fastened to a timber platform elevated 7 ft. above the floor, and back of the pan. Each kettle should be supplied with a chemical tile pan built in the floor, the top of it flush with the floor, directly in front of the outlet of the kettle.

It will be noticed that the outlets of the kettle do not come directly over any part of the tile-pans. In practice a movable trough is used to convey the hot alum solution to the pans from the kettle openings, because the openings have to be closed immediately after the charge is let down; otherwise the alum will solidify and time will be lost in cleaning the parts to make a tight joint for the next charge, while to put the pans any closer to the kettle openings would make it very inconvenient to close the latter.

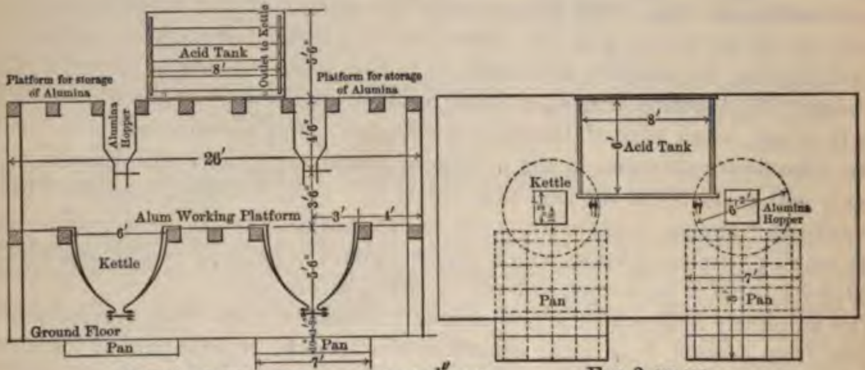

FIG. 1. ELEVATION Scale $\frac{1''}{12}$=1 ft. FIG. 2. PLAN

PLANT FOR MANUFACTURE OF ALUMINUM SULPHATE

The proper dimensions for the pans for the above size kettles are 8 ft. × 7 ft. × 10 in. deep. The alumina hopper, or receptacle, should be of sufficient capacity for only one charge of alum, and should be situated in the floor above the kettle. The acid tank should be situated on the same floor as the alumina hoppers, and should be supplied with a lead pipe steam coil for keeping the acid hot; and also should be equipped with pipes leading into each kettle. The first floor of the plant is preferably of stone, as there are no wooden chips from such a floor to get into the alum as "insoluble matter."

The proper charge of acid and alum for the above size kettles and pans is about 1,670 lb. (or 18·2 cu. ft.) of 48 to 49° B. sulphuric acid, and 500 lb. alumina hydrate, 60 to 62% Al_2O_3. One man, with a copper ladle, will properly work this charge in one hour on an average. From 25 to 28 lb. of sodium bicarbonate is added to make it porous, after the alumina is dissolved and the charge ready to be let down into the pan. Before the charge is let down the pan should be greased with lard oil to prevent the alum adhering to it. Lard oil is used because it does not discolor the alum. Before the alum entirely solidifies it is marked or cut into blocks with a copper spud of convenient size for handling. While the alum is solidifying the maker begins another charge, and in this manner, with two kettles in operation, 17 tons per day can readily be produced.

AMMONIA AND AMMONIUM SULPHATE.

Ammonium sulphate is produced most extensively in Germany and the United Kingdom, especially the latter, where it is recovered by gas works, iron works, the Scotch shale works and producer gas and coke works, the total production having been 198,280 long tons in 1897, against 190,909 in 1896 and 179,651 in 1895, divided as follows:

	1895	1896.	1897.		1895.	1896.	1897.
Gas works........	119,645	127,498	132,724	Shale works......	38,335	37,822	37,153
Iron works.......	14,588	16,511	17,779	Coke works.......	7,083	9,078	10,694

In the United States ammonium sulphate is recovered by various chemical works which buy the ammoniacal liquor from local makers of illuminating gas, and is also made by the by-product coke oven plants. There are no statistics available as to the production by the former. The by-product coke oven plants made 3,589 short tons in 1898 against 3,111 in 1897. They also recovered 16,021 short tons of coal tar in 1898 against 13;304 in 1897. The average value per 100 lb. of gas liquor ammonium sulphate, basis 25%, in New York in 1897 and 1898 was as follows:

Year.	Jan.	Feb.	Mch.	April.	May.	June.	July.	Aug.	Sept.	Oct.	Nov.	Dec.	Average.
1897....................	$2·275	$2·219	$2·275	$2·218	$2·150	$2·125	$2·115	$2·106	$2·169	$2·190	$2·294	$2·275	$2·20
1898....................	2·365	2·618	2·457	2·415	2·394	2·481	2·445	2·488	2.575	2·535	2·606	2·630	2·50

Recovery of Ammonia and Ammonium Sulphate from Gas Liquor.

By Hans A. Frasch.

Sources of Ammonia.—The principal source from which the United States derives its ammonia supply is the condensation product (gas liquor) obtained in making coal gas and coke by the distillation of coal. Small amounts are obtained by the distillation of bone and other animal substances, as in the manufacture of bone black, and it is still produced from animal excrements at New Orleans. Within the last few years the coke makers of the United States have begun to adopt by-product ovens, and a considerable quantity of ammonia is now being supplied from that source; though by far the larger amount is still obtained from the gas works.

Disadvantages in Practice.—Unlike many other industries, that of gas making is not yet compelled to depend upon the utilization of by-products for the sake of economy to overcome increasing competition, and the ammonia industry experiences on that account considerable inconveniences and some needless expense. The means for washing the gas in many of our gas works are arranged with entire disregard of economy as to the strength, purity, proper storage and convenient handling of ammoniacal liquor. It is to be hoped that the growing competition of electricity will induce the manufacturers of coal gas to work toward complete recovery of the by-products at minimum cost to the consumers. In most instances the matter of transportation figures prominently in the cost of ammonia, and in only a few cities are the ammonia works located at the site of the gas works. The expense connected with the transportation is a tax upon both the producer and user of gas liquor; the gas manufacturer should therefore aim to produce a liquor of the highest possible strength. A tank car of 5,000 gal. of 1% liquor represents about 400 lb. available ammonia; of 5% liquor the same quantity contains 2,000 lb. ammonia. The cost of transportation from gas works to ammonia factory being $6 per car, the freight charge on the available ammonia is $1·50 per 100 lb. for the 1% liquor and only 30c. per 100 lb. for the 5% liquor. Assuming a value of 6c. per lb. of ammonia at the gas works, the freight on 1% liquor represents in this case one quarter of the total value of the ammonia. Adding to this the cost of handling, storing, heating and distilling five times the quantity of water, which 1% liquor requires above the 5% material, the difference in cost of manufacture of ammonia in the two instances may reach 40 to 50%.

Recovery of Ammoniacal Liquor in Gas Works.—The intimate relation existing between the recovery of crude ammoniacal liquor (gas water) and ammonium products, especially as to the cost of the latter, renders the scrubber of the gas works really a part of the ammonia plant. E. H. Harman,* in discussing washers and scrubbers of coal gas, remarks that both operations could be better performed if complete control were obtained of the temperature of the gas at all portions of the apparatus, as a temperature below 60° F. may result in serious reduction of the heavier hydrocarbons.

Scrubbers.—Mr. Harman describes different types of scrubbers, of which the Mann & Walker tower scrubber is perhaps the most familiar in England. It consists of a cylindrical tower of about three to five diameters in height, having a brush wheel in the upper portion, which receives the water supply from revolving distributors designed to work eccentrically, so that they travel slowly in traversing the outer circumference of the vessel but rapidly when nearing the center. The filling material is either coke or boards on edge. The channeling of the gas and water through coke makes boards preferable. The material is supported on a series of cast-iron grids, leaving each tier separate for the dispersion and diffusion of the gas. William Mann strongly recommends passing the gas through the scrubber under pressure.

When a series of towers is used, weak ammoniacal liquor is pumped into all, except the last or finishing one, in which clean water is employed. The ammoniacal liquor thus becomes a useful purifying agent, and is also itself enhanced in value. One advantage of using the liquor is that the hydrocarbons are not deposited as when water only is used. Properly packed tower scrubbers require cleaning about once in eight or ten years. They may be

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* Paper read at the annual meeting, 1898, of the Institute of Gas Engineers. Printed in *Chemical Trade Journal*, July 9-23, 1898.

partially cleaned at intervals by steaming, care being taken to commence the steaming process at the top.

Livesey gives the following relative area of surfaces in tower packing, in square feet of surface to each cubic foot of material: Coke, 8·5; drain pipes (2 in. diameter), 21; drain pipes (3 in. diameter), 17; boards on edge, 31.

Mr. Harman, in the paper already referred to, says: "In a tower scrubber filled with grids of $\frac{3}{8} \times 9$ in. boards, with $\frac{3}{8}$-in. distance pieces, a very large surface is exposed for wetting, and consequent frictional surface action on the gas. In a tower designed for passing 3,000,000 cu. ft. of gas per 24 hours, say 15 ft. diameter by 60 ft. high, having six tiers of grids, the surface equals 268,000 sq. ft. These grids occupy 40% of the cubical capacity of the tower. In the tower there is approximately 10,602 cu. ft. less 4,356 cu. ft. filled with material, leaving in the interstices 6,246 cu. ft. The inside of the scrubber, together with the surface area of the grids, presents 270,820 sq. ft. The gas passes at 34 ft. per second, and will be in the interstices of the material such a time as it takes to fill up the 6,246 cu. ft. divided by 34, which equals nearly three minutes. Multiplying the surface, 268,000 sq. ft., by the time, three minutes, 804,000 sq. ft. is obtained, which is the value of contact with the gas passing. The result may be written: Total surface ÷ volume of gas; which expresses the amount of contact surface available for each volume of gas.

"The capacity of any washing machine may be gauged from the rule stated above, independent of design. Again, gas may readily be reduced to bubbles of a very small diameter, and by this means the contact surface can be obtained equal to any other. Suppose the gas is divided into bubbles of $\frac{1}{4}$ in. diameter. Then for each foot of gas we obtain about 288 sq. ft. of surface area. As gas rises through 1 ft. of liquor in about one second, for this to equal the capacity of the tower scrubber described, it would have to pass through a column of liquid 27 ft. deep. The expenditure of power to effect this would be very much in excess of the force required to deliver 13 gal. of water to the top of the tower.

"The late Dr. Wyatt recommended 100 cu. ft. per ton per diem of internal capacity of the vessels, with a gas contact of from 15 to 27 minutes for maximum and average makes, and stated that 12·5 to 15% of this volume could be utilized by introducing a washer into the base of the scrubber. The volume of gas contained in the scrubbers at one time amounts to about 1% of the maximum make per diem. The horizontal net area of all the scrubbers recommended by Dr. Wyatt is equal to two superficial feet per ton per diem on the maximum make."

For the following data in regard to the water distributors and washers I am indebted to the same authority:

Water Distributors.—It is difficult to insure thorough distribution over the whole area of a large scrubber, since although accomplished at the first tier the liquid collects and drains in channels through the remainder. Mr. Harman has tried steam jets in each tier with good effect as to saturating the material all over, but the increased temperature of the gas prohibits its use. The

Barker mill arrangement is perhaps one of the best forms. Gurney's jet consists of a sliding tube with a nozzle, having a fine hole $\frac{1}{64}$ in. in diameter. The liquid is forced through this at considerable pressure, causing it to impinge against a circular glass disk. Green's water distributor, Goldsmith's, and many others, might be cited.

Washers are of numerous designs. The gas passes through a series of perforated plates, brushes, bundles of grids, and many other devices placed in chambers, and being constantly wetted. The surface of these, and the contact, are enormously increased in many cases by constant revolution, and such types possess advantages, inasmuch as the liquor is retained without pumping, being gradually run off as the strength increases. The gas leaves behind the larger portion of its ammonia in the chamber. The liquor, being retained, is consequently worked up to a far higher degree of saturation. An analysis of the liquor from one of these washers shows the action of the gas clearly. In the first chamber a little fixed ammonia is obtained, while free ammonia, or that held in solution with such weak acids as carbon dioxide, will seem to have displaced the sulphuretted hydrogen, it being found there in the proportion of 70%, compared with the sulphuretted hydrogen similarly held. In the latter chambers the sulphuretted hydrogen seems to enter more into solution. In the last chamber it is side by side with the carbon dioxide, and is found up to 50%. The acid equivalents throughout the series become gradually less. The percentages of carbonic acid and sulphuretted hydrogen in the liquor vary principally according to the strength of the liquor, and partly to the conditions of its formation. Generally between 50 to 70 equivalents of alkaline ammonia are combined with 50 to 70 equivalents of carbon dioxide and sulphuretted hydrogen, the sulphuretted hydrogen of which forms nearly 20%. The strong liquors contain less than the weaker ones. The operations of washing remove two or three grains of carbon bisulphide per 100 cu. ft. and the illuminating power is slightly diminished.

The Livesey washer, which is probably the most extensively adopted, consists of a rectangular cast-iron box, the upper part being formed into an inlet chamber, together with one or two outlets. To the lower flanges of these inlet chambers is fastened a series of wrought-iron tubes, having holes $\frac{1}{16}$ in. in diameter. The interior of these tubes is in communication at the ends with the outlet chambers, while the spaces between them at the ends are securely closed up. The gas has free access from the inlet chamber direct to these intermediate spaces, passing down them and depressing the liquor until the gas escapes through the inclined portion of the perforated plate to the first space, which is filled with liquor, through which it bubbles until it comes in contact with the horizontal part of the perforated plate, carrying some of the liquor with it to the upper surface of the plate. The bubbles of gas pass through this liquor into the tube space and convert the surface into light foam, which flows along with the gas into the outer chambers. The liquor drains from the washer by means of overflow pipes, by which the level can be adjusted as required. The inlet for the liquor supply is placed at the lower part of the washer. Glass windows are fixed in the sides at opposite ends of the tubes, so that the bubbling and washing of the gas in the tubes may be clearly observed. Special provision is made in the lowest part of the chamber for the collection of the tar, which is separated, and falls to the bottom, being drawn off from the level independently of the liquor at regular periods. The apparatus possesses the advantage of being automatic, and requiring only a trifling amount of attention. By the minute subdivision of the gas, which is brought twice into contact with the liquor, the entire removal of all the tar, and the greater part of the ammonia, is insured. In some works the ammoniacal liquor from the tower scrubbers flows by gravitation *via* a tar and liquor separator direct into the washer, which is fixed at a desirable level for the purpose. The number of perforations through which the gas has to pass per foot of trough is 3,264, thus dividing the gas into many thousands of streams. Where more than one of these washers is used they are placed at different levels, to admit of the liquor gravitating from the last one to the first.

Other washers in use in Great Britain are the Hunt, Marshall, Anderson, Young, Reid, Holme, Kirkham, Whimster, Laycock & Clapham, Cathel, and Walker. The limitations of this paper exclude descriptions of these devices, which pertain especially to the British practice, and for their details reference should be made to the original paper above quoted.

Concentration of Gas Liquor.—In Hill's process the ammoniacal liquor is passed through a kind of still, so regulated that about 50% of the sulphuretted hydrogen and carbon dioxide present are liberated. The purified liquor, as it is termed, is then run back into washers, and is ready to take up another quantity of these impurities.

Ammoniacal liquor used in washers, unless purified, must not be allowed to become too heavily charged with carbon dioxide, or it will displace the sulphuretted hydrogen. This is one of the reasons why the concentration of the liquor in the washers must be kept within certain limits; there is no reason, however, for keeping gas liquor as low in strength as 1%. On the European continent the strength of the liquor varies from 0·3 to 1·8% available ammonia and 0·9 to 5·2% fixed ammonium. The weakest liquor is obtained at Zurich, bearing 0·34% available ammonia and 0·95% fixed; at Bonn the liquor contains 1·81% free and 5·18% fixed ammonium. The discrepancy in this country is still greater, with a strong tendency toward the weaker product. Preliminary removal of the greater part of the carbon dioxide, or sulphuretted hydrogen, before the gas enters the concentrating washer, would permit higher concentration of the gas liquor.

In the early history of the coal gas industry water acidulated with sulphuric acid was used to purify gas, but the practice has become obsolete on account of the affinity of sulphuric acid for some of the hydrocarbons. It is probable that a return to that method in some improved form will be made sooner or later. The technical drawbacks which the conversion of ammonium sulphate by means of lime bring about are against the recovery of the ammonia in the form of sulphate in gas liquor. Periodical additions of small amounts of common salt to the wash water or gas liquor might result in more favorable results, and allow a higher concentration of the wash waters. The sodium chloride, in reaction with the ammonium carbonate and bicarbonate, would form sodium carbonate and sodium bicarbonate respectively, and ammonium chloride; thus the detrimental action of the carbon dioxide could be overcome, while the sulphuretted hydrogen would be absorbed more readily. This process could only be carried out at lower temperatures, not exceeding 110° F. A change in that direction would no doubt be greatly welcomed by the ammonia manufacturers.

In late years concentrators have been introduced in some of the gas works, especially in works located at a distance from the ammonia factory. This may be advantageous for the consumer, but hardly for the producer of ammoniacal liquor. Small gas works, whose output would otherwise go to waste, may profit by erecting concentrators, but often the apparatus used by these works is constructed so cheaply and primitively that it gives very low yields. There are concentrators which operate without lime and only recover the ammonia liberated by heat; such a method, of course, is wasteful. Concentrators should not be necessary, and the problem should be solved in the gas washer.

Storage of Gas Liquor.—The storage of ammoniacal liquor at the gas works is also of importance, and is often sorely neglected. Formerly, when ammonia was sold by the gas works upon a basis of "coal carbonized," the producer did not care how much liquor went to waste by leakage or evaporation. The prices then paid were so low that even the consumer could afford to overlook these shortcomings; he would yet derive satisfactory profits at the price which ammonia commanded. This is changed now, the gas liquor being sold on the basis of delivery of actual ammonia available, and the obligation to secure and deliver to the consumer all the ammonia that can be obtained rests upon the gas works. Yet there are some consumers purchasing and paying for gas liquor delivered into the storage tank of the gas company, standing a risk of leakage, overflow, etc., over which they have no control.

Almost invariably among gas works, all the different ammoniacal liquors, whether from hydraulic mains, condensers, cooling mains, washers, scrubbers, etc., are conveyed into one common storage tank, although this is hardly a scientific way of dealing with them. In many instances no separator exists, and tar and ammoniacal liquor are stored together in one cistern, the tar being sold to one party, the liquor to another. This practice is very disadvantageous for the ammonia manufacturer; and, if not attended by serious loss of ammoniacal liquor, is a great inconvenience, often leading to otherwise unnecessary expense. During the winter season, when the production of gas and by-products is at its highest, the coal tar consumption is at its lowest; so that, between the accumulation of coal tar in the storage tanks, and increased output of ammoniacal liquor, the storage capacity for the latter is insufficient, and the ammonia producer is compelled, often at considerable expense, to provide special storage. Besides this, it is not well to allow the coal tar to be in contact with the liquor, since in course of time the latter becomes more and more charged with organic compounds. As shown in the formation of asphaltum, reaction takes place between the sulphur of the liquor and the hydrocarbons of the tar, though how far, if at all, such a reaction affects the ammonia itself has never been determined.

Well-equipped gas works should be provided with tar separators and separate storage tanks for tar and ammoniacal liquor. As to the construction of the storage tanks, they are generally cisterns excavated and bricked up in the ground. While tanks above ground are certainly safer as far as leakage is concerned, and easier to repair, the underground cistern has the advantage of natural drainage and more uniform temperature, and, if well built and cemented to exclude all possibility of leakage, is preferable. Cheap and carelessly constructed cisterns are in the long run the most expensive on account of leakage.

Amount of Ammonia Obtainable from Coal.—Coal contains from 1 to 2% of nitrogen, of which about 14% is obtained as ammonia. Cannel coal generally contains less nitrogen than common coal. The nitrogen in coal may be reckoned to be distributed in course of distillation as follows:* As ammonia, 14·50%; as cyanogen, 1·56%; free in gas and combined in tar, 35·26%; in coke, 48·68%. According to these figures a ton of coal should yield 5·7 lb. of ammonia.

Various methods have been suggested to increase the output of ammonia. W. I. Cooper treats the coal with about 2·5% of lime before distillation, and claims to obtain 30 to 35 lb. of ammonium sulphate per ton of coal. R. Tervet conducts hydrogen over the coal during distillation, and states that he has obtained as much as 120 lb. of ammonium sulphate per ton of coal. Steam also is suggested as a means of increasing the yield of ammonia.

Determination of the Value of Gas Liquor.—Until recently the Baumé hydrometer has been used. Specific gravity, however, is very unreliable as an index in determining the amount of ammonia contained in gas liquor, and is practically worthless when free ammonia is present. The latter is hardly ever the case, but the proportions of the fixed ammonium vary, and, though the specific gravity of the liquor may be the same, the amount of available ammonia fluctuates. Besides, the presence of bodies other than ammonia in more or less quantities affects this test. The old English method of direct neutralization of the liquor with sulphuric acid, and determination of the number of ounces acid required for one gallon of the liquor is still used in a few works. By far the best mode is a chemical determination of the actual amount of ammonia available; in fact, this method is now generally adopted. The prices paid for ammoniacal liquor range from 4 to 6c. per lb. actual ammonia, according to the strength of the liquor. Contracts generally are based, by sliding scale, on the market price of ammonium sulphate.

Ammonium Products and their Relative Value.—The manufacture of ammonium products from ammoniacal liquor is now accomplished wholly by distillation, ammonium sulphate being the primary product. From the sulphate the chloride, carbonate, and aqua ammonia are obtained. The development of the refrigerating industry within the last decade, and the consequently increased demand for anhydrous and aqua ammonia have created a tendency to produce high grade aqua ammonia direct from gas liquor, and numerous ammonia fac-

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* *Journal Chemical Society*, XLIII., p. 105.

tories are now producing a fair quality of aqua ammonia by the direct method of distillation. The manufacture of ammonia from the sulphate offers a number of difficulties. The formation of gypsum in course of the distillation with lime results in low yields, the process of conversion is slow, and blockades of pipe lines and other parts of the apparatus are frequent. Besides these difficulties the value of the sulphuric acid is entirely lost, no matter if the ammonia sulphate is used for the production of aqua ammonia or for fertilizer. For the latter purpose the use of sulphate to-day seems to be from mere force of habit, and there is apparently no reason why aqua ammonia should not be used in place of sulphate by the fertilizer manufacturers. More concentrated goods would be obtained if aqua were used in place of sulphate, and the time may not be far distant when it will displace the sulphate in the manufacture of ammoniated fertilizer. Assuming a value of 10c. per lb. for anhydrous ammonia, the relative value of material and available ammonia represented in 100 lb. are: (1) In sulphate: 75 lb. sulphuric acid at 1c., $0.75; 25 lb. ammonia at 10c., $2.50; material value (25 lb. available ammonia), $3.25. (2) In aqua ammonia of 26° B.: 28·6 lb. at 10c., $2.86; 71·4 lb. water, *nil;* material value (28·6 lb. available ammonia), $2.86. This shows a relative cost of material for each pound of available ammonia of 13c., if derived from sulphate, and 10c. if obtained from water. As to the cost of production, the amount of lime and heat required for the primary distillation of the gas liquor is the same in both instances. For the absorption, 75% of 66° B., or an equivalent amount of weak sulphuric acid, is required to make sulphate, and about 71% of water to obtain 26° B. crude aqua. For many technical purposes the latter can be used without further purification, while the sulphate requires draining and drying.

To make aqua ammonia from sulphate sufficient lime has to be used to convert the total amount of sulphuric acid represented in the sulphate and about one-third more; the solution of sulphate has to be greatly diluted, so that at least two or three times the quantity of liquid has to be treated and taken care of than is required for the distillation of 26° B. crude aqua ammonia.

The production of refined aqua from the crude primary distillate requires very little lime, and the time occupied for distillation is not half of what the conversion of sulphate consumes. The direct production of aqua from gas liquor, however, requires more perfect and expensive apparatus and more attention in operating it; though, even for sulphate, it is advisable to employ modern apparatus.

Ammonia Distillation.—The modern apparatus used for the distillation of gas liquor involves the principle of the alcohol still. The ordinary cylinder (or boiler) still is obsolete and in use only in rare cases, where cheapness of construction retards progress and no doubt lessens profit.

In all modern apparatus the vapors from the distillation are concentrated by dephlegmation, that is, the more volatile part (ammonia) is separated from the less volatile (water and impurities) by partial condensation in a system of reflux coolers. These may be arranged for indirect cooling in coils or pipes, or direct cooling, that is to say, in contact with the cooling medium; or both may be combined. The latter mode is most in favor and gives the best results; part of the apparatus serves not only as a direct cooler but also takes the place of a scrubber and additional distilling apparatus.

The indirect cooling devices consist of a series of pipes immersed in water or having a stream of water running over them; they may have a vertical or horizontal position. If the latter case they should incline slightly toward the inlet. In either case the vapors pass in a reverse direction to that in the ordinary condenser—from the bottom to the top, so that the descending condensed liquid flows in opposite direction to the ascending vapor. The solubility of ammonia diminishing with the increase of temperature, the condensed water, which in the upper and cooler part of the apparatus has absorbed considerable ammonia, meets in its downward flow the incoming hot vapor and thereby gives off again the ammonia in proportion to its solubility at the existing temperature.

Condensers.—The cooling and condensation of the vapors are affected by water or, as more economical, by gas liquor. In the latter case the condenser also serves the purpose of a primary heater for the gas liquor as it flows to the distilling apparatus. The use of gas liquor for condensation purposes has, however, several drawbacks; as a rule the amount of liquor is not sufficient to assure perfect condensation, which is an absolute necessity if a good product of distillation is desired, so that a second condenser is required. Another and more serious objection is the corrosive effect crude gas liquor has upon iron at high temperatures. The action appears to be the strongest at the point of contact of wrought and cast iron, and condensers in which gas liquor is used as the cooling medium do not last long and require frequent repair. When distillation is conducted by exhaust steam, and fresh water is abundant, it is doubtful if the use of gas liquor in the condenser is economical. To obtain the best effect the temperature in the condenser should be kept lowest at the top and gradually increase toward the inlet. For this reason the open air condenser gives the best results. This apparatus consists of a series of horizontal pipes connected by a manifold joint with the main vapor pipe, and forming by means of return bends a coil of any desired height. A perforated pipe (the perforations facing upward), placed over each row of pipe coils, of which there are four to six in one system, serves to distribute a stream of water evenly over the whole length of the top pipes of the coil, so that the water drops over the next following vapor pipes, becoming warmer at each successive pipe line, leaving the bottom decidedly warm. This style of condensers requires the least amount of water, but has, on the other hand, the disadvantage of requiring constant regulation of the water flow and frequent cleaning. The perforations soon become clogged and the vapor pipes accumulate deposits on the outside which greatly diminish the cooling effect of the water. The best results are no doubt given by the horizontal cooler shown in Fig. 1. This consists of a pipe coil built like the open condenser, but immersed in a water tank. The tank may be constructed of wood or iron. If provided with a suitable cover it can be used for gas liquor as well as water. The vapor enters at *a*, passes through the pipes *b*, *b*, and leaves the apparatus dehydrated at *c*. The water or gas liquor flows from *d* in a tortuous course between the shelves *e*, *e*, to *f*, where it leaves the condenser; *g*, *g*, are openings to facilitate cleaning; *h* conducts the condensed liquid back to the distilling apparatus. In constructing this style of condenser care must be taken that all the pipes incline (about 1 in. in 10 ft.) toward their inlets, to avoid accumulation of condensed water in any part of the condenser which would cause "pounding." If the condenser is properly constructed pounding will only occur if an unreasonable excess of steam is carried in the distilling apparatus, in which event the steam should be reduced at once.

The vertical condenser is constructed similarly to an upright tubular boiler, the tubes serving as vapor pipes and being surrounded by a body of cold gas liquor or water. These condensers are most in use. In connection with a direct condensing column they are satisfactory, but hardly so if the distillate is intended for aqua ammonia, when it is necessary to remove all the steam from the ammonia gas. They are more expensive to construct, and the vapors pass too fast through the condenser.

Direct condensation of the vapors in an ammonia still takes place in dephlegmators consisting of a number of compartments arranged to form a column in which the ascending vapor passes successively through the descending liquid. This column serves the double purpose of washing and concentrating the vapors, while at the same time each individual compartment constitutes a still. The apparatus is inserted between the still proper and the condenser, and forms the most important feature of the modern ammonia still; it facilitates the concentration of the ammonia gas so that 28° B. aqua can be produced from the weakest gas liquor. The arrangement of the dephlegmator varies in detail in different apparatus, but in all instances its primary purpose is the concentration of the ammoniacal vapor. In some systems it is used in connection with a still and a condenser; in others it operates without a condenser but in connection with a still; and again it fills the place of both still and condenser, so that in itself it forms a complete distilling apparatus. According to the purpose intended the column is composed of a larger or smaller number of compartments. The action of the apparatus is based upon

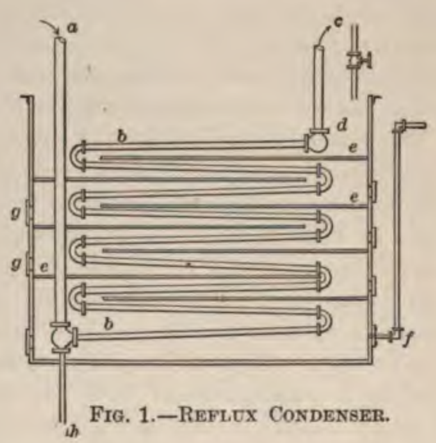

FIG. 1.—REFLUX CONDENSER.

the difference of temperature of the liquid contained in the several compartments and the solubility of ammonia at the existing temperatures. The upper section of the column, being the coolest, contains the strongest solution, while the bottom compartments are kept at boiling temperature at which the solubility of ammonia is zero; the contents of all the compartments, being at saturation point at their respective temperatures, are not capable of absorbing more ammonia, but will condense aqueous vapor rising from the preceding and hotter compartments, especially as there is a constant downward flow of the cooler liquor from the upper part of the column. In each compartment a certain amount of steam is condensed, coming from the lower sections of the column, and ammonia is liberated from the inflow of the stronger liquor of the upper and cooler sections. Thus, if the column consists of a sufficient number of compartments and the ammoniacal liquid to be distilled is charged cold into the top of the column, the steam introduced into the bottom of the column will all be condensed and the ammonia will pass on after the liquid contained in the several compartments has once been brought to saturation. The ammonia absorbed by the inflowing cold ammoniacal liquid is expelled again in the lower and hot sections of the column.

Construction of Condensing Columns.—The sections of which the column is constructed are generally cast-iron rings provided with flanges and bottoms having a vapor inlet and fluid overflow. Fig. 2 shows the arrangement; a^1, a^2, are rings from 2 to 8 ft. in diameter, according to the desired capacity of the apparatus; they are provided with bottoms *b*, *b*, which have central openings *c*, *c*, surrounded by necks *d*, *d*, over which the caps *e*, *e*, are placed so as to force the vapor arising from a^1 through the liquid contained in a^2; the cap is immersed about 1 in. in the liquid; *f*, *f*, are overflow pipes extending about 3 in. above the bottoms and projecting downward to within 1½ in. of the bottom of the preceding chamber. The liquid thus flows through pipes *f*, *f*, while the vapor passes through the openings *c*, *c*, and below the caps *e*, *e*, through the liquid contained in a^1, a^2. In some instances the apparatus is built of sheet iron, and several modifications of the arrangement are in use, but all have the same underlying principle. The bottom plates may be provided with partitions to force the liquid to travel in a circuitous flow to the outlet. In some cases the caps *e*, *e*, are flanged to a shaft and kept revolving to avoid the settling of the lime. If the dimensions of the different parts of the apparatus are correctly proportioned mechanical agitation is not necessary.

Distillation in these columns alone is applicable for the production of ammonium sulphate or chloride, but not for aqua from gas liquor, where an absolutely dry gas must be obtained. The low strength of ordinary gas liquor also limits the use of these stills. They are from 60 to 80 ft. high and 8 to 10 ft. in diameter, and have 24 to 30 compartments. The lower half or two-thirds of the column consists of compartments equipped with vapor caps and overflows, as shown in Fig. 2; the upper half or third is built up of rings between which perforated plates or shelves and disks are inserted. The gas coming from these stills has yet a temperature of 65° to 75° C. and for all purposes where a dry gas is required they should be connected with a condenser. They have the disadvantage that if ever the quantity of lime or steam is insufficient large amounts of ammonia may be lost with the waste liquor before the trouble can be remedied.

Column stills with condensers are in use mostly in the ammonia-soda industry for recovery of the ammonia from ammonium chloride. A column 10 ft. in diameter and 75 ft. high is assumed to be sufficient for a daily production of 100 tons soda ash. In this case the column is connected with a condenser to avoid dilution of the brine by aqueous vapor; the vertical condenser is generally used, the ammonium chloride liquor being the cooling medium. The temperature of the gas is reduced to about 60° C.; when cooler there is danger of deposition of ammonium carbonate in the vapor pipes. In operating these stills the ammonium chloride solution, which contains ammonium and sodium bicarbonate, is introduced at the top of the column and ammonium carbonate is first expelled before the lime is added. Unless connected with an auxiliary still, which in case of need can quickly be put in use, loss of ammonia may occur, and much more so because in case of an insufficiency of lime the presence of ammonia in the waste fluid cannot be detected by the odor, and unconverted ammonium chloride may go to waste unless the waste fluid is constantly controlled by chemical tests.

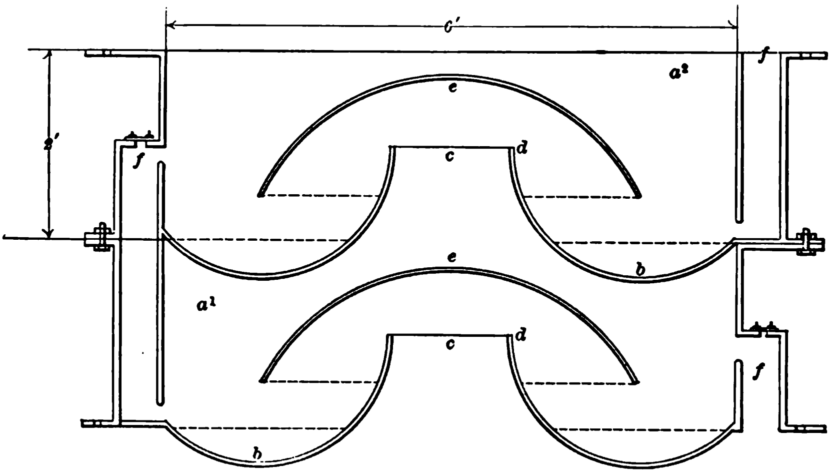

FIG. 2.—ARRANGEMENT OF DISTILLING COLUMN.

Distillation in columns provided with condensers and auxiliary stills is by far the safest and gives the best results. For the production of sulphate the condenser may be omitted, but every apparatus should be equipped with one form or another of auxiliary stills. These may be in continuous operation or periodical, in which latter case two or three stills are required. For periodical operation, the best equipped apparatus have three auxiliaries, of which one is being filled, the second is in operation and the third is in reserve. All are connected with the column by a vapor line and pipes to receive the waste fluid, and are provided with a 3 or 4-in. discharge pipe. The steam supplied to the column passes first through the auxiliaries to the vapor line and into the column; or, as is done in some cases when the column is heated by exhaust steam, separate steam lines may supply the auxiliaries. Whenever one still is full the connection with the overflow pipe of the column is closed and opened into the second still; under ordinary circumstances the contents of the first still are free from ammonia long before the second still is filled, and can be discharged and the still made ready for use before the second still is full. When emergency requires and through some cause the waste fluid runs strong, causing the second still to be filled before the first one is boiled off, the third still is put in operation. In this case the supply of ammoniacal liquor to the column is interrupted until the waste liquid is reduced to normal strength. At times when the discharge from the column is free from ammonia the waste is permitted to flow through the auxiliary without interruption and the process is continuous. In this case, however, it is well to supply the stills with mechanical agitators to prevent settling of the lime. This form of apparatus operates with the highest yields and is most in favor.

Supply of heat to the apparatus is by either direct firing or the application of steam. The latter may be under pressure in coils, or it may be injected into the still, for which purpose either live steam or exhaust steam from nearby engines may be used. Direct firing is practiced only in rare cases, though there are such systems in operation which give good results. To supply the heat by steam under pressure is rather troublesome; the lime is deposited on the coils, so that they require frequent cleaning. In most cases live steam is used in preference to exhaust steam, which is injected into the liquid through perforated disks or pipes. The condensed steam, increasing the volume of liquid in the apparatus, renders the application of direct steam impossible for intermittent stills; and even for continuous operation allowance must be made, in the construction of the apparatus, for the increased amount of liquid by the condensed water of the steam. The results obtained from different sources of heat are shown by Lunge: Heating 7,000 gal. of gas liquor by direct firing for 22 hours, 90% of the ammonia was recovered; with indirect steam in 18 hours, 92%, and with direct steam in 14 hours, 98·5%.

Grueneberg's Apparatus.—The types of apparatus in use for the distillation of ammonia from gas liquor do not differ much; although they are numerous nearly all are constructed on the same principle and only vary in the details of application. Among the direct firing stills the Grueneberg apparatus is probably the most popular for making sulphate or concentrated liquor. It operates with about 100 lb. of coal for each ton of gas liquor. Fig. 3 shows the apparatus, which consists of the still *a*, divided in two sections by the cylinder *b*; *c* is the overflow pipe from the lime vessel and washer *d* and reaches below the perforated bottom *x* of division *b*; *f*, *f* are vapor outlets covered by the caps *g*, *g*, which compel the vapor to pass through the milk of lime in *d*; *h* is a column still; the gas liquor enters the column at *i*, the lime at *k*; *l* is the overflow connected with trap *m*; the ammonia vapor leaves the apparatus at *o*. For the production of concentrated ammonia liquor the column is connected with a condenser. This apparatus can also be heated by steam, which in this case enters below the perforated bottom *x*.

Other Apparatus.—The apparatus of Mallet, Solvay, Margueritte, Hirzel and others are described explicitly in Lunge's *Coal Tar and Ammonia.* They differ but little in the mode of distillation and aim more or less toward heat economy and removal of the impurities from the ammonia vapor. As to the most economical manner of utilizing the heat in the vapor from an ammonia still, it appears unreasonable to construct elaborate heaters such as form part of almost every still. The best heater in that respect is surely the one which absorbs all the heat represented in the vapor, and by the introduction of the cold gas liquor into the top of a distilling column this effect is obtained by direct contact of cold liquor and hot vapor. The gas liquor must eventually be brought to the boiling point, and whatever heat it requires beyond what has been absorbed from the vapor must be supplied by steam or fire; no particular benefit can be gained by recovering part of the heat of the vapor by the indirect action of a condenser or heater, and diminishing correspondingly the condensation capacity of the liquor by increasing its temperature before it enters the column; while a much better effect can be obtained by direct contact condensation from the gas liquor when it is delivered cold into the column. In both instances the heat recovered is obtained from the same source. The matter assumes, however, a different aspect if the heater is applied to recover some of the heat represented in the waste liquid flowing from the

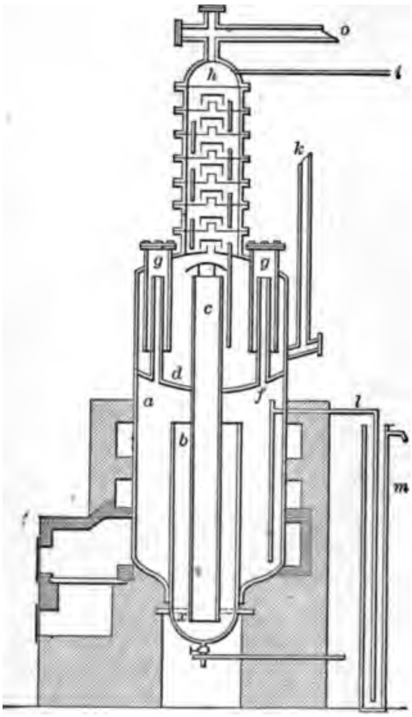

FIG. 3.—GRUENEBERG'S APPARATUS.

still; there a large amount of heat goes constantly to waste which might be recovered. None of the apparatus referred to indicates the utilization of this waste heat, and such utilization apparently is not in general practice.

Application of the Lime.—This is in nearly all instances accomplished by the introduction of milk of lime (CaO, H_2O). In rare cases the lime is applied in the form of caustic lime (CaO), with the view of utilizing the heat evolved during the slaking process, as in the Solvay process. The uncertainty of the time it takes the lime to slake makes this practice appear inadvisible if not dangerous. If the lime is of the slow-slaking sort temporary shortage of lime may result, which is followed by a momentary excessive reaction; if at that time any part of the apparatus becomes blocked explosion of the still may result, which in fact has occurred. It is better to introduce the lime into the apparatus in the form of lime milk; for this purpose the lime is slaked in the ordinary manner with about three parts of water, so that it will flow readily, and is conveyed to the apparatus by gravity or by means of membrane pumps. Care should be taken that no sand or particles of unslaked lime pass into the apparatus, and the lime milk should previously be strained. Magnesia in the lime is detrimental and retards distillation if ammonium chloride or sulphate are present in the gas liquor, for with them magnesia forms a double salt which cannot be decomposed except at high concentration. It is advisible to use only high grade lime even if at a somewhat greater cost. The lime as a rule is introduced into the still below the inflow of the gas liquor, so that the greater part of the carbonic acid and sulphuretted hydrogen is expelled before the gas liquor mixes with the lime. This practice is advantageous in so far as it consumes less lime and avoids the formation of calcium carbonate, which is deposited in the apparatus and makes frequent cleaning necessary; it can, however, be applied only in the manufacture of ammonium sulphate, where the escaping gases other than ammonia have no detrimental effect upon the product. For the manufacture of aqua ammonia these gases must be removed from the vapor and the lime milk should enter the apparatus above the inflow of the gas liquor, so that any carbonic acid or sulphuretted hydrogen, etc., that may during the process of distillation escape with the ammonia will be retained by the fresh lime milk. At times the lime is mixed with the gas liquor and introduced into the column at a lower point, while at the same time a small amount of fresh lime milk is charged into the top of the column. The amount of lime used is generally three times that of the fixed ammonium present in the gas liquor. For the production of aqua, however, the total amount of the ammonia present should be treated as fixed salt unless the carbonic acid and sulphuretted hydrogen have been previously removed.

Impurities in Ammonia.—The purification of the vapor in cases where the final product is to be aqua ammonia is of the greatest importance. Many suggestions have been made in this regard, but few give entirely satisfactory results. The sulphur and cyanogen compounds are mostly retained in the still if distillation has been properly conducted, but organic bases and sulphur compounds are present which cannot be removed by ordinary means. The largest quantity of aqua ammonia produced is used for refrigerating purposes either as 26° solution or for anhydrous ammonia, and where it is used for this purpose the presence even of small quantities of organic impurities is most detrimental and may produce rather serious results. Investigation into the cause of explosions of refrigerating machines such as have occurred within the last year at Indianapolis and other places might bring to light the fact that the accidents were due to impure ammonia. I have seen a refrigerating apparatus the efficiency of which regularly decreased from day to day; the cause was found to be accumulation of hydrocarbon gas, which had to be withdrawn before the apparatus resumed its normal capacity. This was especially observed each time after replenishing the stock of 26° aqua with which the apparatus was supplied. The aqua had been made direct from gas liquor. The gas when ignited would burn with a bluish flame, and had an odor resembling ordinary coal gas; the aqua, while water white, showed presence of organic compounds. For refrigerating purposes aqua ammonia should not be used which shows even the slightest discoloration when treated with an excess of nitric acid. This test is fairly reliable and can easily be made by neutralizing in a beaker about a teaspoonful of the aqua with strong chemically pure nitric acid. The acid must be added gradually, permitting it to drop down the side of the slightly inclined glass, as at first the reaction is very violent. The presence of organic impurities is indicated by a light yellow to red and even brown discoloration.

Methods of Purifying Ammonia Gas.—The first requirement for complete purification is the dehydration of the ammoniacal gases coming from the distilling apparatus; and ample condensing capacity should be provided so that at no time aqueous vapor can escape with the gas. The dehydrated gas is treated in various ways for the removal of the empyreumatic substances which may be in chemical combination with the ammonia or only as admixtures volatile at low temperatures. The method most employed consists in passing the vapor through a series of cylinders filled with freshly burned charcoal, and this appears to give fair results; but it is rather a roundabout operation requiring two distillations, attended with more or less loss of ammonia, and unless the charcoal is changed frequently the aqua obtained is far from being perfect. The filters consist of a number of vertical pipes 10 to 18 in. in diameter, which are filled with charcoal. They are connected with each other alternately at the top and bottom, so that the gas passes successively up one pipe and down another through a series of five or six pipes. It is stated that such a filter will last for months without requiring a change of the charcoal; but aqua thus produced is surely unfit for refrigerating purposes. A. Mallet passes the gas through a column where it comes repeatedly in contact with oil, preferably paraffine oil. The oil is supposed to absorb and retain the impurities. The apparatus is similar in construction to the ordinary distilling column, and the gas is forced through the oil contained in the different compartments. It is not stated what becomes of the oil, which is no doubt highly charged with ammonia, after it has done service.

H. A. Frasch absorbs the impurities in a saturated solution of ammonia* by

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* United States Patent No. 515,909.

passing the gases, after they have been thoroughly dried and cooled, through a column apparatus which is constantly supplied with a small quantity of cold water. The latter, after it has once been saturated, will retain no more ammonia but will absorb the organic impurities which are soluble in ammonia; as they constitute but a small fraction of the volume of the ammonia very little water will suffice to carry out the purpose, and but a small proportion of the ammonia is retained to saturate the inflowing water. The gas passing into the bottom of the column meets first the ammoniacal fluid containing the most impurities; in the middle sections it passes the solution saturated by previously purified ammonia, and in the top of the column it reaches practically pure aqua ammonia. The uppermost compartment, where absorption of ammonia takes place, is provided with a cooling device, so that the temperature in the column never rises above 15° C. The wash product is discharged from the bottom of the column and is best utilized for ammonium sulphate or chloride. An apparatus distilling 15,000 gal. of gas liquor will produce from two to three carboys of wash fluid a day. When not enough water was charged into the wash column it has happened that the waste discharge pipe was blocked solidly with nearly pure napthaline. This apparatus does very efficient work.

For chemical treatment of the impurities potassium permanganate is probably the most effective. There are complex organic compounds in the gas which will withstand the action of the lime used in distillation, and will escape even the carbon filters and scrubbers; if oxidized these can readily be eliminated. To treat the crude gas as it leaves the still requires large quantities of permanganate, which would render the process too expensive. As these particular compounds constitute but a small portion of the impurities which can be removed by ordinary means, the treatment with permanganate is best applied after the gas has passed the carbon filters or other mechanical purifiers. In the Frasch purifier it suffices to dissolve a small amount of potassium permanganate in the water with which the apparatus is supplied. The amount required is from 5 to 8 lb. per day for an apparatus of 15,000 gal. capacity. In many instances the comparatively small amount of these impurities is permitted to remain in the gas, the more so because their presence is ordinarily not observable in freshly made aqua ammonia. If stored in iron drums such aqua will soon become discolored, and even if kept in glass or lead the color will in time be affected. Chloride of lime also is suggested as an oxidizing agent; it, however, is by no means applicable. The use of sodium plumbate is an effective and cheap method; the soda being constantly recovered, only the litharge has to be supplied.

To summarize: The different requirements for the production of a good quality of aqua ammonia by direct distillation from gas liquor are passage of the gas through fresh lime milk before it leaves the distilling apparatus; the condensation of the aqueous vapor must be perfect; the gas when dry should be purified from organic matter by suitable absorbents and oxidation of the complex compounds before final absorption in water.

Absorbing the Ammonia in Acid.—For the manufacture of ammonium sulphate the gas is conducted directly from the still, without further condensation or purification, into a body of sulphuric acid. Formerly concentrated acid was used, and the ammonium sulphate was allowed to crystallize in the absorber; but as it is much cheaper to evaporate a solution of ammonium sulphate than it is to concentrate sulphuric acid, the use of strong acid has been abandoned and chamber acid or weaker sulphuric acid is used. The different styles of absorbers have been described by Lunge and others, and no marked improvement in that direction is recorded. The apparatus most in favor consists of a lead-lined box 14 to 18 ft. in length and 4 to 6 ft. deep. It has an inclined bottom and is divided lengthwise by a partition extending from the top down to within about 8 in. of the bottom, so that when the acid is introduced the partition (sealed by the acid) forms two compartments, the rear section being hermetically closed, while the front one is supplied with a cover. Absorption takes place in the rear section. The gas is introduced through lead pipes which are connected with a perforated pipe running along the bottom the whole length of the absorber. The gas gives off the ammonia in its passage through the acid. The refuse gas, consisting mainly of sulphuretted hydrogen and carbonic acid, is conducted from the top of the absorbing compartment to a furnace, or it may be used for such purposes as the removal of arsenic from sulphuric acid. The refuse gas is best removed from the absorber by means of a pump or exhauster; in this case an extra vapor outlet should be supplied communicating with the atmosphere, so that at no time can back pressure or vacuum exist in the absorber in consequence of imperfect or too rapid exhaustion. In some instances the gas coming from the absorber is passed through a second vessel containing sulphuric acid to guard against loss of ammonia, which might have escaped the absorber. The solution of ammonium sulphate obtained in the absorber is passed through a filter and evaporated in lead-lined vessels by means of steam in lead coils. The concentration may be carried far enough to obtain the crystallized salt in the evaporator, or the hot concentrated solution may be drawn into vats where it is allowed to cool and crystallize, the mother liquor being returned to the evaporator. The purity of the sulphuric acid used for absorption is of importance when a pure white product is desired. The presence of arsenic and iron in the acid is detrimental, but can be overcome to a great extent by partial saturation and constant though slight excess of acid in the pickle. The arsenic in this case separates in the form of sulphide, and can be collected on quartz sand filters; the iron remains in solution. Periodically the accumulated surplus acid of the mother liquor is neutralized in the evaporator and the iron is precipitated.

The drying of the salt is done on shelves in heated rooms or by means of mechanical dryers; the use of the latter is less economical. A shelf dryer which gives satisfactory results consists of a number of stalls arranged in two parallel rows; they are divided by a partition embodying two flues which extend through the whole length of the dryer; each stall has an outlet into either of these flues, which can be opened or closed by means of a bell damper. One of the flues supplies hot air, while the other carries away the air and moisture. The stalls are connected with each other by suitable openings, so that if the hot air is admitted into any one it will pass successively through all the others and escape through the exit flue of the stall ahead of the one it entered, the first stall being connected with the hot air flue and the last one with the exit flue, all other connections with the flues being closed. The air is admitted into the stall in which the drying process is furthest advanced, and leaves the one which has last been charged with sulphate. This dryer is of large capacity and requires little attention.

The air is heated by steam coils supplied with the steam of the exhauster which maintains circulation of the air in the dryer. Along the seaboard it is often necessary to remove some of the moisture carried by the atmosphere, to assure proper drying efficiency. This is done in coolers, in connection with cold water coils over which the air passes before it enters the heater.

Aqua Ammonia.—The absorption of the ammonia in water may be conducted continuously or intermittently. The former method is mostly adopted in making concentrated liquor or crude aqua which has to be submitted to redistillation, in which case uniformity of strength is not of so much importance.

For the manufacture of commercial aqua ammonia, which requires a positive and constant degree of concentration, the intermittent absorber should be used. The continuous absorber is a boiler-shaped tank 4 to 6 ft. in diameter and 8 to 10 ft. long, which is provided with a cooling coil and connected with a wash column by a vapor and liquid supply pipe. The wash column consists of a number of cast-iron rings between which perforated plates are inserted; or it may be constructed like a distilling column. The water is charged into the top of the column and collects in the absorbing vessel; the ammonia enters the bottom of the absorber through a perforated pipe and passes from the absorber to the wash column. The apparatus is provided with an overflow pipe regulated to keep the absorber about two-thirds filled with ammonia solution. For intermittent operation this apparatus has a second absorber, and while the gas passes through the liquid of one tank the water from the wash tower is delivered into the other until the contents of the first one have acquired the desired strength, when the process is reversed, the saturated aqua having first been withdrawn, and the wash water is collected in the first absorber, while the second one is brought to saturation. In some instances the lead bottle system is yet in use for the absorption of the ammonia.

Producing Sulphate and Aqua Simultaneously.—When producing aqua ammonia direct from gas liquor the manufacture of a small amount of sulphate at the same time is advisible, the more so where sulphuretted hydrogen can be applied to advantage. The best results are obtained if about 25% of the ammonia is used for making sulphate. For this purpose the gas liquor is partially distilled in a separate column without the use of lime, so that the greater part of the sulphuretted hydrogen and carbonic acid is expelled before the gas liquor enters the apparatus proper. The height of this primary column and the amount of steam supplied to it regulate the quantity of ammonia evaporating with the gases. The gas coming from this apparatus must always be kept warm, to prevent deposition of ammonium carbonate and sulphide in the pipe lines. To avoid this, steam is injected into the vapor pipe near the outlet of the column. The sulphate in this case is somewhat more discolored, but if care is taken that the absorbing fluid is always kept acidulated, and the pickle is filtered before crystallization, a fair product is obtained. An apparatus producing aqua and sulphate at the same time, the sulphuretted hydrogen being used in connection with a Freiberg tower for the removal of arsenic from sulphuric acid, has been in operation for several years at Cleveland, Ohio, and though it is somewhat of a makeshift built out of an old plant, it gives fair results. The sulphuretted hydrogen is conducted to a sulphuric acid plant nearly half a mile distant, and also to a muriatic acid system located quite as far from the ammonia works. For a capacity of 20,000 gal. gas liquor per day such an apparatus should consist of a column still 60 ft. high, with 20 to 25 compartments, connected with three auxiliary stills of 5,000 gal. capacity. A second column of 8 to 10 compartments serves for the primary distillation for sulphate production. The gas liquor is delivered into a tubular condenser connected with the aqua still, whence it flows through the primary distilling column and thence to the main apparatus and to either of the auxiliary stills. If the main column consists of 24 compartments the gas liquor is delivered into about the eighteenth compartment, and the lime milk to the top or last one. The sulphate column is so placed that the gas liquor from it flows by gravity into the main column. The lime milk is delivered by a membrane pump. The primary or sulphate column is heated by direct steam injected into the bottom compartment, which is about twice the ordinary height. Gas escaping from this apparatus is conducted to the absorber, where it meets the sulphuric acid. The sulphuretted hydrogen coming from the absorber is forced by a direct steam compressor to the Freiberg tower; the steam coming from the compressor and absorber is condensed in a reflux cooler, which is connected with a suitable trap to remove the condensed water. The distilling apparatus for aqua ammonia is also heated by direct steam delivered to the auxiliary stills, the vapors of which serve to heat the column still. The vapor coming from this apparatus passes first the tubular condenser supplied with gas liquor, and then through a reflux cooler (as shown in Fig. 1), which is supplied with cold water. The thoroughly dehydrated vapor thence passes through the purifier and to the absorber. The products of condensation from the reflux cooler are conducted back into the column; the waste fluid from the purifier is delivered to the sulphate absorber. If potassium permanganate has been used in the purifier the waste fluid should be filtered before it is used for making sulphate. Charcoal filters in place of the aqua purifier may be inserted. Circulation of the ammonia vapor is assisted by an exhauster which is connected with a regulator so that any desired degree of vacuum can be maintained in the apparatus. The exhaust of this compressor may be connected with the primary distilling column and the steam may thus be utilized for making sulphate. This apparatus, though it appears complicated, does good work and is easily manipulated. It requires three men during the day and two at night to do all the work connected with the distillation of 20,000 gal. of gas liquor, including packing of products.

ANTIMONY.

There were two producers of metallic antimony in the United States in 1898, namely, the Mathison Smelting Co., of New York (works at Chelsea, Staten Island), and the Chapman Smelting Co., of San Francisco. The latter was a new concern, or rather a reorganization of an old concern—the Chapps Smelting Co. The production of the Mathison Smelting Co. was 900 tons of 2,000 lb. The Chapman Smelting Co. did not make us a report, and in the subjoined statistics its production in 1898 has been estimated at 100 tons. By far the greater part of the output was smelted at the Staten Island works. The domestic production of antimony ore increased considerably in 1898; in fact, in higher ratio than the metal, since the importation of foreign antimony ore decreased. The increase in the imports of antimony metal (or regulus, as it is often called) in connection with the increase in the domestic production indicates an important increase in consumption.

PRODUCTION AND IMPORTS OF ANTIMONY IN THE UNITED STATES.

Year.	Production.				Imports.				
	Metal.		Ores.		Metal or Regulus.		Ores.		Total Value.
	Pounds.	Value.	Pounds.	Value.	Pounds.	Value.	Pounds.	Value.	
1893........	700,000	$63,000	1,700,000	$41,000	2,780,432	$243,341	116,495	$5,253	$248,594
1894........	440,000	39,000	330,000	9,075	1,205,752	79,265	375,468	18,068	97,333
1895........	982,000	70,382	2,166,000	87,905	3,449,901	251,850	668,610	31,893	283,743
1896........	1,226,000	84,717	800,000	4,750	3,583,390	170,092	1,180,828	21,392	191,484
1897........	1,500,000	107,250	1,000,000	16,000	1,146,696	46,955	5,502,132	167,373	214,328
1898........	2,000,000	165,000	(*a*)	(*a*)	2,025,133	143,909	3,725,222	50,256	194,165

(*a*) There was an increase in the production, but complete statistics were not collected.

Antimony was mined in 1898 in Arkansas; in California (especially near Mojave, whence ore assaying 55% antimony was shipped); in Nevada; at Kingston, Idaho; and in the Coyote district, Garfield County, Utah, shipments from the last being made via Belknap. About 160 tons of 2,000 lb. were obtained from the Utah mines, part being smelted in San Francisco and part in New York. The Idaho Antimony Co., at Kingston, Idaho, mined some low grade antimony ore, which was concentrated and reduced to oxide on the spot. A new discovery of antimony ore was reported from the State of Washington.

Renewed attention has been given to the antimony deposits of Sevier County, Ark., where the United States Antimony Co. has agreed to sell its whole property (over 2,000 acres) to the American Mining and Smelting Co., of Chicago. The latter shipped a carload of ore to Chicago to be treated at an experimental works there. Besides this company several individuals were engaged in antimony mining in Sevier County in 1898, and at the end of the year they had perhaps 80 tons of stibnite ready for shipment as soon as the condition of the wagon roads would permit. The Blucher mines are to be equipped with new buildings and machinery and reopened; they are said to show streaks of ore 6 in. and 14 in. wide.

According to F. von Blucher the antimony veins in the vicinity of Antimony P. O., Sevier County, Ark., occur at the contact of graywacke (foot wall) and slate (hanging wall). Recently stibnite assaying 71·2% antimony was discovered in Sevier County, 30 miles south of Antimony P. O. and 4 miles north of White Cliffs. At the new discovery the ore occurs in large bowlders in a bed of dark-red clay in the Upper Cretaceous. This bed of clay lies over a 4-ft. stratum of limestone, which is underlaid by a stratum of marl, and that by the White Cliffs formation of chalk, which is free from flint. Over the clay containing the stibnite there is 2 ft. of sandstone, which in turn is covered by the remnant of the eroded flinty chalk. In the clay are found pieces of pure stibnite, weighing as much as 800 lb. The new discovery of antimony was first made at a depth of 22 ft. in digging a well on the R. McNeal farm.

It should be noted that the statistics of antimony above reported do not include antimony produced in the form of antimonial lead, which is referred to under the caption "Lead," further on in this volume. The antimony obtained in this form is in part used as a substitute for antimony regulus, and in part as an inferior substitute for lead. Antimonial lead sells usually for a fraction of a cent less per pound than good refined lead. With respect to the production, uses, etc., of antimonial lead the reader is referred to The Mineral Industry, Vol. V.

WORLD'S PRODUCTION OF ANTIMONY ORE. (*a*) (IN METRIC TONS.)

Year.	Austria.		Borneo. (*b*)		France and Algeria.		Hungary.		Italy.		Japan.	Mexico. (*b*)	
	Tons.	Value.	Tons.	Value.	Tons.	Value.	Tons.	Value.	Tons.	Value.	Tons.	Tons.	Value.
1893..	441	$24,339	......	....	7,250	$134,809	881	$32,160	1,193	$40,402	1,525	9	$300
1894..	686	32,943	599	$57,320	6,319	86,231	1,293	33,648	1.504	45,059	1,170	80	2,836
1895..	695	26,688	657	64,100	5,703	75,367	1,233	31,680	2,241	40,254	1,045	600	14,281
1896..	905	38,702	(*c*)	(*c*)	6,333	84,509	861	14,739	5,086	60,590	827	3,231	32,093
1897..	864	38,255	(*c*)	(*c*)	5,466	83,538	1,361	26,736	2,150	27,657	(*c*)	5,873	71,835

Year.	New South Wales.		New Zealand.		Portugal.		Queensland.		Spain.		Turkey. (*b*) (*d*)	United States.	
	Tons.	Value.	Tons.	Value.	Tons.	Value.	Tons.	Value.	Tons.	Value.	Tons.	Tons.	Value.
1893 ...	1,802	$125,460	336	$17,335	666	$51,789	30	$1,440	88	$2,784	1,035	771	$41,000
1894....	1,270	93,720	45	3,805	803	47,698	28	1,400	15	300	1,545	150	9,075
1895....	486	36,255	55	7.430	753	33,889	(*c*)	(*c*)	44	700	*Nil.*	982	37,905
1896....	135	9,170	21	2,250	595	26,775	(*c*)	(*c*)	54	1,385	100	136	4,750
1897....	172	18,060	10	763	417	28,063	(*c*)	(*c*)	354	6,483	400	454	16,000

(*a*) The foreign statistics are derived from the official reports of the several governments; those for the United States were collected specially for The Mineral Industry. (*b*) Export figures. (*c*) Not yet reported. (*d*) Fiscal years. The Turkish statistics are of doubtful accuracy.

WORLD'S PRODUCTION OF ANTIMONY METAL. (*a*) IN METRIC TONS.

Year.	Austria.		France and Algeria.		Germany.		Hungary. (*b*)		Italy.		Japan.	New South Wales.		United States.	
	Tons.	Value.	Tons.	Value.	Tons.	Value.	Tons	Value.	Tons.	Value.	Tons.	Tons.	Value.	Tons.	Value.
1893..	175	$35.539	868	$157,300	407	$64,486	361	$58,600	376	$59,569	122	36	$7,010	318	$63,000
1894..	279	51,523	1,012	136,024	424	65,244	385	50.463	351	57,626	403	30	4,535	205	39,200
1895..	296	44 175	779	98,734	989	107,960	465	57,470	423	56,986	640	44	6,460	422	70,332
1896..	422	52,655	969	130,217	1,326	156,111	500	59,292	538	72,440	311	23	3,200	556	84,717
1897..	424	51,350	1,033	141,800	1,665	210,744	523	63,373	404	57,072	(*c*)	(*c*)	(*c*)	680	107,250

(*a*) From the official reports of the respective countries. (*b*) Crude antimony and regulus. (*c*) Not reported.

Canada.—The old antimony mines at Rawdon, Nova Scotia, were reopened during the summer of 1898, with a view of resuming operations in a more systematic manner than ever before. The mine is opened on two fissure veins, said to average nearly a foot in width, which in places show solid stibnite highly auriferous. The old dump, representing several thousand tons of material, is being shipped to England for treatment. It is said to contain from 10 to 15% antimony, and sometimes as much as $18 gold per ton.

China.—Antimony sulphide is said to be abundant in the Nan-ning, Ssuch'eng, Chen-an, and T'ai-p'ing prefectures, and particularly in the district of Hsi-lin, near the Yunnan border. A small shipment of this ore to Hong Kong was made in 1898.

France.—Several veins were discovered showing a considerable quantity of ore, which after sorting assays 15 to 18% antimony, 30 to 35% lead, and over 30 oz. silver per ton. Some difficulty was experienced in marketing this ore, the proper metallurgical treatment being problematical.

Hungary.—The occurrence of antimony ore in the Rechnitzer Schiefergebirges was described by A. Schmidt in a paper, "Ueber einige Minerale der Umgegend von Schlaining, in Groth's *Zeitschrift für Krystallographie u. Mineralogie*, 1898, Vol. XXIX., No. 3, p. 194; abstract in *Zeitschrift für praktische Geologie*, December, 1898, Vol. XII., p. 433.

India.—Lodes of stibnite occur at Shigri in Lahoul, and also in the Jhelum district of the Punjab. The most promising deposits of antimony in India, however, are considered to be those in the Kyaukkaung tract of Tenasserim. This tract adjoins the territory of Siam, and the mines occur on both sides of the line. Stibnite is said to occur there in irregular pockets in fissures in limestone near the contact with overlying slates.

Japan.—The exports of antimony from Japan in 1897 were 1,577 metric tons, against 624 in the previous year.

Mexico.—The production of antimony in Mexico is becoming more important. At present the ore is exported entirely, for the most part to the United States, England and France, but the erection of smelting works in Mexico is contemplated. The Mexican ore averages 50% antimony, while the French and Italian ores do not exceed 25%.

New Caledonia.—Running from Thio across the island westerly to near Noumea is a large belt of quartzite, along which are found veins of stibnite. Some attempts have been made to exploit these, but although a considerable sum of money has been spent at intervals, no successful result has yet been obtained, and the efforts have ceased.

Servia.—The Société Francaise Miniere et Metallurgique en Serbie has purchased the mines at Losnica and proposes to exploit them on a large scale. There was a production of 48 metric tons of antimony metal in Servia in 1895. None was reported in 1896. No later statistics have been published.

Turkey.—There are many deposits of stibnite in Asia Minor, but very little definite is known about them. Antimony ores are obtained near Aidin and exported chiefly to England via Smyrna. In 1895 the shipments amounted to 1,322 metric tons. Only the deposit near Endemisch is exploited at present. The ore of this mine is of very high grade, rising to nearly 60% Sb. The yearly production is from 2,000 to 3,000 tons.* Discrepancies between these figures and those elsewhere published may be due to the difference between the calendar and fiscal years.

Deposits of stibnite are also known on the islands of Mitylene and Chios. Antimony ores have also been found near Adrianople, at Monastyr, in Macedonia, and at several other points in European Turkey. One of the best known places is at Allkhar, near Rozdan, where there is a wide vein at the contact between gneiss (or mica schist) and dolomite. The ore is found in pockets in a clay gangue. At the western end these pockets contain very pure antimony ore, but further on the ores are associated with realgar, and still further arsenic ore only is found. The mines were first opened about 1880, and are well developed. The ore is dressed in a plant which has a capacity of 2,500 tons yearly. The largest production was in 1892, when 1,200 tons, concentrated to about 55% Sb, were shipped to England. The mines are worked now chiefly for the arsenic ores, of which about 500 tons are shipped yearly.†

THE ANTIMONY MARKET IN 1898.

New York.—Inasmuch as the production of refined star antimony in the United States is not as yet sufficient to supply the demand, the course of the market during 1898 was again more or less dependent on prices in Europe. Japan was not quite so important a factor as it used to be. On the other hand fair quantities of Hungarian antimony are reported to have been imported. The year opened with Cookson's quoted at 8c., Hallett's 7⅜c., U. S. Star 7⅜c., and Japanese 7⅜c. to 7½c. Old stocks being quickly cleared off and the foreign market advancing, new supplies were obtainable only at higher prices, and consequently during February prices advanced to 8⅛c. for Cookson's, 7⅝c. for Hallett's, U. S. Star and Japanese. Then, owing to a good consumptive demand, prices advanced steadily during the next six months, until in October the highest point was reached, namely 9¾c. for Cookson's and 9 to 9⅛c. for Hallett's, Japanese and U. S. Star. At the end of the year European, as well as home producers, were more anxious to sell, and values declined. December closed quiet, with the ruling quotations 9c. for Cookson's, 8¾c. for "C" antimony (produced on the Pacific Coast and brought on the market within the last few weeks), 8¾c. for Hallett's, 8⅝c. to 8¾c. for Japanese, 8¾c. for U. S. Star and 8⅜c. to 8½c. for Hungarian.

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* R. Helmhacker, *Engineering and Mining Journal*, Nov. 26, 1898.

† *Montan Zeitung*, Sept. 1, 1898.

AVERAGE MONTHLY PRICES OF ANTIMONY IN NEW YORK, IN CENTS PER POUND.

Year.	Brand.	Jan.	Feb.	Mar.	April.	May.	June.	July.	Aug.	Sept.	Oct.	Nov.	Dec.	Year
1892....	Cookson's	15.88	15.00	14.90	15.00	15.13	14.40	14.20	13.50	11.50	11.80	12.00	11.50	13.75
	Hallett's..........	14.40	11.00	10.90	11.00	11.50	11.25	11.00	10.75	10.50	10.50	10.63	10.75	12.10
	L. X...............	12.13	12.13	12.50	12.50	13.00	12.75	12.50	11.88	11.25	11.20	11.19	10.80	12.00
1893....	Cookson's	11.00	10.75	10.75	10.75	10.13	10.50	10.50	10.25	10.30	10.25	10.00	10.25	10.50
	Hallett's..........	10.25	10.38	10.00	10.00	10.00	9.90	9.88	9.75	9.60	9.75	9.50	9.25	9.88
	L. X...............	10.50	10.63	10.38	10.25	10.38	10.00	10.13	10.00	10.00	10.00	10.00	10.00	10.20
1894....	Cookson's	10.25	10.00	10.13	10.13	10.13	9.75	10.00	10.00	9.50	9.63	8.50	8.38	9.63
	Hallett's..........	9.50	9.38	9.50	9.38	9.25	9.13	8.75	8.88	8.88	8.25	8.25	8.13	8.88
1895....	Cookson's	8.00	8.25	8.15	8.00	8.00	8.00	8.00	7.85	7.85	7.75	7.75	7.75	7.95
	Hallett's..........	7.25	7.25	7.10	7.00	7.00	7.05	7.10	7.20	7.25	7.13	7.00	7.00	7.11
	U. S. Star........	8.25	7.50	7.38	7.50	7.35	7.50	7.50	7.75	7.50	7.50	7.50	7.50	7.56
	Japanese	7.00	7.00	6.88	6.88	6.88	7.00	7.00	7.20	7.00	6.88	6.88	6.88	6.94
1896....	Cookson's	7.55	7.43	7.37	7.44	7.50	7.37	7.00	7.00	7.00	7.00	7.00	7.18	7.23
	Hallett's..........	7.10	6.90	6.87	6.81	6.75	6.75	6.75	6.50	6.50	6.50	6.46	6.68	6.71
	U. S. Star........	7.25	7.09	7.00	7.00	7.00	6.91	6.75	6.75	6.75	6.75	6.75	6.93	6.91
	Japanese	6.75	6.75	6.75	6.75	6.75	6.75	6.75	6.75	6.75	6.75	6.75	6.75	6.75
1897....	Cookson's	7.12	7.31	7.37	7.44	7.25	7.25	7.25	8.00	8.00	8.00	7.81	7.81	7.55
	Hallett's.	6.80	6.90	7.06	7.00	7.00	7.00	7.00	7.28	7.37	7.50	7.40	7.40	7.14
	U. S. Star........	6.80	7.00	7.06	7.22	7.05	6.78	6.87	7.28	7.37	7.50	7.40	7.50	7.15
	Japanese	6.75	6.80	6.75	7.00	6.75	6.78	6.87	7.15	7.25	7.25	7.15	7.18	6.91
1898....	Cookson's.	8.00	8.00	8.25	8.37	9.20	9.65	9.75	9.75	9.75	9.70	9.25	9.25	9.08
	Hallett's..........	7.37	7.56	7.81	7.96	8.64	8.97	9.06	9.06	9.06	9.03	8.81	8.81	8.51
	U. S. Star........	7.50	7.59	7.81	7.92	8.64	8.97	9.06	9.06	9.06	9.03	8.81	8.81	8.52
	Japanese	7.44	7.58	7.81	7.90	8.59	8.97	9.06	9.06	9.06	9.03	8.81	8.81	8.51

London.—The market was affected throughout the whole year by a great scarcity of suitable raw material, which was consequently in great request. Prices opened firm at £30 to £30 10s., and remained steady thereat until the middle of February, when there was an advance to £31 to £31 10s., followed in March by a further improvement to £31 10s. to £32. This price was quoted until nearly the end of April, when there was another rise to £32 to £33. In May £34 10s. was asked. The price then rose almost weekly until at the beginning of July £36 10s. to £37 was quoted, at which point it remained stationary until the end of the year.

PROGRESS IN THE METALLURGY OF ANTIMONY.

The control of the antimony industry by a few producers and the secrecy with which they envelop their operations prevents the appearance in technical literature of anything but scattered and incomplete notes as to progress in the methods of winning this metal, and there is nowhere any account of the best practice of the present day. This is unfortunate since there is known to have been considerable improvement in the methods of at least the English smelters, who have been probably the most progressive in this branch of metallurgy.

There were few patents granted in 1898 for the treatment of antimony ores. One of the most interesting was that of N. W. Edwards of London, who proposes to treat gold and silver-bearing antimony ores by lixiviation with a 7% solution of calcium sulphide (cold) in large vats provided with stirring apparatus, whereby the antimony is dissolved as sulphide and the gold left in the residue, whence it may be recovered by an ordinary method. The solution is drawn off and treated in closed vessels with carbonic acid obtained by burning lime, whereby the calcium is precipitated as carbonate, which is filtered off. The solution is then treated again with carbonic acid which precipitates antimony sulphide. The sulphuretted hydrogen which is given off by the decomposition of the double sulphide solution is conducted through milk of lime to regenerate calcium sulphide.*

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* English Patent No. 15,791, July 2, 1897.

Antimony-Copper Alloy.—A new alloy which is claimed to be a wonderful substitute for gold consists of 94 parts of copper and 6 parts of antimony. The copper is melted and the antimony added, together with a little magnesium and calcium carbonate. The product can be drawn, wrought and soldered like gold, to which it bears a striking resemblance when polished, it is said.

Analytical Methods.—Josef Hanus proposes to determine antimony volumetrically by precipitating it as sulphide, filtering and washing the precipitate with water, then washing the precipitate into a beaker and adding ferric sulphate (either in powder or in solution) in the proportion of six parts to one part of antimony sulphide. The solution is then boiled for 15 minutes, the following reaction taking place:

$$Sb_2S_3+5Fe_2(SO_4)_3+5H_2O=H_2Sb_2O_5+10FeSO_4+5H_2SO_4+3S.$$

The contents of the beaker are then diluted considerably and 15 c.c. of concentrated sulphuric acid are added for each gram of ferric sulphate used. The solution is then cooled quickly to 25° C. and titrated to a faint rose color with potassium permanganate. The permanganate solution is made up to a strength of 1 to 1·5 g. per litre and is standardized against antimony trioxide. The amount of antimony sulphide in the precipitate treated with the ferric sulphate should not exceed 0·2 to 0·3 g.*

H. Causse describes a new method for the volumetric estimation of antimony, which is based on the fact that antimonious acid, free or combined, reduces iodic acid, with the formation of antimonic acid and the liberation of a proportionate amount of iodine, this reaction being expressed by the following equation:†

$$5Sb_2O_3+4HIO_3=5Sb_2O_5+4I+2H_2O.$$

The solutions required are as follows: Iodic acid, 50 g. in 250 c.c.; decinormal sodium thiosulphate; 20% potassium iodide; and starch paste. Into the flask of a Fresenius or Mohr's iodometric apparatus, 0·5—0·6 g. of antimonious oxide is weighed, 20—25 c.c. of iodic acid added, and 10 c.c. of the potassium iodide solution is placed in the condensing apparatus. The contents of the flask are boiled until colorless, and the collected iodine determined with thiosulphate. From antimony compounds, the sulphide is precipitated by sulphuretted hydrogen, then dissolved in hydrochloric acid, and the oxide obtained by the action of sodium carbonate. The halogen acids, sulphurous acid, and sulphuretted hydrogen must be completely removed, since these act on iodic acid. The antimonious oxide may be dissolved in tartaric acid or potassium bitartrate, which are without action on iodic acid.

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* *Chemiker Zeitung*, Dec. 25, 1897.

† *Comptes Rendus*, 1897, CXXV. (25), pp. 1100-1103.

ARSENIC.

There is no production of arsenic in the United States at the present time with the exception of that which is recovered as sulphide in the purification of sulphuric acid, and is at present a useless product. The Pennsylvania Salt Manufacturing Co., at its works at Natrona, Pa., recovers in this way about 150 short tons of arsenic sulphide per annum and the Grasselli Chemical Co., at Cleveland, Ohio, probably recovers 140 tons, but at both places the product is buried in the earth to get rid of it. The erection of the white arsenic plant at the works of the Puget Sound Reduction Co., at Everett, Wash., to which reference was made in The Mineral Industry, Vol. VI., was discontinued for some reason not made public.

The domestic resources in arsenical ore are so large that the preparation of arsenic and arsenical compounds should be undertaken in this country instead of importing them from abroad. The imports of metallic arsenic, white arsenic (arsenious acid) and arsenic sulphides (orpiment and realgar) into the United States during the last six years have been as follows: 1893, 6,092,377 lb. ($180,333); 1894, 7,063,442 lb. ($218,636); 1895, 6,984,273 lb. ($237,745); 1896, 5,813,287 lb. ($215,281); 1897, 7,242,004 lb. ($352,284); 1898, 8,686,681 lb. ($370,347).

The world's supply of arsenic and arsenical compounds continues to be derived almost entirely from Prussia, Saxony (Freiberg) and Cornwall. The methods of arsenic winning employed in Cornwall were fully described in The Mineral Industry, Vol. III.; those of Saxony in Vol. IV.; while in Vol. V. there was a description of an interesting plant for the treatment of an arsenical gold ore, with recovery of the arsenic, at Bovisa, Italy.

ARSENIC PRODUCTION IN ENGLAND AND GERMANY. (*a*)

Year.	Prussia.		Saxony. (*b*)		Total Germany.		England. (*c*)	
	Metric Tons	Value.	Metric Tons	Value.	Metric Tons	Value.	Metric Tons	Value.
1893..........	710	$31,939	1,084	$87,648	1,794	$119,588	6,072	$288,470
1894..........	1,147	57,877	1,229	109,118	2,389	166,798	4,878	243,070
1895..........	1,788	98,363	1,217	108,826	3,014	207,416	4,875	260,990
1896..........	1,750	130,539	882	90,626	2,637	221,397	3,674	227,415
1897..........	1,924	143,775	853	....	2,989	270,949	4,232	373,975

(*a*) From the official reports of the respective countries. (*b*) Includes arsenious acid, red, yellow and white glass, and metallic arsenic, except the figure for 1897 which represents metallic arsenic alone.. (*c*) Arsenious acid.

There are large deposits of arsenical ores in Hastings County, Ontario, but no production of arsenic in 1898 was reported by the Canadian Geological Survey. However, an output from this source may be expected in 1899 since the Toronto Smelting Co. of Belleville, Ontario, is carrying on experimental work with a view to the production of arsenic and mineral wool, while the Kaladar & Anglesea Mining and Development Co. is experimenting on arsenical ores with the Eames process, and the Canadian Goldfields Co. of Deloro expects later to recover arsenic from the ores of the Deloro mines.

Turkey is a considerable producer of arsenic ore. According to R. Helmhacker the mines at Allkhar, near Rozdan, in Macedonia, yield about 500 metric tons of orpiment per annum, while this mineral and realgar are mined elsewhere in Macedonia to a small extent.* Large and rich deposits of arsenical pyrites are said to exist in the Province of Aidin, in Asia Minor, in the vicinity of Endemisch and elsewhere. The most important deposit now worked is situated in the neighborhood of Jeniköï, where from 600 to 800 tons per annum are produced. This ore assays as high as 42% arsenic and carries considerable gold. There is probability of more attention being devoted to these deposits.†

Realgar is mined in the Tahkt-i-Soleiman Mountains, in the Aserbeishan district, in Persia, 14·5 km. from the northern fork of the Saruk River. The country rock is slate, penetrated by melaphyres and basalts. The realgar occurs in veins 0·33 to 0·50 m. wide at the contact of melaphyre and slate. The mines are worked under lease from the government for a royalty of 7·5 tons of first grade realgar per annum. This grade of mineral sells at Hamadan at 330 fr. per ton. The second grade brings 170 fr. and the third 100 fr. The mines employ eight men, who work seven months of the year, the region being inaccessible in the winter.

Five hours north of this mine, near the village of Goromis, there is another occurrence of realgar in thin veins, accompanied by orpiment. Occurrences are also noted in the vicinity of Kazvin, and near the Afshar lead mine.‡

Market Conditions.—Notwithstanding an increase in the importation of arsenic into the United States in the early part of 1898 stocks continued light, and prices were expected to advance but efforts were made more or less successfully to keep it down. The demand from the glass trade was delayed somewhat by the difficulties of the glass manufacturers in settling the wage scale with the cutters and flatteners. The demand from the paris green makers, on the other hand, started up earlier than usual, many of the makers preferring to manufacture by the slow process, while there is generally a desire to make shipments to the West before the close of canal and lake navigation. Early in the summer the arsenic market was easy, the demand being slack and some of the makers of paris green offering to sell their surplus stock of arsenic. In September a combination of the English producers was effected and imparted an upward tendency to the market and in October 3·87c.

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* *Engineering and Mining Journal*, Nov. 26, 1898.

† *Montan Zeitung*, Sept. 1, 1898.

‡ R. Helmhacker, *Engineering and Mining Journal*, July 9, 1898.

per lb. was quoted for shipments and 4 to 4·5 c. for immediate delivery, while at the end of the year futures also were quoted at 4 to 4·5c. The cost of white arsenic in England is governed to certain extent by the position of the tin mines from which the arsenic ore is obtained as a by-product, but chiefly by the arsenic combination. Red arsenic, which comes largely from Germany, sold at 7·75 to 8c. per lb.

Paris Green.—In February some of the large producers of this substance formed a combination under the name of the Commercial Chemical Co. of the United States, which was entered by A. B. Ansbacher & Co., I. Pfeiffer, Frederick L. Lavenburg and Morris Hermann & Co. The new concern at first fixed the price at 16 to 16·5c. per lb. for strictly pure paris green in kegs or casks, but in the middle of May was forced to reduce it to 15 to 15·5. These prices could not be maintained, however, against the competition of such outside producers as the Highlands Chemical Co. In December the outside producers quoted 9 to 9·5c. per lb. for deliveries in 1899, whereupon the combination gave up the fight and dissolved.

Paris green, which is an aceto-arsenite of copper, is made commonly by adding to a copper sulphate solution sufficient sodium carbonate to precipitate one fourth of the copper. Acetic acid is then added until the precipitate is just redissolved and the solution is heated to boiling. By the addition of a solution of sodium arsenite (arsenious acid dissolved in a solution of sodium carbonate) a precipitate of paris green is thrown down. This is filtered, washed and dried at a low temperature. In order to produce the best grade of pigment the solution should be dilute.

Since many silver-lead smelters and refiners produce a considerable amount of copper sulphate and have a supply of arsenic (available as arsenious acid) in speiss which has to be roasted for recovery of its gold and silver values, the manufacture of paris green, which is a readily marketable product, would appear to be a promising field for these smelters to enter. Paris green is used as a pigment, having a peculiar bright green shade possessed by no other. Owing to its highly poisonous character, however, its employment for this purpose is somewhat restricted. The larger part of the present production is used as an insecticide in agriculture.

ASBESTOS.

The production of asbestos in the United States in 1898, as in the previous year, was derived chiefly from the Sall Mountain mine at Sauter, White County, Ga., while the imports which furnished most of the requirements of domestic consumption came as heretofore from the Province of Quebec. Outside of Georgia there was no production in the United States in 1898 with the exception of a few tons in California. The increase in the production at Sall Mountain was due to an improved demand, a considerable quantity of mineral having been exported to Germany and England. The value of the product ranged from $16 to $25 per ton of 2,000 lb. at the mine, varying according to color, length and quality of the fiber.

ASBESTOS STATISTICS OF THE UNITED STATES.

Year.	Production.				Imports.		
	Short Tons.	Metric Tons.	Value. (*a*)	Value per Metric Ton.	Manufactured.	Unmanufactured.	Total.
1891	66	60	$3,960	$66·00	$4,872	$353,589	$358,461
1892	100	91	5,000	54·94	7,209	262,433	269,642
1893	120	109	6,000	55·00	9,403	175,602	185,005
1894	265	240	4,300	18·91	15,989	240,029	256,018
1895	664	602	11,837	19·66	19,731	225,147	244,878
1896	716	650	12,670	19·49	15,654	229,084	244,738
1897	840	762	12,950	16·99	10,570	264,220	274,790
1898	885	803	13,425	16·72	12,899	287,636	300,535

(*a*) The values given for 1891, 1892 and 1893 are of doubtful accuracy.

A new product called "asbestolith," a German invention, is now being made by the Sall Mountain Asbestos Co. It is used chiefly for the preparation of cement and tiling for floors, for which it is claimed to have many advantages, among them impermeability to water, and elasticity as high as that of wood, a hardness equal to that of cement, greater durability than asphalt, and lightness in weight, while it is, moreover, a non-conductor of sound. It is said that it will not crack, warp or bend, and shows greater resistance to abrasion than stone, brick or marble. Short fiber asbestos is worked up into this material, and since the mineral of Sall Mountain is chiefly of this character there will be a considerable future for the new branch of the business if the real merits of the product prove to be equal to the claims made for it.

ASBESTOS IN FOREIGN COUNTRIES.

Bosnia.—A discovery of asbestos was reported at Halilovac in the northwest of this province.

Canada.—Mr. J. Obalski contributes the following notes:

"In the eastern townships of Quebec, during 1898, asbestos was regularly mined in the districts of Thetford and Danville, and some work was done also at Broughton and Black Lake. The demand for Canadian asbestos was good, and there was a small increase in the price. Thetford remains the most active center of the industry, the three companies, namely the Bell, King Bros., and Johnson, having worked the whole season at their full capacity, employing over 600 men. These mines are very well equipped with steam plants and air compressors, traveling derricks, tramway accommodation, and grinding and separating mills, and they can produce, at any time, a much larger quantity, if required. The shipping facilities are excellent, the Quebec Central Railroad passing over the surface of the mines.

"The Canadian asbestos industry has become well settled, and the different marks of the products are now well known. The former classifications have been changed since a few years into a new one. The different companies prepare, by hand sorting, the first crude of a length from ¾ in. up, and sometimes the second crude for special orders. The other grades are treated at the mill, and known as fiber, paper stock, asbestic, and asbestic sand, each being marked by letters.

"These mines, and their plants and mills, have been already described so fully that it is needless to give further accounts of them, there having been no new features in 1898. The deepest quarry is down 120 ft. The grinding is done mostly by the cyclone process, and the separation of the fiber from the rock is accomplished by screening and blowing.

"Some work was done in 1898 at Broughton, where a small mill was operated by the United Asbestos Co., and at Black Lake, by the Johnson Co. It is hoped that in 1899 other companies will be operating at Black Lake.

"At Danville, the Asbestos and Asbestic Co. worked 250 to 300 men. A description of its plant was given in THE MINERAL INDUSTRY, Vol. V. Since then there have been a few improvements. The works are now equipped with seven boilers able to supply 1,000 h. p. and the mill has been enlarged. A branch line, 4 miles long, connecting the mine with the Grand Trunk Railroad has been built. The deepest opening is about 110 ft. The mining is done mostly by contract, and with about 10 traveling derricks an average of 750 tons of rock is handled per day, of which a large part goes to the mill. This company has made special preparations for the manufacture of "asbestic," which is especially well adapted as a fire-proof material for plastering, and a large quantity of this is produced besides the ordinary asbestos fiber.

"In the Laurentian district a special variety of short fiber asbestos is mined on a small scale. It is found in a honey yellow serpentine, and is considered not to contain iron. This material is disseminated in several places through the Ottawa region, but practically it would not pay to sell it in a crude form, and it has to be dressed by cyclone and blowing apparatus. Two such mills have been erected, one near Pointe au Chêne, on the Canadian Pacific Railway, formerly known as the Non-Magnetic Asbestos Co.'s, and actually owned by the Cass Asbestos Co., of Montreal, but it was not in operation in 1898; the other (formerly the Asbestos Mining and Manufacturing Co., of Newark, N. J., but actually controlled by Chas. Wurtele & Co., of Ottawa) situated on the Gatineau River, near Low Station on the Gatineau Valley Railroad. The latter property was regularly worked, and produced a considerable quantity of fiber and asbestic, these products being remarkable for their whiteness. This mine is well developed and has a depth of 90 ft.; the mill is in good working order, and the company is able to supply a fairly large output.

"The quantity of asbestos and asbestic shipped from the eastern townships during 1898 was 23,015 tons of 2,000 lb. of an estimated value at the mines of $500,000, including about 7,000 tons of asbestic; more than 900 men were employed through the whole year. These figures do not comprise the shipments from the Ottawa district, which amount to only a few hundred tons. In 1897 the shipments were 23,565 tons, of an estimated value of $400,000, including about 11,000 tons of asbestic."

New Zealand.—The New Zealand Asbestos Co., of Christ's Church, exploited veins of asbestos on the upper Takaka River, in the Province of Nelson, but the enterprise failed after a lot of time and money had been spent on the outcrops, and no asbestos is being produced in New Zealand at present.

South Africa.—Paul Kersting investigated the composition and properties of South African asbestos, making comparison with typical samples of Canadian asbestos.* The first sample of the latter showed long silver gray, lustrous and silky threads which could be separated easily. The cost of this material was 50c. per lb. The second Canadian sample was of a dirty gray color and possessed a dull, short and woolly fiber. It was valued at 25c. per lb. The African asbestos was a very light, voluminous, silky, bluish gray mass, traversed by darker fibers. The compositions were as follows:

	SiO_2	Al_2O_3	Fe_2O_3	FeO	CaO	MgO	K_2O	Na_2O	P_2O_5	SO_3	H_2O	Total.
I. Canada	39·04	......	0·25	2·10	0·51	42·57	0·09	0·22	......	0·10	15·20	100·08
II. Canada	41·56	3·42	0·41	6·67	0·83	33·38	0·11	0·30	......	0·08	13·15	99·91
III. Africa	43·61	0·21	16·57	12·15	0·89	7·02	3·06	2·14	0·06	0·06	14·30	100·07

It appears from the above that each of the three samples was hydrated magnesium silicate and iron protoxide, or serpentine asbestos. Before the blowpipe they were infusible. When heated in a muffle to 1,000° C. sample I. changed but little; it only became somewhat brittle. Sample II. assumed a reddish color and became strongly brittle. Sample III. changed to a hard red powder in which here and there small crystal needles were preserved.

The acid-resisting power was determined by immersion in chlorhydric acid for 48 hours. Under this treatment the color of the Canadian sample became purer and whiter, while that of the African remained unchanged. The loss in weight was as follows: I., 41·24%; II., 37·48%; III., 12·62%.

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* Communication from the Technological Laboratory of the Royal Technical High School at Berlin.

A ¾-in. rope of South African asbestos carrying 220 lb., and entirely surrounded by the flame from a gas burner, did not break until after 22 hours. As regards fineness, softness of fiber, and suitability for spinning the African asbestos was considered superior to all others and especially suitable for acid filters. On the other hand Canadian asbestos is decidedly preferable where a resistance to heat is required, since while it still possesses a certain cohesion after loss of its chemically combined water, the African asbestos is entirely decomposed to powder at the same temperature. As non-conductors of heat the Canadian and African products are equally serviceable and the determining factor is chiefly the price.

In a paper, "Notes on Blue Asbestos," read before the Institution of Mining and Metallurgy, January 18, 1899, H. F. Olds described the occurrence in Griqualand West, where it is found in veins, seldom less than 2 in., and more often 4 to 5 in. wide, formed of closely compacted parallel fibers, which extend from wall to wall of the vein. The grain is very fine, and even in the rough state the fibers are singularly distinct. The fiber always lies at right angles to the sides of the deposit. They are somewhat elastic, and easy separable by the fingers. The inclosing rock is a dark brown shale. Its character varies considerably, some places are soft and some hard; the better quality asbestos occurs in the hard rock.

The color of the mineral is a peculiar lavender blue, and is caused by the large proportion of iron protoxide it contains. Its composition is given as: Silica, 51·1%; protoxide of iron, 35·8; soda, 6·9; magnesia, 2·3; water, 3·9; total, 100%. The present output is over 100 tons per month, and there is a large capacity for an increased turnout. Very little skilled labor is required, the mining being mostly surface work. The cost of production is about £5 per 2,000 lb.

However, with all the merits of this mineral, the business in it does not become prosperous. At the annual meeting of the Cape Asbestos Co., October 11, 1898, it was stated that the operation of the previous year showed a net loss of £3,808. Although the blue Cape asbestos was claimed to be as good as, or superior to, the white or Canadian asbestos for many special purposes, it was, nevertheless, admitted that the introduction of this material into the trade was not making the rapid progress that was looked for. In 1895 the sales amounted to £9,000; in 1896, £15,000; 1897, £15,500, while the first six months of 1898 indicated a business of £9,000. However, to cover all the expenses, sales of at least £25,000 would be required.

Western Australia.—The Mining Warden of the Kalgoorie district in Western Australia reported the discovery of asbestos in paying quantity near Feysville.

Uses.—A new use for asbestos is in the manufacture of asbestos leather, which is made by dividing asbestos into very fine fibers, immersing and thoroughly coating them with a solution of rubber and afterward evaporating the solvent, the fibers then cohering perfectly. A mass may be pressed or rolled into any desired form, and the product is said to resemble leather very closely in its structure and general characteristics.

ASPHALTUM.

UNDER this caption are treated all the mineral pitch or tar products, which pass by gradations from solids through various stages of liquidity into petroleum, and for commercial purposes may be classified satisfactorily as (1) asphaltum proper, solid or liquid, the latter being known also as maltha; (2) asphaltic, or bituminous, limestone; (3) bituminous sandstone; and (4) varieties of commercially high grade hydrocarbons known as albertite, grahamite, ozokerite, uintaite and wurtzilite, which substances may be referred to generically as glance pitch. Bituminous limestone and bituminous sandstone are, as indicated by their names, limestone and sandstone impregnated with asphaltum. This classification corresponds substantially to that of Gen. F. V. Greene, who classifies bitumen as follows; (1) *a* liquid (petroleum), *b* viscous (maltha), *c* solid (glance pitch); (2) bitumen mixed with earthy matter, as asphalt from the Trinidad pitch lake; (3) bituminous sandstone; (4) bituminous limestone.

PRODUCTION OF ASPHALTUM AND BITUMINOUS ROCK IN THE UNITED STATES.
(In tons of 2,000 lb.)

States.	1896.			1897.			1898.		
	Tons.	Value.	Per Ton.	Tons.	Value.	Per Ton.	Tons.	Value.	Per Ton.
Bituminous sandstone:									
California	(*a*) 49,456	(*a*) $122,500	$2·47	*a*45,470	*a*$128,173	$2·82	*a* 46,836	*a*$137,575	$2·93
Kentucky	2,500	10,000	4·00	2,000	8,000	4·00	2,700	8,700	3·22
Total	51,956	$132,500	$2·55	47,470	$136,173	$2·87	49,536	$146,275	$2·95
Asphaltic limestone:									
Utah	800	4,000	5·00	2,140	10,700	5·00	(*c*) 150	750	5·00
Texas	4,200	51,000	12·14	None.	....	....	None.	....	....
Indian Territory	....	....	....	250	750	3·00	13,949	69,745	5·00
Total	5,000	$55,000	$11·00	2,390	$11,450	$4·79	14,099	$70,495	$5·00
Asphaltum:									
California	(*a*) 20,414	(*a*) 362,590	17·76	*a*27,397	*a*486,620	17·76	*a*25,690	(*a*)482,175	18·77
Grahamite:									
Utah	1,282	38,460	30·00	1,756	52,680	30·00	(*b*)2,675	80,250	30·00

(*a*) Statistics of the California State Mineralogist. (*b*) Including 115 tons produced in Colorado. (*c*) Estimated.

California.—The production of asphaltum and bituminous sandstone in this State is derived chiefly from Santa Barbara, Santa Cruz and San Luis Obispo counties. The asphaltum proper exists in two forms, solid and liquid (the latter being known also as maltha). It is produced largely by the Alcatraz Asphalt Co., at Carpenteria, by the California Asphaltum Co., Ltd., at Ventura, the Standard Asphalt Co. at Asphalto, and the Western Oil and Asphalt

Co. at Los Angeles. The bituminous sandstones are scattered at various points through the coast ranges, but are found chiefly in the counties first mentioned. There is an intimate association between the occurrences of asphaltum and petroleum in California, the petroleum of the State being of an asphaltic character, and most of the petroleum refiners produce a small amount of asphaltum from their residuum.

California was the only State which produced refined asphaltum in 1898. The average price realized for the product at Ventura, Asphalto, and other important points of production was between $20 and $30 per ton. The Standard Asphalt Co. removed its refinery to a more desirable location within a few hundred feet of the mine workings at Asphalto. The plant was entirely overhauled and is now one of the best designed in the State, being arranged so that the transmission of all material, from the time it leaves the mine shaft until ready for shipment, is done by gravity. The California Asphaltum Co., at Ventura, refined some asphaltum by the use of brimstone developing hydrogen sulphide which was burned to sulphuric anhydride and utilized for the fabrication of sulphuric acid. A new discovery of bitumen was reported in the Mattole Valley, 12 miles from the coast in the southern part of Humboldt County.

California was the largest producer of bituminous sandstone, as heretofore, the chief producers being the City Improvement Co., the Consolidated Bituminous Rock Co., H. T. Cowell & Co. (Union Paving Co.), and the San Luis Obispo Bituminous Rock Co. This product is consumed chiefly for paving in cities on the Pacific coast. There is no production of asphaltic limestone in California worth mentioning.

Indian Territory.—Further developments were made in the asphaltum deposits of this Territory, which begin at a point 3 miles from Dougherty and Davis on the Sante Fe Railway and extend over an area 15 miles in length and from 3 to 5 miles in width. The asphaltum occurs impregnating a blue limestone and also in admixture with sand. The deposits are said to be very large, and promise to be an important source of supply. There are three plants already in operation for preparation of mastic, turning out from three to four carloads per day, and the manufacture of asphalt blocks is contemplated. There was a gradual growth in the production of asphaltum and mastic in 1898, shipments being made both from Dougherty and Dallas. Late in the year and early in 1899 new discoveries of both asphaltic limestone and sandstone were made not only in the district above mentioned, but also in that adjacent to it. There are two companies operating in the vicinity of Dougherty, namely, the Gilson Asphaltum Co. and the Brunswick Asphalt Co. The total production of the Indian Territory in 1898 was 14,099 short tons, valued at $5 per ton, shipped chiefly as mastic. The mastic itself was worth $8 per ton.

Indiana.—A new discovery of asphaltic limestone was made at a depth of 85 ft. in drilling a gas well near Logansport, Ind. The deposit will be developed during 1899.

Kentucky.—The production of bituminous sandstone in this State fell off in 1898 on account of the fight forced on the local producers by the Barber Co. and other makers of asphaltic compounds. It was the effort of the outside concerns to have the specifications in all large cities framed so as to exclude natural rock asphalt. The combination made its strongest fight at Louisville, knowing that if it could keep the Kentucky asphalt out of that market it would have a strong card to play in other cities. However, the attempt was a failure. There were two producers in Kentucky in 1898, their product being worth $1·80 to $5 per 2,000 lb., according to quality and distance from market.

Oklahoma.—Deposits of asphaltic stone are exploited near Oklahoma City.

Texas.—There was no production of asphaltum in Texas in 1898, the mines near Cline, Uvalde County, remaining idle. However, a new company, called the Uvalde Asphalt Co., was organized in New York to take over the property belonging to the Litho-Carbon Rubber Co., which had previously come to grief, and operations were begun by the new company early in 1899. The mines are situated about 6 miles south of the main line of the Southern Pacific Railway, with which they are connected by a branch. The discovery of high grade asphaltum was reported near Sourlake. A small production of asphaltic stone is made in Montague County by the St. Jo Asphaltum Co.

Utah.—Some asphaltic limestone was produced at Pariette by the Assyrian Asphalt Co. The deposit near Thistle, in Utah County, was not worked. Some developments were made in other deposits of asphaltic limestone, but their output was unimportant. No bituminous sandstone was dug in Utah in 1898, although there are deposits of this rock. However, the conditions of transportation practically limit the market for the bituminous limestone and sandstone of Utah to the requirements of Salt Lake City and other towns in the State, although shipments have been made so far east as Chicago.

ASPHALTUM IN FOREIGN COUNTRIES.

Outside of the United States the principal producers of asphaltum and asphaltic stone are France, Germany, Russia and the Island of Trinidad. These statistics are given in the subjoined table. However, it should be remembered that these statements of quantity are not properly comparable, since they represent products of widely different tenor in bitumen, and many of them are used in their crude state, without undergoing any refining process.

THE WORLD'S PRODUCTION OF ASPHALTUM AND ASPHALTIC ROCK. (*a*) (IN METRIC TONS.)

	Austria.	France.	Germany	Hungary.	Italy.	Russia.	Spain.	Trinidad. (*f*)	United States.		Venezuela (Bermudez.) (*f*)
	Asph. Rock.	Asph. Limestone. (*b*)	Asphaltum.	Mineral Pitch.	Asph. Rock.	Asphalt. (*d*)	Asph. Rock.		Asphaltum. (*g*)	Asphaltic Rock. (*h*)	
1893	88	24,079	47,238	(*c*)	25,500	15,925	820	96,625	8,301	28,489	1,771
1894	116	33,274	55,981	2,370	60,000	16,054	985	109,933	10,612	31,018	7,151
1895	404	39,891	59,563	2,284	46,093	18,809	790	92,862	23,156	42,976	3,073
1896	390	34,394	61,552	2,740	44,905	18,188	1,117	100,400	18,519	39,531	6,197
1897	300	30,946	61,645	3,057	54,647	(*e*)	1,656	133,310	24,854	51,670	11,528

(*a*) From the official reports of the respective countries, except where noted to the contrary. The production of grahamite in the United States and Manjak in Barbadoes is not included. (*b*) France produces a large amount of bituminous shales, used for distilling oil, which is not included in these statistics. (*c*) Not stated in the official report. (*d*) The statistics for 1895 and 1896 are official returns furnished by the Department of Mines, St. Petersburg, to C. Le Neve Foster for his Blue Book on Colonial and Foreign Statistics for 1897. (*e*) Not yet reported. (*f*) Exports (crude equivalent) reported by The New Trinidad Lake Asphalt Co. (*g*) Statistics reported by the California State Mineralogist, the entire American product being derived from California. (*h*) Statistics based on direct reports from the producers, including asphaltic limestone and sandstone.

NOTE.—There is a considerable production of asphaltic stone in Switzerland of which no account is taken in the above table, the Swiss government not publishing any mineral statistics.

Austria.—All of the Rothschild asphalt mines in Dalmatia have been acquired by the firm of Ludwig König & Son, of Vienna. The asphalt occurring at Vrgorac is said to be of excellent quality and the deposit of considerable thickness.

Congo.—Asphaltum has been discovered in the lower Congo by Dr. Peschuel-Loesche.

Cuba.—The asphaltum resources of this island, at present unexploited, were described by Raimundo Cabrera,* who said that deposits of asphaltum are abundant and of excellent quality. In the Province of Matanzas, at scarcely a kilometer from the northern coast, there exists a large deposit of bituminous shale. At the Bay of Cardenas, in the Province of Havana, there is a similar deposit. The rocks underlying a large extent of the bottom of this bay contain liquid asphaltum, and some of this product was once exported to New York, where it sold at $80 to $120 per ton. The exploitation of these deposits in the layers at shallow depths was begun with considerable success; but the enterprise languished for want of facilities, and eventually failed entirely through legislative interference inevitable under the Spanish regime.

Asphaltum is found extensively in the provinces of Havana and Pinar del Rio. In the former are the mines of Potosi and Santa Rosa, distant half a league by rail from the harbor of Havana where bitumen occurs embedded in Cretaceous marl. The mineral is earthy and impure, but is advantageously utilized as fuel in grates and ordinary stoves, and is dug on a small scale. In Pinar del Rio are the Canas and Tomasitas mines, close to the Bay of Mariel, which are opened on large veins and produce over 1,300 tons annually for the use of nearby plantations, whose owners combine in working the deposits. Asphaltum is also very abundant in the Vuelta Abajo, where it occurs in surface exudations. There are many small workings, among them that of Santa Helena, at Bahia Honda, which district contains undeveloped deposits of an asphaltum of great purity.

Turkey.—According to R. Helmhacker,† asphaltums of different kinds are found in a number of places in Turkey. The best known deposits are those in the neighborhood of the Dead Sea, but the only deposit worked is one near Salonica, in European Turkey, which is in the hands of a French company. Small quantities are mined in other places, but there is no other production of commercial importance. A large deposit of bitumen exists in Kurdistan, but little is known about it, and it has not been explored.

Trinidad.—The exports of asphalt from the island of Trinidad are given in the following tables, for which we are indebted to the courtesy of Mr. O. B. Thurber, of the New Trinidad Lake Asphalt Co., Ltd., who explains the falling off in 1898 by the fact that most of its customers in the United States had imported very heavily in the early part of 1897 in anticipation of the Dingley Tariff act, in consequence of which they had large stocks of material on hand for the season of 1898. The remarkable increase in the production in 1897 is accounted for by the same reason.

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* *Engineering and Mining Journal*, Sept. 10, 1898.

† R. Helmhacker, *Engineering and Mining Journal*, Nov. 26, 1898.

EXPORTS OF LAND ASPHALT FROM TRINIDAD, IN TONS OF 2,240 LB.

Year.	To United States.			To Europe.			To Other Countries.			Grand Total of Exports in Crude Equivalent.
	Crude.	Épuré.	Total Equivalent in Crude.	Crude.	Épuré.	Total Equivalent in Crude.	Crude.	Épuré.	Total Equivalent in Crude.	
	Tons.	Tons.	Tons.	Tons.	Tons.	Tons.	Tons.	Tons.	Tons.	Tons.
1887....	1,195	2,100	4,345	220	....	220	....	....	....	4,565
1888....	5,316	1,536	7,620	619	....	619	....	....	....	8,239
1889....	10,490	2,052	13,568	....	....	....	833	....	833	14,401
1890....	15,406	1,341	17,417	....	....	....	....	....	....	17,417
1891....	20,507	7	20,517	139	....	139	40	....	40	20,696
1892....	17,406	....	17,406	699	....	699	....	....	....	18,105
1893....	3,450	....	3,450	2,432	1,862	5,225	110	178	377	9,052
1894....	3,365	325	3,853	2,200	4,699	9,249	13	94	154	13,256
1895....	4,445	199	4,744	1,770	2,368	5,322	....	169	254	10,320
1896....	11,943	71	12,049	842	1,988	3,824	....	....	....	15,873
1897....	19,243	*Nil.*	19.243	293	700	1,343	415	178	682	21,268
1898....	18,160	*Nil.*	18,160	700	258	1,087	404	312	872	20,119

EXPORTS OF PITCH-LAKE ASPHALT FROM TRINIDAD, IN TONS OF 2,240 LB.

Year.	To United States.			To Europe.			To Other Countries.			Grand Total of Exports in Crude Equivalent.
	Crude.	Dried.	Total Equivalent in Crude.	Crude.	Épuré and Dried.	Total Equivalent in Crude.	Crude.	Épuré and Dried.	To'al Equivalent in Crude.	
	Tons.	Tons.	Tons.	Tons.	Tons.	Tons.	Tons.	Tons.	Tons.	Tons.
1887....	21,915	....	21,915	10,205	5,771	18,861	....	....	....	40,776
1888....	24,321	....	24,321	8,445	8,248	20,817	....	....	....	45,138
1889....	45,410	....	45,410	9,738	9,581	23,750	....	....	....	69,160
1890....	39,907	....	39,907	11,755	9,951	26,681	668	....	668	67,256
1891....	52,510	....	52,510	9,984	9,969	24,937	901	....	901	78,348
1892....	70,806	....	70,806	11,596	9,458	25,783	1,076	....	1,076	97,665
1893....	65,436	....	65,436	10,640	6,650	20.615	....	....	....	86,051
1894....	71,860	....	71,860	8,967	9,413	23,086	....	....	....	94,946
1895....	61,702	2,256	64,976	5,058	7,365	16,104	....	....	....	81,080
1896....	60,637	....	60,637	8,320	8,052	20,391	....	1,300	1,918	82,946
1897....	71,969	1,769	74,407	14,629	13,510	34,856	....	500	680	109,943
1898....	46,089	1,692	48,424	15,703	13,228	35,537	693	1,646	2,999	86,960

Venezuela.—Mr. O. B. Thurber, of the New Trinidad Lake Asphalt Co., reports to us the shipments of asphalt from the State of Bermudez, as follows: Previous to 1891, 41 tons of 2,240 lb.; 1891, 250; 1892, 1,131; 1893, 1,743; 1894, 7,038; 1895, 3,025; 1896, 6,099; 1897, 11,346; 1898, *nil.* All of this material has come into the United States with the exception of 950 tons in 1893 which was lost at sea.

Literature.—Among the important literature of the year should be mentioned a paper by Clifford Richardson on "The Nature and Origin of Asphalt" in the *Journal of the Society of Chemical Industry*, January 31, 1898, describing the Trinidad and Bermudez deposits and the methods of asphaltum analyses; S. F. Peckham, "On the Technical Analysis of Asphaltum," *idem*, May 31, 1898; S. F. Peckham, "Notes Upon Asphalt and Asphaltum," and H. Endemann "On Asphalt," *idem*, November 30, 1898. For an exhaustive account of the present methods of asphaltum refining, reference should be made to the paper by H. A. Frasch further on in this volume.

THE HIGH GRADES OF GLANCE PITCH; THEIR OCCURRENCE, PROPERTIES, PRODUCTION AND USES.

Under this caption we have grouped for consideration various high grade, solid hydrocarbons which are used in the arts for similar purposes, although chemically and mineralogically they present many dissimilarities. These may be enumerated as albertite, grahamite (gilsonite), uintaite, wurtzilite, ozokerite and manjak. It will undoubtedly help to an understanding of these useful varieties of mineral pitch and wax by a summary of their mineralogical properties which is based chiefly on Dana's *System of Mineralogy*, sixth edition, 1896, and various papers of the American Institute of Mining Engineers.[*]

The mineralogical classification of these substances is involved in much difficulty owing to the complex and variable nature of some of them. Ozokerite is recognized by Dana as a distinct mineral type of the simple hydrocarbon group as distinguished from the oxygenated hydrocarbon group, the mineral succinite (amber) being a type of the latter. He considers that petroleum, passing into viscous bitumen (maltha), asphaltum (solid bitumen), and mineral coal and the numerous varieties of these substances, has less claim to be regarded as a distinct mineral species. These substances pass gradually from one into another without any sharp dividing line. For example there are certain products in California as to which it is difficult to pronounce whether they partake more of the character of petroleum or maltha, while at the other end of the series there are substances resembling both coal and glance pitch which cannot be identified with certainty. With respect to the last Dr. Watson Smith recently described[†] a lignite from Central America which was mostly soluble in benzine and in many respects resembled uintaite from Utah.

ALBERTITE differs from ordinary asphaltum in being only partly soluble in oil of turpentine and in its very imperfect fusion when heated. It emits a bituminous odor, and when rubbed becomes electric. It is brittle, with a conchoidal fracture. It is 1 to 2 in hardness and 1·097 sp. gr. Its luster is brilliant and pitchlike; its color jet black, and its streak black to brownish black. It softens a little in boiling water and intumesces in the flame of the spirit lamp, emitting gas, but it does not melt like asphalt. In the closed tube, however, it can be melted with some intumescence. It is soluble to the extent of 4% in ether, and to the extent of 30% in oil of turpentine, but is practically insoluble in alcohol. Its composition is carbon 86·04%, hydrogen, 8·96; oxygen, 1·97; nitrogen, 2·93; sulphur, trace; ash, 0·10—total, 100%. Another analysis showed the mineral to contain 0·4% water, 57·2% volatile combustible matter and 42·4% coke, the coke itself having 0·27% ash. This analysis showed it to be of about the same composition as jet from Whitby, a specimen of which gave 1·5% water, 57·1% volatile combustible matter, and 41·4% coke, the last with 4% ash. The proper classification of albertite has always been a disputed point, certain mineralogists considering it to be a variety of asphaltum

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[*] *Journal Society of Chemical Industry*, Nov. 30, 1898.

[†] Reference has been made especially to "Asphalt and its Uses," by F. V. Greene, *Transactions of American Institute of Mining Engineers*, Vol. XVII., and "Uintaite, Albertite, Grahamite and Asphaltum," by W. P. Blake, *idem*, Vol. XVIII.

while others have held it to be a species of bituminous coal, placing it at the head of the series of pitch coals or jets. Albertite was once mined extensively in New Brunswick, and for an account of the geological occurrence of the mineral and its industrial exploitation reference should be made to a recent paper by John Rutherford, "Notes on the Albertite of New Brunswick."*

GRAHAMITE resembles albertite in its pitch-black lustrous appearance, but is distinguished from it by its action with various solvents. It is brittle and has a subconchoidal fracture. Its streak and powder are a dark chocolate brown. It is rated at 2 in the scale of hardness and has a sp. gr. of 1·145. Heated to above 400° F. it softens so that it can be molded and drawn out into threads, but it melts only imperfectly and with a decomposition of the surface. On heating it decrepitates and behaves very much like a caking coal.

It is dissolved rapidly by chloroform and carbon bisulphide; it is slowly but mostly dissolved by oil of turpentine and coal tar benzol, and partially by ether, naphtha, benzine and gasoline. It is insoluble in alcohol and is unacted upon by caustic alkalies or nitric or chlorhydric acid. Strong sulphuric acid is colored brown by it.

The identification of grahamite has been as uncertain as that of albertite. Some mineralogists, indeed, have claimed it to be identical with albertite, while others have held it to be a mineral pitch or inspissated petroleum. Dr. Henry Wurtz pronounced it to be neither coal, asphaltum nor albertite, and this view is perhaps held most generally. Grahamite was found in Ritchie County, W. Va., in a vein which is now exhausted. The origin and occurrence of the mineral was described recently by I. C. White,† who referred to it as a black, coaly mineral looking like anthracite, and very closely allied to the albertite of New Brunswick. It consists essentially of 79% carbon, 6·5% hydrogen and 14% oxygen. In Ritchie County it occurred in a vertical fissure at a point about 7 miles from the ridge known as the Oilbreak. This fissure attained a maximum width of 5 ft. The mineral was mined as a fuel, but its exploitation was never profitable. Within a year well-paying oil wells have been put down within a few hundred feet of the fissure. The theory of origin is that the crack, which was produced by the forces causing the anticline, of which the Oilbreak is the top, tapped the Pottsville conglomerate oil sand and then the oil flowed until nature plugged up the orifice with the products of the oxidization of the oil, paraffines, tarry products, etc., from the consolidation of which the grahamite resulted. In West Virginia grahamite and its related oxygenated hydrocarbons are considered economic guides to petroleum, if the country is not too much distorted by folds.

Uintaite, or gilsonite, is considered by Dr. Henry Wurtz to be a variety of grahamite, and we have adopted this term. It has a brilliant and lustrous black color and gives a rich brown streak with a shade of red. It is very brittle, showing a conchoidal fracture, and is easily crushed to powder. In hardness it is from 2 to 2·5 and its sp. gr. is from 1·065 to 1·070. It is a non-conductor of electricity, and is electrically excited by friction. It fuses easily

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* *Journal of the Federated Canadian Mining Institute*, Vol. III., 1898.

† Geological Society of America, Eleventh Annual Meeting, December 28-30, 1898.

in the flame of a candle and burns with a brilliant flame, much like sealing wax, and like sealing wax it will take a clean sharp impression from a seal, but unless the melted mineral is very hot it does not adhere to cold paper. It has considerable plasticity while warm and is not sticky, but retains after melting its lustrous black and smooth surface, and can be melted and cooled repeatedly without any apparent change in composition, though doubtless losing some of its volatile matter. Upon distillation a very small quantity of clear white dense oil is driven off and a little gas or vapor. Uintaite dissolves readily in melted wax, ozokerite, spermaceti and stearine; it also dissolves readily in crude petroleum and heavy lubricating oil, but the white distillates from petroleum have little or no effect upon it at ordinary temperature. It dissolves freely in oil of turpentine when warm, but does not dissolve readily in cold spirits of turpentine. It is soluble in ordinary alcohol. Ether does not attack fragments, but slowly dissolves the powder. In composition it is about 80·5% C, 10% H, 3·3% N and 6·2% O.

Grahamite was produced in the United States not only in Utah but also in Colorado, new mines having been opened in the latter State. The production in Utah was made chiefly by the Gilson Asphaltum Co. Some was produced by the Assyrian Asphalt Co. The Uintah Gilsonite Co., a Colorado concern, shipped a few carloads from a mine adjoining that worked by the Gilson Asphaltum Co., but was promptly enjoined by the latter on the ground that the property really belonged to it. Some gilsonite was also mined near Vernal, on the White River, Utah. There are numerous veins of grahamite in the Uncompahgre Indian Reservation up to the Colorado line, but their exploitation is not permitted by the government. Two veins of grahamite just over the Colorado line were opened and exploited in a small way.

The Castle Peak gilsonite mine, in Wasatch County, near the Uintah Indian Reservation, 75 miles north of Price (the nearest railway point), was sold to Denver men, who propose to develop it extensively. The Raven Mining Co., of St. Louis and Chicago, was granted a leasehold of asphalt and gilsonite land in Southern Utah by the Secretary of the Interior, December 2, 1898. This lease gives the company the right to prospect over a territory comprising perhaps 250,000 acres in the Uintah Indian Reservation, south of the Strawberry River, but within a specified time it must file a map showing the deposits it wishes to work. The Chicago Varnish Co. is said to be interested in the Raven Mining Co.

The newly opened deposits of grahamite in the Middle Park, Colo., consist of small fissures in the Cretaceous formation extending along the 106th meridian, north of latitude 40. The elevation of the country is about 10,500 ft. The veins, which dip from 20° to 40°, vary in size from mere seams to bulges 3 ft. between walls, and are apparently of lenticular form. Very little was accomplished in their exploration in 1898, but so far as ascertained the product consists of two kinds, namely, a hard brownish substance, tough and granular in structure, with a smooth fracture, readily fusible in an ordinary flame and soluble in cold benzol. This is found from the outcrop down to a depth of 50 ft., at which point the vein pinches. The other product, which is found in the swells of the vein, is a brittle mineral, showing a conchoidal fracture and brown streak, resembling the uintaite of Utah.

These deposits are situated 6 miles from the nearest railway point, and for the present at least can be worked only during the summer months, if they turn out to be worth working at all. About 115 tons of 2,000 lb. were shipped in 1898, which was marketed in Chicago at $40 per ton, delivered there. The cost of mining and sacking was $16 per ton, carriage to railway $18, and railway freight $12·50. It is extremely doubtful if the mines can be exploited successfully until cheaper means of transportation can be secured. The Middle Park mines are owned by the Colorado Gilsonite Co.

The chief markets for gilsonite are St. Louis and Chicago, where the first-class mineral is worth about $45 to $50 per ton, costing the producer about $35 per ton laid down there. It can be mined cheaply, but the dust in the working is very annoying to the miner, and when mixed with air is explosive. A serious accident occurred on this account in one of the mines in Utah a few years ago. It is used chiefly for making varnish for japanning, 1 lb. of gilsonite being heated with about five pints of turpentine.

William C. Day is said to have prepared an artificial product closely resembling gilsonite (grahamite) in appearance and properties, by distillation of a mixture of fish and wood.

MANJAK is a bitumen from Barbadoes, an island in the West Indies, which has only recently become an article of commercial importance, and, so far as we are aware, has not yet been identified with any of the recognized types. It is a mineral of black color, high luster, and bright conchoidal fracture, in appearance closely resembling freshly broken pitch. Its sp. gr., according to W. Merivale, is 1·123, and melting point 420° F., as compared with 1·139 and 360° F. for Trinidad glance pitch. It is composed of 97% of matter soluble in carbon bisulphide, and 2·32% ash and 0·68% insoluble organic matter, against 88%, 7·4%, and 4·58% respectively for Trinidad glance pitch.

This mineral occurs in Barbadoes in veins varying between 0·25 in. and 30 ft. in thickness in a country rock varying from a soft clay to a hard shale. In general this rock is blue where soft and white where hard. Its joints are often filled with liquid bitumen, and in places this has saturated the entire rock over extensive areas, producing a shale from which as much as 37 gal. of oil per ton have been obtained by destructive distillation in the laboratory. Sometimes the mineral occurs in a simple vein, and at other times it is contained in a lode of soft clayey material. The veins and lodes generally cut the country rock across the lines of stratification, following the direction of some natural fissure. In some veins horses are frequent, but in other cases the veins are unbroken, one having been followed for 200 ft. on its strike, 100 ft. in depth, and 9 ft. in width without interruption.

The Merivale mines, which are the most important, have been exploited for three years. The method of mining was described in the *Engineering and Mining Journal* of December 31, 1898. The mineral is so friable that it can be ground to powder between finger and thumb, and for this reason cannot be handled like coal, but must be carefully sacked in the pit. The bags hoisted out are taken to the sorting room, where boys are employed to pick out by hand any small bits of dirt and rock. This sorting is very important, since the consumer will not pick it out himself, and demands large reductions in price if a speck of foreign matter is found in his consignment. Up to the present time the Merivale mines are the only ones which supply the high grade manjak, which is used for varnishes and paints. The Emptage and Howe mines produce a lower grade of mineral, which sells more cheaply, and is used for car roofings and insulating purposes.

OZOKERITE is a mineral of the simple hydrocarbon group, which consists chiefly of members of the paraffine series. In appearance and consistency it resembles wax or spermaceti. When pure it is colorless to white, often leak green, yellowish, brownish yellow and brown; translucent; and greasy to the touch. Its sp. gr. is 0·85 to 0·90 and fusing point 56° to 63° C. It is soluble in ether, carbon bisulphide, oil of turpentine, benzine and naphtha, and slightly soluble in boiling alcohol. The variety obtained at Boryslaw, Galicia, is blackish brown in color and has a sp. gr. of 0·944 and melts at 60° C. Its composition is about 85·5% C and 14·5% H.

United States.—There was no production in the United States in 1898, and under existing conditions (namely, the cheaper supply available from Galicia) it does not seem probable that this industry will be revived. A new discovery was reported in Utah of mineral which assayed paraffine 86·5%, asphaltum 8·5%, earthy matter 2·5% and water 1·5%.

Austria-Hungary.—The exports of crude ozokerite from Austria-Hungary during the first 10 months of 1898 were 3,619·8 metric tons, against 3,929·7 in the corresponding period of 1897. The exports of cerasin were 1,147 and 1,081·7 metric tons respectively. The imports were: Ozokerite, 25 (23); cerasin, 31 (40). The figures in brackets are for 1897. The ukase forbidding the use of cerasin candles in the churches in Russia has been repealed, to the great advantage of the Russian cerasin industry. Austria has now to take a second place to Germany in the export trade in cerasin, owing largely to the Austrian Land Bank monopolizing the sale of ozokerite and considerably raising its price.

According to A. Lukaszevski* between 6,000 and 7,000 shafts have been sunk at Boryslaw, in Galicia, during the last 30 years, to an average depth of 100 yards. The shafts are of small section (3 × 4 ft.) and short working levels (5 to 8 yards long) are driven direct from the shaft at various depths. Winding has always been a difficulty, owing to the small section of the shafts, the unsatisfactory mode of timbering, and the irrational, unsystematic method of mining. Hitherto hand-windlasses have been used for hoisting, buckets with a capacity of 140 lb. being used. The buckets themselves weigh 24 lb. For winding at shafts 100 yards deep, five men are required; at deeper shafts, seven men. The output per 12-hour shift is dependent on the depth and condition of the shaft. In the case of shafts under 100 yards in depth it amounts to 100 to 130 buckets, while in shafts of 170 yards in depth it does not exceed 50 to 60 buckets. The French Commercial Co., which is now exploiting mines

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* *Oesterreichische Zeitschrift für Berg- und Hüttenwesen*, Vol. XLVI., pp. 133-137.

at Boryslaw, has lately installed an electric plant for hoisting, ventilation, pumping and lighting the surface works.

Moldavia.—In a paper read before the Society of Sciences of Bucharest,[*] C. I. Istrati gave an exhaustive account of the Moldavian ozokerite, which occurs in extensive beds a foot in thickness in Miocene sandstone. When first mined, it is pale yellow and soft, but on exposure to the air it becomes heavier, harder and darker in color, the change being due to the loss of volatile hydrocarbons. It contains 84 to 86% of carbon and 14% of hydrogen.

Technology of the Cerasin Industry.—B. Lach[†] describes the recent improvement in this industry, among which the introduction of an emulsifier is of special interest. This consists essentially of two flat plates revolving in opposite directions, and between which the hot ozokerite, mixed with the requisite quantity of acid, is introduced by means of a central opening in the axis of the upper plate. It is stated that a more thorough admixture can be effected in this way than by any other, and there is a considerable saving of acid and an accelerated output. Many attempts have been made to discard sulphuric acid altogether in the preparation of cerasin, but these have speedily been abandoned. By extracting the ozokerite with benzine after treatment with bleaching powder, lycopodium powder and magnesium silicate, there is an increased yield (up to 96%), but the product is inferior, being greasy and often having an unpleasant smell. A method of increasing the yield, but little known, consists in adding to the ozokerite from 5 to 10% of a heavy mineral oil. This gives a more wax-like character to the cerasin, but the addition is objectionable when a white product is required. Attempts to extract the filter-press residue in the filter-press itself, instead of in an extractor, have not given satisfactory results, owing to loss of solvent, tediousness, and danger of manipulation, and the necessity of having filter-presses of special construction.

WURTZILITE is a solid black mineral, with brilliant conchoidal fracture, and a general resemblance to jet or some of the cannel coals, which is found in the Uinta Mountains in Wasatch County, Utah, not far from the source of the grahamite, or uintaite, previously described. It is sectile and the shavings are somewhat elastic, but if bent too far or too suddenly while cold they snap like glass; but when slowly pressed and warmed a flake may be bent nearly double. The mineral in very thin plates is of a deep red color, but by reflected light it is jet black. Its hardness varies from 2 to 3, and its sp. gr. is 1·03. It does not melt in boiling water, but becomes softer, tougher and more plastic. In the flame of a candle it melts, takes fire and burns with a bright luminous flame, with very little smoke, giving off a strong bituminous odor. Fused in a glass tube it emits a dense cloud of white and yellow smoke and a distillate of a thick brown tarry oil with a strong odor, leaving a small residue of fixed carbon. Fragments warmed in the hand emit a strong odor, rather offensive, like that of some of the crude petroleums. Wurtzilite resists the usual solvents of bitumen.

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[*] *Journal of the Iron and Steel Institute*, No. 1, 1898, p. 398.

[†] *Chemiker Zeitung*, 1897, XXI., [88], pp. 916–917.

ASPHALTUM REFINING.

BY HANS A. FRASCH.

THE occurrence and nature of asphaltum have been described in previous volumes of THE MINERAL INDUSTRY,* but it remains to treat of the methods of refining this substance. These vary with the nature of the crude mineral, and nearly all refining works have devices of their own to accomplish their purpose. It appears as if the apparent simplicity of refining asphalt has retarded the development of new and better methods, and only within the last few years has it been realized that the successful application of asphalt depends greatly upon the method of refining. When it is considered that enormous quantities of asphalt are consumed in paving, mostly in places far removed from the source of production, so that the matter of transportation figures prominently, it is evident that methods of refining which result in a product of only 60% purity will soon become a thing of the past.

In refining asphalt it must be borne in mind that the purity of the product is important, both as to the cost of transportation and its covering capacity; since the richer an asphaltum the greater its spreading capacity, and the cheaper the packing and transportation. The relations of petrolene and asphaltene and their nature must also be considered, since they govern the quality and physical properties of the asphalt. For the paving industry especially the nature of the petrolene is of importance. This is not a fixed body, but a mixture of various hydrocarbons, each having individual properties, whose aggregate composition may show the required formula of what is called petrolene and yet differ sufficiently in physical properties to render an asphalt unfit for paving purposes. The same may be said as to asphaltene.

Few of the methods employed for refining asphalt allow control of the composition of the product, and the fluctuations and shortcomings in quality of a commercial asphalt are generally adjusted by the consumer adding to his mixture more or less flux, such as mineral oil, maltha, and similar substitutes. The proper place, however, to provide for regularity in quality should be the refinery. The refining of asphalt is accomplished according to its physical properties.

Refining with Boiling Water.—Asphalt of low melting point and of specific gravity lighter than water, such as mineral tar or pitch found in asphaltic sandstone, is extracted by boiling the asphalt-bearing rock in water. This is one of the oldest methods, and is practiced on the continent of Europe in the production of "Goudron mineral." It is carried out in the following manner:

The crude rock is crushed and deposited in iron kettles which previously have been about half-filled with water. These kettles are arranged in batteries of four or six in such manner that the contents of any or all of them can be brought to boiling (see Figs. 1–4). As soon as the temperature of the water and rock reach the melting point of the asphalt, the latter separates from the rock and floats to the surface of the water, whence it is collected. The mineral tar thus obtained contains water and earthy substances which must be removed, and to accomplish this the impure bitumen is again heated in proper vessels until the water is evaporated, and the larger part of the impurities separate and settle, when the bitumen

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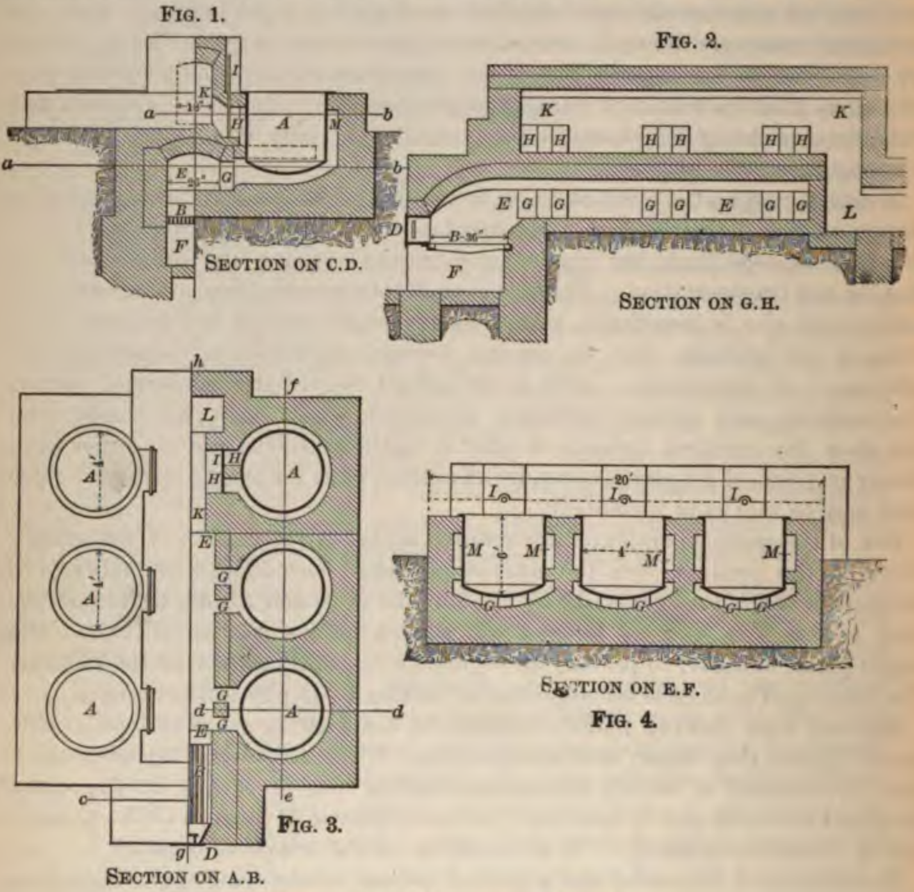
fer especially to Vols. I., II., IV., V., VI.

is cast into barrels. The residuum of the second operation is utilized in the manufacture of mastic blocks. The apparatus employed for this boiling process consists of the kettles *A* (Figs. 1–4), which are 3 to 4 ft. in diameter and about the same in depth. They are arranged in two rows of three kettles each on either side of the flue *E* and grate *B*. The fire-gases from *B* pass through the flue *E* and openings *G* below the kettles *A*, through the flues *M* and the openings *H* to the flue *K*, which is directly above the flue *E*. From *K* the gases pass through *L* to the chimney. The openings *H* can be closed by means of dampers *I*, and thereby the temperature of any of the kettles can be regulated.



Figs. 1–4.—The Goudron Furnace.

While this apparatus is rather antiquated, it forms the basis of a number of modern furnaces, with but slight modifications of the original. In some instances the open fire has given way to steam heat, and calcium chloride or sodium chloride (common salt) solutions are used in place of water to accomplish quicker separation on account of the greater difference of specific gravity between the bitumen and the solution. The higher boiling point of the salt solution also facilitates the separation and renders the process applicable to bitumen of higher melting point and greater specific gravity.

Refining by Liquation.—The heavier asphaltum, or asphaltum proper, such as occurs in the West Indies, Trinidad, etc., carrying from 40 to 60% of bitumen; cannot be refined by the boiling process on account of its high specific gravity and because of the nature and occurrence of the impurities, which are mixed through the asphaltum in a state of so fine division that they cannot be separated without effecting a solution of the asphaltum. The Syrian and Egyptian asphalt occurs so pure as not to require treatment, and this class of asphalt is principally consumed by the paint and varnish trade. The West Indian and Trinidad product, however, containing earthy impurities and water, must be refined. This is accomplished in a crude way by melting the asphaltum and keeping it at a temperature sufficiently high to evaporate the water and light hydrocarbons, while at the same time part of the heavier impurities settle out. In cases where the asphalt contains a large amount of coarse earthy admixture the refining is done in two operations. The asphaltum is first melted and left to rest for several days in a melted state without raising the temperature high enough to cause it to boil, then the asphalt is drawn from the impurities and in separate vessels concentrated by boiling.

The refining is done in batteries arranged similar to the Goudron furnace above described; the vessels, however, are of larger capacity. Square or semi-circular tanks constructed of boiler iron are generally used. They are arranged in rows of four to six, each having its own fireplace. In some instances the open fires have been abandoned and steam coils, with steam at 60 to 80 lb. pressure, are used for the heating. A battery of four semi-circular steam-jacketed pans set in terrace form, so as to permit the melted asphalt to flow from the uppermost pan into the next lower, etc., does very efficient work. In cases where the consistency of the asphalt does not permit the coarser impurities to settle, liquid asphalt or petroleum residuum is added to obtain the desired consistency.

The California asphalts are refined by the same method as the Trinidad. The crude asphalt as mined is richer in bitumen than the Trinidad, but not of so favorable a composition. It contains on an average 85% bitumen, 9% infusorial earth, and 6% water. In refining it is heated to about 350° F. until the water is evaporated, and liquid asphalt is used for fluxing. The latter is obtained by driving off the water and volatile matter from maltha.

Numerous improvements for the refining of this class of asphaltum have been suggested, but few appear to have been of sufficient importance to be generally adopted. The fact that the most important of the asphalts, the Trinidad product, gives satisfactory results though raised to a purity of only about 60%, seems to render efforts toward more complete refining unnecessary. While this may apply to an asphalt which in the native state happens to have the proper composition, it prevents the utilization of other deposits, which, while faulty in their crude state, might be rendered valuable by judicious treatment.

R. Alexander suggests passing a current of air through the heated asphalt to assist the evaporation of water and undesirable light oils. This appears, however, applicable only to asphalt of excessive contents of petrolene, and more especially if the latter contains constituents which are readily volatilized and consequently would render the product unfit for pavement. It is established that certain asphalt, for instance the liquid asphalt of lower Mexico, although having an appearance of permanency at a temperature of 280° F., will evaporate almost entirely at a much lower temperature if exposed to the action of air in presence of a capillary substance. No doubt this is due to the consistency of the asphalt and the low tension of the vapors of the constituents of the petrolene, which do not allow the vapors to overcome the tenacity of the asphalt in a body. Such a product, of course, is unfit for technical purposes without chemical treatment. It will be found in nearly every instance that asphalt, while constant if heated to a certain degree in a larger body, will show volatilization if heated in presence of a capillary substance and air. For this reason the method suggested by Alexander appears advantageous in removing undesirable products, while an asphaltum which would stand the treatment must necessarily be of good quality.

Miscellaneous Processes.—Dubbs claims[*] to produce asphalt from petroleum refuse or liquid asphalt (maltha) by subjecting it to the action of sulphur at a temperature above 250° F., the sulphuretted hydrogen, which is evolved in the process, being utilized for the manufacture of sulphuric acid. His attempt seems to be to convert petrolene into asphaltene.

It is reported that the Alcatraz Asphaltum Co. of California is now conducting experiments on a large scale to refine asphalt by the naphtha process, but so far no definite success is known to have been attained.

Bayerly treats petroleum refuse with air at a high temperature and obtains an asphaltic product which finds use in the varnish trade.

Rock Asphalt.—Rock asphalt, and more especially asphaltic limestone, is beyond doubt the most suitable for paving purposes, and the refined asphalt from calcareous deposits is also superior for technical purposes. Asphaltic limestone is generally used for pavements on the European continent. The reason why it has not been adopted in this country appears to be the high cost of transportation. Within the last few years extensive deposits of asphaltic limestone have been opened in different parts of the United States, but little of it has been used.

The Utah deposits are being worked, and to some extent the asphalt is employed in Western cities for pavements. The pure product, known as grahamite, or gilsonite,[†] mostly finds use in the varnish trade. The Indian Territory deposits are being operated by a strong company and their production is increasing. The Cherokee deposits are mostly of a sandstone nature. The asphaltic limestone of Uvalde County, Texas, promises to be the most important of the American asphalts, and a good deal of money has been expended in developing these deposits, but it appears that the object hitherto was more speculative than industrial. The Texas rock compares favorably with any of the European rocks, the bitumen contained in it being of excellent quality and especially adapted for paving purposes on account of its great tenacity and durability. Excellent pavements have been constructed from this rock at San Antonio and Houston, Tex. An attempt to pave with it in Troy, N. Y., failed chiefly for the reason that the rock alone was used, without flux or any other admixture; nor

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[*] U. S. Patent No. 606,142.

[†] The nomenclature of this substance is involved in confusion, "gilsonite," "grahamite," and "wurtzilite," being commonly used, while nearly similar products are known as "albertite" and "manjak." Uniformity is desirable. In this volume preference is given to the word "grahamite," although "gilsonite" seems to be most used commercially, at least for the Utah product.—Ed.

was the pavement compressed properly, while the work was done by entirely inexperienced persons. A good paving mixture is obtained from the Texas rock as follows: 80 lb. powdered asphaltic limestone (15%), 20 lb. ground granite, quartz or sand, 2 lb. petroleum, refuse oil or liquid asphalt. One ton of the Texas rock will lay 10 to 12 yards of pavement. On account of the high cost of transportation it would be more advantageous to extract the bitumen.

Extraction by Liquation.—For the recovery of bitumen from asphaltic rock several methods are employed. The oldest process consists in subjecting the rock to a temperature above the melting point of the bitumen and collecting whatever drains from the heated rock. This process is carried out in reverberatory furnaces, and yields from 25 to 75% of the asphalt, according to the density of the rock. When the mineral substance exists in amorphous form, capillary action prevents the application of this process. The asphalt obtained by the direct heating process contains mechanically intermixed impurities, and besides more or less of the petrolene is destroyed, so that it is very difficult to obtain a uniform product. Better by far are the methods which extract the asphalt by means of a solvent. Benzine or petroleum naphtha and carbon bisulphide are used for solvent agents.

Extraction by Means of Naphtha.—In selecting a solvent, however, to extract asphalt, it should be considered that the solubility of both petrolene and asphaltene is unlimited in carbon bisulphide, while petroleum naphtha will only dissolve a fixed amount of asphaltene at a given temperature, though it dissolves petrolene in all proportions. This property of petroleum naphtha offers excellent means to control the constituents of asphaltum, and in the process of extraction to adjust the desired proportions of asphaltene and petrolene to a nicety.

An apparatus for the extraction of asphalt by means of naphtha is shown in Figs. 5–7.* It consists of a series of extracting vessels constructed of boiler iron, each vessel being surrounded by a steam jacket. Six to eight extractors are connected with each other in such a manner that the naphtha solvent can flow from any vessel to and through the next in series. The vapors arising from each extractor are conducted to a central condenser located at a point of elevation which allows the condensed solvent to return to any of the vessels by gravity, so that there exists, when the battery is in operation, not only a constant current of solvent from the first to the last vessel, but also a continuous circulation of the solvent by evaporation and return of the condensed solvent to the first vessel of the series.

The apparatus consists of the extractors 1–6 which are 6 ft. in diameter and 14 to 16 ft. high. They are provided with flanged openings, *b* and *c*, which serve for charging and withdrawing the rock. Furthermore, each extractor is provided with flanges for the inlet and overflow pipes *n* and *p*, the vapor pipes *l* and *i*, and the solvent return pipe *g*. A 1-in. pipe supplies the steam to the jacket, which is connected with a steam trap. The outlet *h* serves to draw off the solvent after exhaustion of the rock and is connected with the solvent storage tank.

To operate this plant the asphalt rock is crushed and delivered by elevator or tramway to the top of the extractors and all of the extractors are filled with the rock. The naphtha is admitted through pipe *g* and thence

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* U. S. Patent No. 581,546.

FIG. 5.

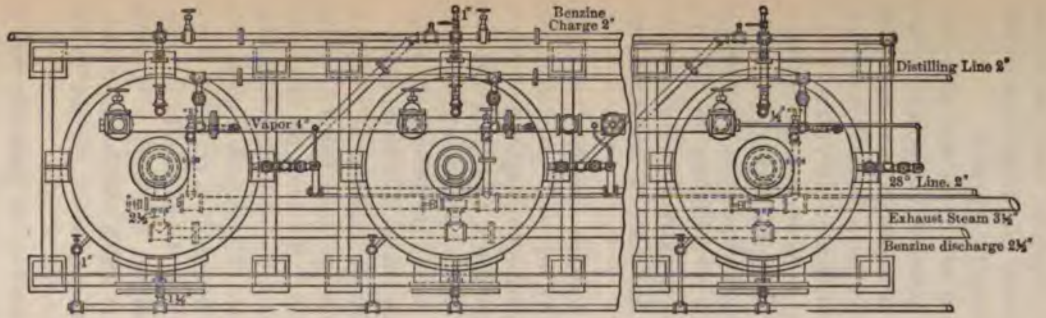


FIG. 6.

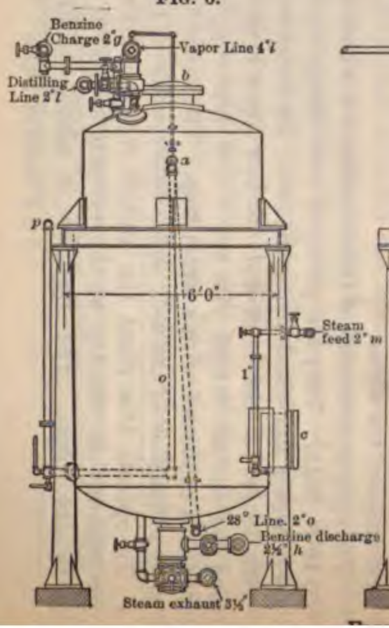
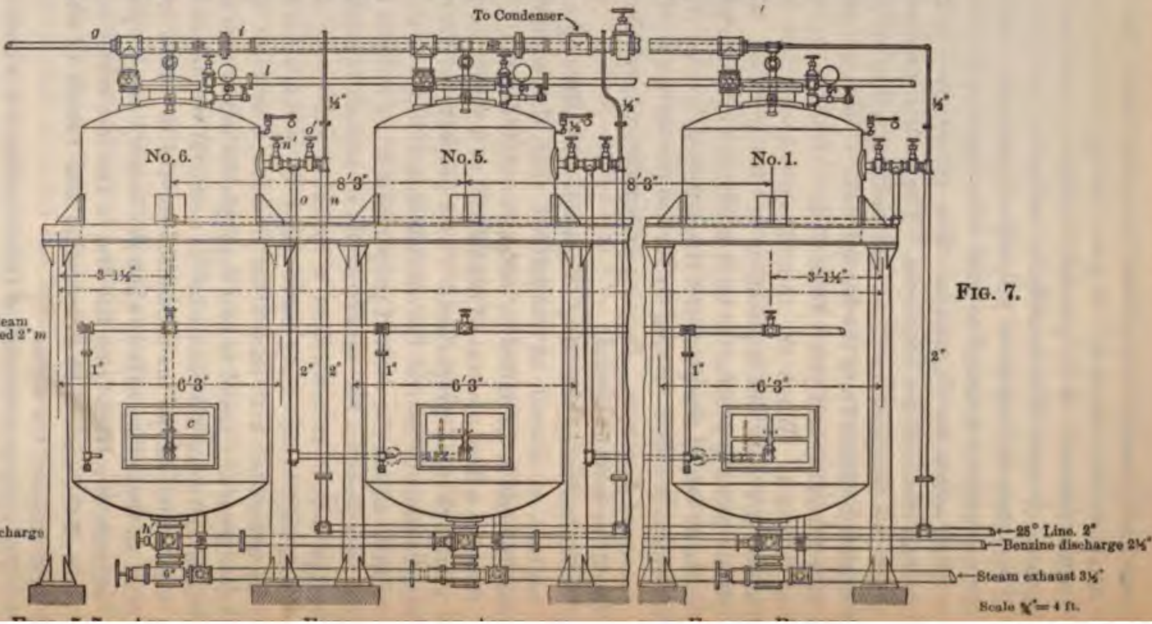


FIG. 7.



passes into the first of the series of extractors (in this instance extractor No. 1), the valves of pipe *g* of all of the extractors being closed against the admission of the solvent through the other branch pipes. The valve *n'* of extractor No. 1 is closed, while the similar valves of the other extractors are opened. The solvent then flows from extractor No. 1 through pipe *o'* and valve *n'* into extractor 2 and from extractor 2 through its pipe *o'* and valve *n'* of extractor 3 to that extractor, and so on from 3 to 4 and 5 and 6, and from 6 back through pipe *p* to the overflow pipe *o* of extractor 1, and the valve *o'* of that pipe being open the solvent charged with the bitumen, taken up by it in its flow through the series of extractors, descends through pipe *n* into the main pipe line *o* which conducts it to storage vessels or suitable distilling apparatus. While the valve *o'* of extractor No. 1 is open the similar valves of the other extractors are closed. The contents of the extractors are brought to the desired temperature by introduction of steam into the steam jackets. The vapors rising from the solvent in the several extractors pass through the pipes *i* to the condenser, whence they are returned through pipe *g* to the initial extractor, thereby causing a constant circulation of the solvent and a continuous flow of bituminous solution from the first to the last of the extractors, though no additional solvent is added after the extractors have once been filled. By this method all the bitumen eventually is carried to the last vessel of the battery and any desired degree of concentration may be managed. The bitumen thereby obtained is absolutely free from impurities, outside of a small percentage of the solvent, which latter is expelled in specially constructed stills. These stills consist of a series of shallow pans each provided with a heating coil and vapor outlet. The pans are arranged above each other so that the hot liquid asphaltum runs in a tortuous stream from each pan to the one next below. About 20 of these pans form one still. Each pan is provided with a vapor outlet connected with a main vapor pipe and condenser. The pans are of ½-in. cast iron 5 × 10 ft. in dimension and flanged on both sides; the bottom extends to within 10 in. from the end and is finished off with a bridge 2 in. high which serves as overflow and maintains 2 in. of asphalt in the pan.

A plant of 16 extractors arranged in two batteries, and eight pan stills will take care of over 100 tons of asphalt rock per 24 hours. The labor required to operate the apparatus proper consists of two men for running extractors, four men to empty and fill extractors (assuming the rock to be delivered by automatic tramway) and two men to take charge of pan-stills. Allowing at an average $1.60 per 12 hours' wages per man, the running of extractors and stills costs $12 per day for 100 tons of rock.

The apparatus if properly conducted will operate with a total loss of less than 2% of the solvent. Each extractor when filled contains 700 gal. of naphtha, so that about 12,000 gal. are in circulation. Allowing a loss of 250 gal. naphtha per day at 8c., the loss of solvent amounts to $20 for the extraction of 100 tons of rock with a daily output of 15 tons of pure asphalt.

The rock outcropping at the surface, as in Texas, can be mined, crushed and delivered to the extracting house for 25c. per ton. The packing of 15 tons refined asphalt requires 80 to 90 bbl. at 30c., or about $25 per day, and four men can take care of the packing house and shipping.

The total running expense of a plant extracting 100 tons of 15% asphalt rock would be per day: Mining, crushing and delivery of rock, $25; labor for extraction and distilling, $12; labor in packing and shipping, $5; package, $25; fuel, including electric light, $25; engineman and assistant, $4; sundry labor, $5; loss of naphtha, $20; superintendence and office expenses, $15; total, $136.

The bitumen extracted by this method in Texas would cost delivered in New York $15 per ton and would lay 120 yards of pavement per ton of 2,000 lb. A paving mastic made with this asphalt consisting of 1,200 lb. powdered lime rock at $1.50 per ton, 90c.; 600 lb. powdered quartz or sand at $1.25 per ton, 38c.; 200 lb. refined asphalt 100% at $15 per ton, $1.50; 10 lb. refuse oil, 16c.; would cost per ton for material, $2.94. This would lay 12 yards of pavement 2½ in. thick so that the cost for material per yard of pavement would amount to 24½c.

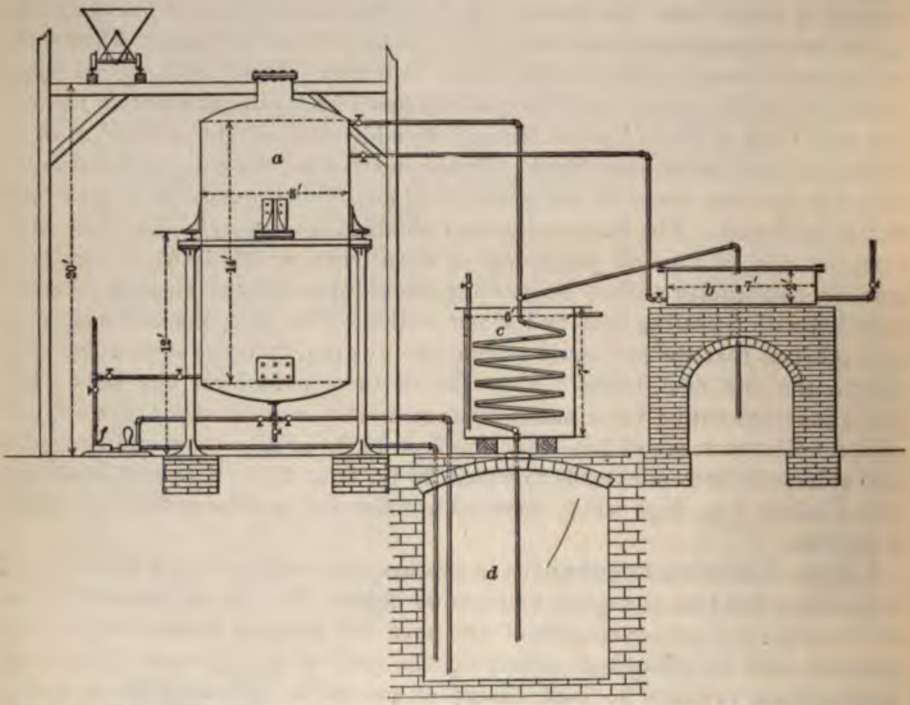
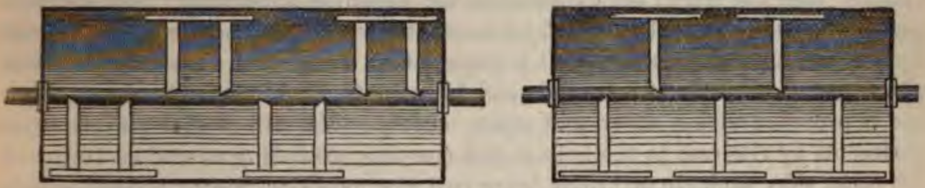


FIG. 8.—APPARATUS FOR EXTRACTION OF ASPHALTUM BY THE CARBON BISULPHIDE PROCESS.

When asphalt rock is used the cost would be (1,600 lb. powdered asphalt rock at $7.75 per ton, $6.20; 400 lb. powdered quartz or sand at $1.25 per ton, 25c.; 40 lb. refuse oil, 64c.), $7.09 per ton of material, or 59c. per yard.

The Carbon Bisulphide Process.—This has the advantage that complete extraction is obtained at ordinary temperature. The extraction also is more rapid than with naphtha, and were it not for its greater volatility and the greater danger of explosion carbon bisulphide might be preferable to naphtha as a solvent for asphalts which are of proper composition in their native state.

The use of carbon bisulphide as solvent requires about the same amount of labor and about 10% less steam, while a plant of the same dimensions will have a larger capacity than when naphtha is used for solvent. It has, however, as stated above the disadvantage that the constituents of the asphalt have to be accepted as they exist and cannot be adjusted as in the naphtha process. Furthermore, the solvent is much more expensive, while the loss of solvent in process will be larger. Carbon bisulphide is capable of combining chemically with products that may be contained in the asphalt, and this may be a source for more or less loss of solvent, according to the composition of the asphalt. Before adopting carbon bisulphide for the solvent the nature of the asphalt should first be carefully determined.



FIGS. 9a AND 9b.—ARRANGEMENT OF MASTIC MIXER.

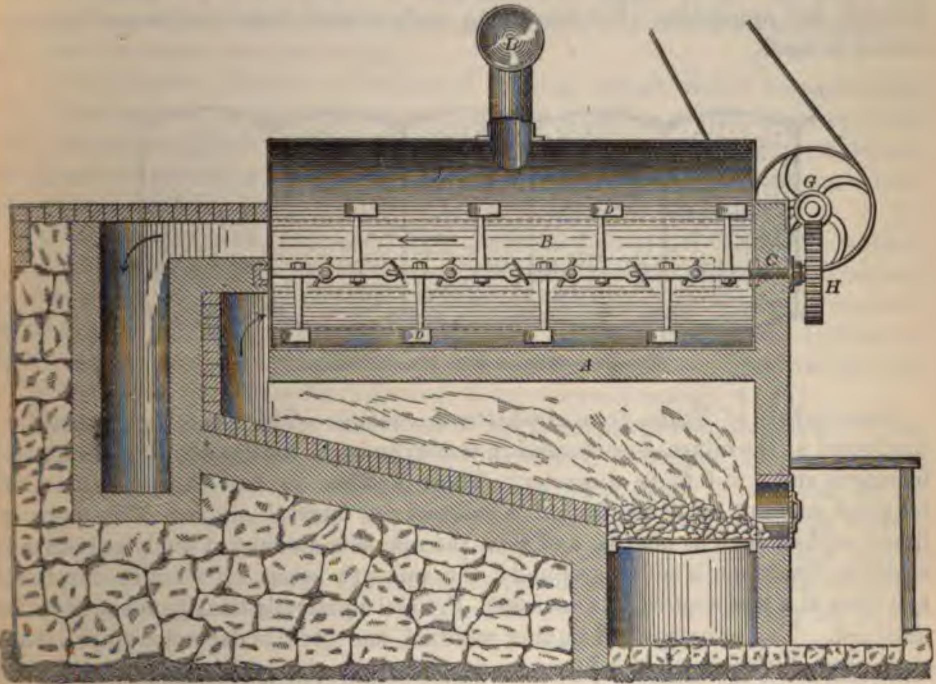


FIG. 10.—MASTIC MIXER.

For the extraction of asphalt from rock with carbon bisulphide the same apparatus may be used as above described for naphtha, only the current of the solvent is reversed and the extraction is conducted from the bottom upward. When naphtha is used the flow in the extractors is from the top downward since naphtha becomes heavier as it is charged with bitumen. Carbon bisulphide becomes lighter in proportion to the amount of bitumen dissolved in it.

For the extraction of not more than 30 tons of rock per day a double extractor system is used as shown in Fig. 8. This consists of the extracting vessels *a*, distilling apparatus *b*, condenser *c*, cistern *d*, and pump *f*. Two extractors are generally used, one being in operation while the other is being emptied and recharged. The extractors hold about five tons each and a charge is exhausted in six hours. They are filled with rock of about 1 in. size, and the carbon bisulphide is forced by a suitable pump into the bottom of the extractor. As the solvent flows through the vessel it dissolves the bitumen and passes through the overflow near the top of the extractor to the distilling apparatus. The solution passing through the still is heated by steam, the vapors are conducted to the condensers and the condensed solvent is returned to the cistern. The still consists of a cast-iron shell about 8 ft. wide and 14 to 16 ft. long. It is about 2 ft. high and is provided with a cover and vapor outlets. The still is divided into compartments by ribs cast to the bottom and one side, extending to within 10 in. of the other side, each rib or bridge being cast alternately to one or the other side of the body of the still so that the asphalt solution flows in a zigzag stream through the apparatus. For conveying carbon bisulphide diaphragm pumps should be used.

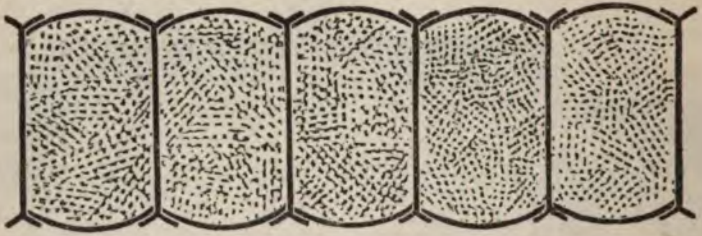


FIG. 11.—MOLDS FOR MASTIC BLOCKS.

Manufacture of Mastic from Asphaltic Limestone.—This operation, generally conducted at the mines, is performed in semicylindrical kettles about 10 to 12 ft. in length and 4 to 6 ft. in diameter. These kettles are first charged with the required amount of "Goudron," a mixture of mineral oil and Trinidad, or liquid asphalt such as maltha, and then the requisite amount of finely pulverized asphaltic limestone is added gradually, the mixture being kept under constant agitation at a temperature of about 300° F. until a perfectly homogeneous mass is obtained. The proportion of fluxing material and rock is dependent upon the amount and nature of the bitumen contained in the rock; the charge when finished should contain from 15 to 20% of bitumen. Val de Travers mastic as a rule contains only 11 to 12% of bitumen, while Limmer mastic contains 24%. The mass should not be too rich, and a dry piece of wood immersed in the hot mixture must remain clean when withdrawn, *i.e.*, no bitumen should adhere to it. The best way is to determine the amount of bitumen by analysis. If it be intended to add sand or gravel for paving mixture the mastic should be made richer in bitumen. On the continent of Europe as a rule the relation of mastic mixture is 1 part flux (Goudron) and 9 parts asphalt rock of 10 to 12% bitumen.

It is important that the powdered asphalt rock should be added in small quantities at a time, as otherwise the bitumen becomes chilled and cannot be mixed properly. Continuous agitation is necessary during the whole of the working process, and this is done mechanically. Each kettle is provided with a shaft running horizontally a fraction above the center-line of the semicylindrical kettles. To this shaft radial cutters or paddles are attached as in Figs. 9, *a* and *b*. The cutters are set diagonally so that they propel the mastic mass to and fro, constantly changing contact and surface of the mixture. The cooking and agitation should be continued for at least an hour after the last portion of the powdered rock has been charged. Care should be taken that the mixture is not heated too fast and at no time should the vapors coming from the mixer be yellow as this would indicate decomposition. Particularly toward the end of the process, after all the rock has been added, the firing must be limited. To prevent chilling the surface of the mixture, the kettles are covered with cone-shaped or semicylindrical covers provided with charging doors and vapor outlets. Fig. 10 shows the arrangement of a mastic mixer.

The mastic when completed is cast into iron forms (Fig. 11) which are arranged on brick floors. Before casting the molds and floor are whitewashed to prevent the mastic from adhering.

Cost of Refining Asphalt.—The cost of refining asphalt cannot be generalized since the facilities vary with the location of the mine, etc., while large amounts can be saved by judicious arrangement of the refinery. Assuming, however, the cost of procuring the crude material to be the same in all instances, the cooking and extraction methods may be compared as follows:

Of an asphalt of the kind occuring in the West Indies and Trinidad, containing an average of 40% bitumen, compared with asphalt rock of 15% bitumen, 5,000 lb. are required to obtain 2,000 lb. of pure asphalt or its equivalent, against 13,333 lb. of asphaltic limestone. The expense of mining and delivery in the refining apparatus being 25c. per ton in either case, the mining cost per ton of pure bitumen would be for Trinidad, 62½c; for rock asphalt, $1.66½.

The refining of asphalt of the nature of the Trinidad product requires: (1) 6% petroleum refuse oil at 8c. per gal. f. o. b. refinery, or 40 gal. per ton of pure bitumen, at a total cost of $3.20. (2) The heating of 5,300 lb. material for 36 hours at a temperature of 300° F. The crude material having originally a temperature of 60°F., 240° have to be supplied. Assuming 100 lb. for a unit a total of 12,720° of heat have to be supplied during a period of 36 hours, which is equal to 457,920 unit degrees in one hour, no allowance being made for latent heat of evaporation. (3) Attendance to apparatus requires one man for 36 hours per 20 tons at $1.60 per day of 12 hours which for 5,000 lb. crude material amounts to 20c. per day and for 36 hours comes to 60c. (4) Packing requires about 6 bbl. per ton of 2,000 lb. Trinidad asphalt when refined containing 60% bitumen, 3,333 lb. have to be barreled to deliver one ton of pure asphalt which take 10 bbl. at 30c. or a total cost of $3.

Asphaltic limestone, containing 15% bitumen, extracted by the naphtha process yields 100% pure asphalt and the manipulating expenses of the process consist of:

(1) Mining, crushing and delivery, 13,333 lb. of rock at 25c. per ton, $1.66.

(2) Loss of naphtha: By the diffusion process 750 gal. petroleum naphtha are

in circulation to extract 2,000 lb. of bitumen of which 2% may be lost or 15 gal. at 8c. equivalent to $1.20. The solution obtained contains about 30% naphtha or 600 lb. per 2,000 lb. of bitumen, in distillation of which another 2% may be lost, or a total of 2 gal. at 8c., 16c.

(3) Heating 13,333 lb. of rock and 5,000 lb. of naphtha for eight hours to a temperature of 180° F. The crude material having a temperature of 60°F., 120° have to be supplied to 18,333 lb. of material for eight hours. At 100 lb. per unit 22,000 unit degrees or 176,000 degrees in one hour are maintained for the extraction of 2,000 lb. bitumen. The asphalt solution consisting of 2,600 lb. takes a temperature 300° F. during a period of eight hours to distil the remaining naphtha, which is an addition of 120°, or 3,120 unit degrees, during eight hours, and a total of 24,960° in one hour, making a total requirement of 200,960° per hour.

(4) Packing requires 6 bbl. per ton at 30c., and since the product obtained in this process is pure asphalt only one ton has to be barreled at a cost of $1.80.

(5) Attendance to an apparatus of 15 tons capacity demands for extraction and distillation two men for 24 hours each at $1.60 per day of 12 hours, or for 2,000 lb., 43c.; two men for 12 hours each at $1.60 per day, amounting for 2,000 lb. product to 21c.; a total of 63c.

To produce 2,000 lb. of bitumen the actual cost of the two methods may be summarized as follows, the figures for Trinidad asphalt being placed first and those for rock asphalt immediately following in parentheses: Amount crude material mined, 5,000 lb. (13,333); degrees temperature per 100 lb. 457,920 (200,960); expense of mining at 25c. per ton, 62c. ($1.66); materials consumed (oil or solvent) $3.20 ($1.86); Packing for shipment, $2.50 ($1.80); labor consumed in the process, 60c. (63c.); total, $6.92 ($5.45).*

The above estimate does not include office force, shipping, packing or yard labor, which will be about the same in either instance.

It appears, therefore, that rock asphalt can be produced cheaper than Trinidad or similar asphalts, especially when the freight is considered. Asphalt delivered in Chicago for instance, taking the above crude cost per ton of pure asphalt as a basis and assuming that the Trinidad product is refined in New York and the rock asphalt in Texas would cost, the figures for Texas being placed in parentheses after the corresponding figures for Trinidad: Refining, $6.92 ($5.45); freight at $6 per ton from either point, $9.99 ($6.00)—total, $16.91 ($11.45). The difference in freight is due to the tenor of impurities; 3,333 lb. Trinidad must be shipped to deliver 2,000 lb. bitumen, while the rock asphalt is pure bitumen. Considering that in addition to this freight difference the crude material in one instance must be transported from the West Indies and the contractor must store and handle so much more material it is surprising that the American asphalt mines are not further developed.

Asphaltum finds application for many purposes besides paving. The finer grades are used extensively in the manufacture of varnishes and as an electrical insulator. Both natural and artificial asphalt are used in making roofing paper, the former being preferable. Asphalt is also employed in the manufacture of rubber, waterproof cloth, in enamelling iron, and for many minor purposes.

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* The temperature calculations are purely empirical and are not intended to show actual calorific requirements, further than a comparison of temperatures and periods of maintenance called for by the two methods.

BARYTES.

The production of barytes in the United States in 1898 was derived wholly, as in the previous year, from three districts, namely, the vicinities of (1) Hot Springs, N. C., (2) Lynchburg, Va., and (3) Mineral Point, Potosi and Cadet, in Washington County, Mo. In the last district the mineral is mined in connection with galena from the veins and stockworks occurring in the Potosi limestone. The product is shipped to St. Louis, Mo., and Quincy, Ill., for grinding. In the Hot Springs and Lynchburg districts barytes is mined as a primary product. These deposits supply the Atlantic coast, the requirements of the Mississippi Valley being furnished by the mines in Missouri. The freight rates are prohibitive of any encroachment from one district into the other.

PRODUCTION, IMPORTS, AND CONSUMPTION OF BARYTES IN THE UNITED STATES.

(In tons of 2,000 lb.)

Year.	Production.			Imports.			Consumption.	
	Quantity.	Value Per Ton.	Value.	Quantity.	Value Per Ton.	Value.	Quantity.	Value.
1894	23,758	$4·00	$95,032	3,046	$5·19	$15,826	26,804	$110,858
1895	20,255	4·89	99,020	4,682	5·27	24,673	24,937	123,693
1896	21,900	4·00	87,600	3,333	7·39	24,619	25,233	112,219
1897	26,430	4·00	105,720	2,018	7·14	14,401	28,448	120,121
1898	28,247	4·00	112,988	1,914	5·93	11,356	30,161	124,344

Barytes Washing.—At certain places in Virginia pockets of barytes gravel are found, the mineral pebbles being coated with more or less clay and being apparently the residuum of a former deposit that has undergone decomposition. Usually this clay is free from other minerals than barytes, and when it is sufficiently heavy with the latter it is put aside until there is an accumulation large enough to make it worth while washing. The washers employed for this purpose are of the well-known log type, used in washing iron ores, phosphate rock, etc. If the clay contains pebbles of quartz, feldspar or other minerals besides barytes these are picked out by hand after washing.

Uses.—Barytes is employed chiefly as a pigment, only a comparatively small quantity being used in pyrotechny and as a reagent in certain chemical processes. As a pigment it is employed either as the native mineral finely pulverized and sometimes purified by levigation, or perhaps by treatment with acid to dissolve certain soluble impurities; or it is employed as blanc fixe, which is an artificial barium sulphate. A good deal of barium sulphate is used in the form of lithophone, which is an artificial compound of barium sulphate, zinc oxide and zinc sulphide. In the preparation of the artificial barium pigments barytes is not the only raw material, a large amount of witherite, or barium carbonate, being used for this purpose abroad. This mineral being attacked readily by chlorhydric acid has that advantage over barytes which is inert with most acids and has to be reduced to barium sulphide by furnacing with carbon in order to bring it into a soluble form.

Blanc fixe is prepared by adding sulphuric acid to a solution of barium chloride or some other barium salt. The barium sulphate thus produced is a better pigment than the natural sulphate, being amorphous while the latter is crystalline, and possessing a greater opacity and covering power.

Lithophone, which was introduced in England under the name of Charlton White, and is also known in the trade as Orr's White and Griffith's Patent Zinc White, is prepared by precipitating zinc sulphide and barium sulphate by mixture of solutions of zinc sulphate and barium sulphide, the precipitate being washed, dried and calcined. By the calcination a portion of the zinc sulphide is converted into zinc oxide, so that the composition of a commercial grade of lithophone may be barium sulphate 68%, zinc oxide 7·28%, and zinc sulphide 24·85%, there being some variation in different makes. Lithophone has advantages as a pigment for certain purposes.

Prices.—The value of crude barytes at the mines in Washington County, Mo., in 1898 was about $4·50 per 2,000 lb. against $4·25 in 1897. Delivered in St. Louis the crude mineral cost the grinders $7·25 per ton in 1898. In Virginia and North Carolina pure lump ore was worth about $4 and washed ore (concentrates) $3·75 on board cars at the mines.

Ground barytes was in steady demand throughout 1898, with an upward tendency in price. At the beginning of the year No. 1 domestic sold in New York at $16, No. 2 at $14, and No. 3 at $11. At the end the prices were $18, $16 and $14 respectively. In St. Louis and Chicago prices averaged somewhat lower. The Western market is supplied to a considerable extent with floated, or water sorted, barytes, while none of the products which reach the Eastern market is prepared in that way. At the close of the year a schedule of $14·50 for No. 2 and $16 for floated was established in Chicago and St. Louis. In December the principal barytes grinders of the United States formed a pool.

Formerly there was a large importation of German barytes into New York, but now this business has practically ceased, the difference in quality between it and the best domestic product not being sufficient to outweigh the existing tariff of $5·25 per 2,000 lb., notwithstanding the advantage in freight rates, which are only $1·50 per ton from Hamburg and Bremen to New York, while Virginia barytes has to pay $2·50 per ton. Notwithstanding the exclusion of foreign ground barytes, there is still a large importation of "lithophone" in which barytes is an important constituent. According to a tariff decision of 1898, under a protest of Gabriel & Schall, this product was assessed for duty at the rate of 1c. per lb. under classification as paint containing zinc, instead of 1·25c. per lb. as white sulphide of zinc, as appraised by the New York Custom House.

BISMUTH.

Small lots of bismuth ore have been produced occasionally in the United States and marketed abroad, but there is no regular output of this ore. The world's supply of metallic bismuth is obtained from Germany and England, the producers in both countries belonging to a combination to regulate production and prices. This combination controls the trade absolutely.

The importations of bismuth into the United States since 1892 have been as follows: 1893, 110,974 lb. ($200,848); 1894, 150,169 lb. ($198,047); 1895, 154,013 lb. ($128,714); 1896, 124,263 lb., ($90,950); 1897, 151,374 lb. ($172,236); and 1898, 137,205 lb. ($162,846). A few medicinal preparations of bismuth are also imported but the amount is very small.

The European producers of bismuth obtain their supply of ore and crude metal from Saxony, Australia, and Bolivia. In Bolivia bismuth occurs at Tasna and Oruro in the form of sulphide more or less associated with sulphides of antimony, copper, iron and lead, containing silver, the percentage of bismuth ranging from 15 to 30. Lack of fuel prevents much smelting, for which only dried cactus and llama dung are available for fuel, and it is consequently found better to ship the ore to England. The production of metallic bismuth in Bolivia was 26 metric tons in 1896 and 50 tons in 1897.

Nothing new in the metallurgy of bismuth was published in 1898. In analytical methods a new process of determining bismuth was described by L. Vanino and F. Treubert.* A slightly acid solution of a bismuth salt is mixed with alkaline formaldehyde and a large excess of 10% caustic soda solution is added. The mixture is heated on a water bath with continual stirring, until the precipitate of metallic bismuth has settled and the liquid has become quite clear. More formaldehyde and caustic soda are added and the mixture is heated over the open flame for a few minutes. The precipitate is repeatedly washed with water by decantation and is pressed to a metallic mass with a glass rod. The bismuth is collected on a weighed filter or in a Gooch crucible, washed with absolute alcohol and dried at 105° C. If the heating over the open flame be continued too long the liquid is apt to become yellow, but this is without influence on the result of the analysis. The reaction is applicable for the separation of bismuth from other metals.

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* *Journal of the Society of Chemical Industry*, Aug. 31, 1898.

BORAX.

THE production of crude borax in the United States fell off in 1898, owing to the abnormally high production in 1897, which overstocked the refineries with raw material. As in 1897, most of the output in 1898 was obtained from the colemanite deposits of California, a comparatively small amount being recovered from the marsh deposits of California and Nevada and a little from Oregon. Marsh deposits were worked at Candelaria, Nev., Humboldt, Nev., Amadee, Cal., Lovelocks, Nev., Independence, Nev., and at a place in Oregon about 120 miles N.W. of Winnemucca, Nev., whither the Oregon product is carted for shipment by railway. All the producers with two exceptions sold their product to the Pacific Coast Borax & Redwoods Chemical Works, Ltd., which continued to control this business in the United States. Boracic acid was made in California by two companies in 1898, while a third plant is in course of construction and will be in operation in 1899.

The California Borax Co. was organized in 1898 to erect works at Borax Lake, Cal., and to operate deposits in San Bernardino, Kern, and Inyo counties. The construction of a plant capable of turning out 100 tons of refined borax per month, located near the old Searles works, was begun, and it is contemplated to begin borax refining at this place early in the spring of 1899. It is reported that borax works are to be established in Rose Valley, Harney County, Ore., about 15 miles from Disaster Peak, Nev. At this place there are strongly flowing hot springs of which the water is heavily charged with sodium biborate. The evaporation of this water led to the formation of borax deposits over the surface of the adjacent desert. The Pacific Coast Borax & Redwoods Chemical Works, Ltd., erected a new refinery at Bayonne, N. J., which was put in operation December 20, 1898. These works, which have a nominal capacity of 12,000 tons per annum, are supplied with raw material from California and make boracic acid as well as borax. The old works at Alameda, Cal., which also have an estimated capacity of 12,000 tons per annum, continue in operation although probably on a smaller scale.

Early in 1899 a company called Borax Consolidated, Ltd., was organized in London with a capital stock of £1,400,000 to take over all the important borax-producing properties in the world and control the business. The organizers of this combination were L. H. De Friese, F. M. Smith, and R. C. Baker, who were directors in the Pacific Coast Borax & Redwoods Chemical Works, Ltd. The properties which were taken over are mentioned in detail further on in this review. In the prospectus of the new company the Pacific Coast Borax & Redwoods Chemical Works, Ltd. (formed by the consolidation of Redwoods Chemical Works and the Pacific Coast Borax Co. April 1, 1896), the Borax Co., Ltd., Mear & Green, Ltd., and the Société Lyonnaise are said to have realized aggregate profits, before charging interest on loans, income tax, and directors fees, as follows: 1893, £175,711; 1894, £133,154; 1895, £105,788; 1896, £109,067; 1897, £119,457. Those of the Empresa de Ascotan in the year ended January 31, 1895, were £12,328; 1896, £12,936; 1897, £14,565. The profits of the Pacific Coast Borax & Redwoods Chemical Works for five years ending March 31, 1898, averaged £73,721 per annum.

The property in the United States taken over by the new company comprised the colemanite deposit at Daggett, Cal., together with the calcining works and other equipment at that mine, of which the capacity is said to be over 30,000 tons per annum and the production in 1897 22,000 tons; the borax deposits in Death Valley, Cal., Esmeralda County, Nev., and Curry County, Ore.; and the refineries at Alameda, Cal., and Bayonne, N. J. The new company also required 7,142 out of 10,000 shares of the San Bernardino Borax Mining Co., of San Bernardino, Cal.

Statistics.—The production of crude borax in 1898, chiefly colemanite or calcium borate, was 15,335 short tons (13,912 metric tons) against 19,400 (17,600) in 1897 and 13,569 (12,310) in 1896. This product, which is turned out almost entirely by one company, is valued by us at a nominal price of $20 per 2,000 lb. Previous to 1896 we had statistics of the production of refined borax which were furnished us by the company controlling this business, but now for private reasons it refuses to give them. The production of refined borax in 1895 was 13,506,356 lb. (6,126 metric tons) which at 5·5c. per lb., or $121 per metric ton, represented a total value of $742,850. In 1894 the production of refined was 13,140,594 lb. (5,950 metric tons) worth at 7c. per lb. ($155 per metric ton) a total of $919,842.

THE WORLD'S PRODUCTION OF BORATES, ETC. (*a*) (IN METRIC TONS.)

Year.	United States. Calcium Borate.	Chile. Calcium Borate. (*b*)	India. Borax. (*b*)	Germany. Boracite.	Italy. Boric Acid, Crude.	Peru. Calcium Borate. (*b*)	Turkey. Pandermite (*b*) (*c*)
1894	5,950 (*d*)	6,700	367	176	2,746	800	9,100
1895	6,126 (*d*)	4,532	400	150	2,633	4,000	9,081
1896	12,310	7,486	340	184	2,616	1,179	12,626
1897	17,600	3,168	280	198	2,704	11,850	11,375

(*a*) From official reports of the respective countries except the United States. (*b*) Exports. (*c*) Fiscal years. (*d*) Product of refined borax. The manufacture of boric acid was begun in the United States in 1896, in which year there was a production of 621,000 lb. There are no statistics for subsequent years.

Chile.—Workable deposits of calcium borate occur extensively in Northern Chile. Besides small deposits near Tacna and in the Province of Tarapacá there are the following in the Province of Antofagasta: Carcote, Ascotan, Gran Salar de Atacama (282,740 hectares in area, south of San Pedro de Atacama and 115 km. from the Calama Station of the Antofagasta railway), Caurchari (17,062 hectares west of Susques), Rincon (46,078 hectares), Arizaro (343,086 hectares), Pocitas (59,376 hectares), Pastos Grandes (42,531 hectares), Pozuelo (12,187 hectares), Ratones (88,653 hectares, north of Antofagasta de la Sierra), and Antofaya (85,671 hectares, west of Antofagasta de la Sierra). The last eight lagoons are situated in the Desert of Atacama. In the Province of Atacama borate of lime deposits have been discovered in the lagoons of Pedernales (29,359 hectares), Maricunga (25,736 hectares), Laguna Verde, Negro Francisco and Laguna Brava. They are situated in the hinterland of the ports of Chañaral and Caldera.

Of these deposits at the present time only Carcote and Ascotan are worked extensively and rationally. The reason that the others, especially Pedernales and Maricunga, are not exploited is lack of communication and the general difficulty of mining in the provinces of Antofagasta and Atacama. In some places where ulexite occurs in considerable quantity in the nitrate deposits, as for example, 20 leagues from Iquique, it is recovered as a by-product.

Ascotan and Carcote are situated near the Bolivian boundary, not far from the railway from Antofagasta to Bolivia, respectively 361 and 411 km. from Antofagasta. The Ascotan deposits have been worked by the Empresa de Ascotan, which has been the principal producer in Chile. They form the bottom of an old lake and are 25 English miles long and 8 miles wide. The stratum is from 1 to 3 ft. thick. The company is equipped to produce 15,000 tons of calcium borate per annum. Its shipments in 1896 amounted to 6,974 tons. The exports of calcium borate, calcined borax, etc., from Chile are given in the following table (in metric tons):

Year.	Calcium Borate.	Calcined Borax.	Year.	Calcium Borate.	Calcined Borax.	Year.	Calcium Borate.	Calcined Borax.	Year.	Calcium Borate.	Calcined Borax.
1886.....	200	1,519	1889....	3,670	*Nil.*	1892....	2,908	108	1895....	4,425	107
1887.....	1,453	3,053	1890....	3,536	37	1893....	4,587	*Nil.*	1896....	8,486	*Nil.*
1888.....	1,132	538	1891....	6,342	803	1894....	6,700	*Nil.*	1897....	3,154	14

Most of this material is exported from Antofagasta, but a small quantity is shipped from Iquique and Caldera

The property in Chile acquired by the Borax Consolidated, Ltd., comprised the deposits of calcium borate belonging to the Empresa de Ascotan, which are considered to be the most important in the republic. These occur in the partially dried up Lago de Ascotan, which is estimated to contain the equivalent of 200,000 tons of calcined calcium borate. Connected with this property is a private railway line and calcining plant at Cebollar together with other works. For some years past it has shipped annually about 6,000 tons to Europe. The deposit of calcium borate, near Ascotan, known as Carcote, estimated to contain 300,000 tons of calcium borate of commercial quality (formerly the property of the Sociedad Boratera de Carcote) was also taken over; the Cosapilla deposit at Calpitas, near Tacna, estimated to contain about 50,000 tons (formerly the property of the Compania Boratera de Cosapilla); the Chilicolpa deposit near Tacna, estimated to contain 24,000 metric tons (formerly the property of Marchant & Co., of Arica, Chile); and the part of the

Pintados deposit at Iquique belonging to the Products Distribution Co., Ltd., which is estimated to contain 100,000 tons of borate.

Great Britain.—The Borax Consolidated, Ltd., took over the works at Belvedere, Kent, capable of refining 1,500 tons of borax per annum, formerly the property of the Pacific Coast Borax & Redwoods Chemical Works, Ltd., together with the refinery at Kidsgrove, Staffordshire, capable of producing 2,000 tons of borax per annum; and that at Connah's Quay, Flintshire, North Wales, capable of producing 1,000 tons of borax per annum and the same amount of boracic acid, formerly the property of Mear & Green, Ltd.

Peru.—The Peruvian deposits occur about 20 miles inland from Arequipa, shipments being made from Mollendo, the first exportation having been in 1893, when 400 metric tons were sent out. Since then the business has grown rapidly.

The Borax Consolidated, Ltd., took over the property of the Cia. Boratera de Arequipa, which has a plant capable of calcining 1,500 tons per month, and during the last three years is said to have produced 31,000 tons from 120,000 sq. meters, or more than 500 lb. per sq. meter, with estimated reserves equivalent to 1,100,000 tons of calcined calcium borate; the deposit at Arequipa, belonging to the Cia. Boretara de Urbinas, with a plant capable of calcining 400 tons per month; and the deposits at Arequipa owned by Señor Pena and Señor Caballero, of that place, which have not yet been worked.

Turkey.—The chief producers of borax mineral in Turkey have gone into the Borax Consolidated, Ltd. These are the mines and works near Sultan Tchair in the Villayet of Karassi, Asia Minor, near the port of Panderma, on the Sea of Marmora, belonging to the Borax Co., Ltd., which produced an average of nearly 8,000 tons per annum during the last three years, with estimated reserves of 250,000 tons in sight; and the refinery belonging to this company at Maisons Lafitte, near Paris, which is said to be capable of producing about 3,000 tons per annum of borax and boracic acid; also the mines of the Société Lyonnaise des Mines et Usines de Borax, at Karassi in the Villayet of Khudavendghar, Asia Minor, near the port of Panderma, which have been producing at the rate of 7,000 to 7,500 tons per annum and have an estimated reserve of 150,000 tons of mineral in sight; together with the refinery of this company at Lyons which is capable of producing 1,200 tons of borax and 400 tons of boracic acid per annum, and the refinery at Vienna which can produce about 600 tons per annum. The mines of the Borax Co., Ltd., are held under a concession from the Turkish government for 50 years from June, 1887, and those of the Société Lyonnaise for 99 years from October 1, 1889.

Other Countries.—H. W. Edwards states that a vein of calcium borate, 4 to 5 ft. thick where exposed, exists near Kououa in New Caledonia. Crystals of scheelite occur in the same vein. In Thibet there are known to be large deposits of borax which are unprofitable to work at present on account of their remoteness from railways. In Persia borax is said to occur abundantly on the border of the Sivdjan, and it is produced at numerous places for local consumption.

Market Conditions.—The borax industry in the United States at least continued to be immensely profitable. The Pacific Coast Borax & Redwoods Chemical Works, Ltd., reported for the year ended March 31, 1898, a profit of £88,372 which was an increase of more than 50% over the previous year. This increase was attributed especially to the remarkable expansion in the business in America, and the more profitable range of prices since the Dingley tariff law went into effect whereby the duty on refined borax was raised from 2c. to 5c. per lb. At the same time the cost of production was lowered by the construction of a branch line of railway to the mines and the introduction of improved machinery. A further reduction in costs is looked for.

During the last quarter of 1898 the American branch of the company reported the largest volume of business in the history of the industry. On January 31, 1899, the price of refined borax was raised to 7c. per lb. and that of refined lump in sacks to 6·75c., New York quotations. Later an advance of 0·5c. per lb. was put into effect and this price was maintained throughout the remainder of the year. Boracic acid sold at 10·5 to 11c. per lb. In England borax was as low as £13 during the first half of the year, and for a while the demand was comparatively small. The price of the crude mineral fell in about the same proportion as the refined article. During the second half of the year business improved and the price for both crude and refined advanced, the latter being £16 at the end of the year.

With respect to the future the indications are for a continued increase in the demand for borax and boracic acid and the number of industrial purposes to which these and their compounds are applicable. They are used now by earthenware, china and glass makers, enameled ironware makers, smelters, tanners, calico printers, cement makers, color manufacturers, bleachers, soap makers, meat packers, in laundries and for many household, toilet and surgical purposes. There was considerable discussion in England in 1898 as to the use of boracic acid as a preservative for food, but the preponderance of evidence seemed to be in favor of its use in small quantities as an entirely harmless preservative. However, only a small part of the world's production of borax is consumed in this manner although the meat packers are using considerably larger quantities than heretofore.

The chief feature of the borax industry in 1899 will be the action of the new borax combination in attempting to control the supply of raw material. This company has secured the most important deposits in Chile, Peru, Turkey, and the United States, and besides refining on its own account contemplates that the independent refiners in England, France, Germany, Austria, Russia, and the United States will be obliged to come to it for their supply of raw material. However, the world's borax deposits are so extensive outside of those acquired by the new company that it is exceedingly doubtful if it will be able to dictate terms to the independent refiners for more than the time required to open new mines.

PROGRESS IN THE TECHNOLOGY OF BORAX.

Treatment of Calcium Borate.—C. Masson, of Gembloux, Namur, and C. Tilliere, of Brussels, Belgium, propose to prepare borax from pandermite in the following manner:[*] An ammonium salt is treated with lime in a distilling column, and the ammonia set free is passed into water contained in a digester, and the solution is then charged with carbonic acid (or sulphurous acid). The digester is heated gently and calcium borate is introduced. The digester is then closed and more strongly heated for several hours, while the contents are mechanically agitated, ammonium biborate being formed. When the reaction is complete they are transferred to filter presses, whence the liquor passes to a "reaction chamber," where it is agitated with sodium chloride to produce sodium biborate, and after addition of a little ammonia the solution is discharged into crystallizers. After removal of the borax that crystallizes out the mother liquor is used instead of water in repeating the operation, but after several repetitions the mother liquor requires a special treatment. A modification of the process is employed for the treatment of boronatrocalcites.

Another process was patented[†] by Drs. Rickmann and Rappe, of Kalk bei Coeln, who propose to heat pulverized calcium borate with a proper amount of a suitable phosphate, for example, sodium phosphate, with or without the addition of soda. The following reaction takes place:

$$3CaB_4O_7+2Na_3PO_4=Ca_3(PO_4)_2+3Na_2B_4O_7.$$

The borax is dissolved in water, separated by filtration from the calcium phosphate, and crystallized out in the usual manner.

Estimation of Boric Acid.—L. C. Jones describes the various methods employed for the estimation of boric acid and a new volumetric method of his own devising, which appears to have much merit.[‡] According to Jones' method the solution is made clearly acid to litmus by chlorhydric acid and 5 c.c. of a 10% solution of barium chloride are added. An amount of potassium iodide and iodate sufficient to liberate an amount of iodine at least equivalent to the excess of chlorhydric acid in the acidified solution is mixed with starch in a separate beaker, and the iodine, which is usually thrown out by this mixture, is just bleached by a dilute solution of sodium thiosulphate. To the now neutral solution of iodide and iodate a single drop of the solution to be analyzed is transferred by a glass rod. If a blue coloration is developed the solution is acidic with chlorhydric acid, and all the boric acid is in free condition. The amount of iodide and iodate required depends upon the acidity of the solution containing boric acid. Usually 10 c.c. of a 25% solution of iodide and 10 c.c. of a saturated solution of iodate is sufficient. Any larger excess of chlorhydric acid should be neutralized by sodium hydroxide before the iodide and iodate mixture is added. After this addition the liberated iodine should be carefully bleached by thiosulphate. The presence of carbonates prevents a definite indication of the neutral point by thiosulphate and starch iodide, and therefore the barium chloride is added before this point in the process. The solution, after the bleaching, is colorless and contains only starch, neutral chloride, potassium tetrathionate, iodide, and iodate and all the boric acid (now in an uncombined condition). A few drops of phenolphtha-

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[*] D. R. Patent No. 98,680, April 24, 1898; English Patent No. 10,361, April 26, 1897.

[†] D. R. Patent No. 98,759, May 14, 1897.

[‡] *American Journal of Science*, February, 1899, pp. 147–153.

lein are now added, and the standard solution of approximately N÷5 sodium hydroxide is run in until a strong red coloration is produced. A pinch of mannite is then added, which bleaches the color, and the alkali solution is again run in to a faint indication, which if permanent on the addition of more mannite, may be taken as the reading point. About 1 to 2 g. of mannite are required. The small amount of carbonate present in the standard solution of alkali is precipitated by the barium chloride already in the solution. The calculation of the result must be based on the amount of free hydroxide in the standard solution according to the following equation:

$$B_2O_3+2NaOH=2(NaO,BO)+H_2O.$$

The best results and most definite indication are obtained with cold solutions of a volume not greater than 50 c.c. The presence of soluble silicates, fluorides, or carbonic dioxide is not objectionable, but ammonium salts interfere with the indications given by the phenolphthalein, and must be removed by boiling with potassium hydroxide in excess, or an indicator must be used which is not affected by them. A series of determinations of colemanite gave closely concordant results. A determination of boric acid can be made in five minutes.

H. Copaux proposes* to determine boric acid in alkaline borates by dissolving in 5 c.c. of water, and titrating with standard acid (sulphuric or chlorhydric) two drops of methyl orange being used as indicator. Since boric acid does not affect methyl orange, this titration gives the amount of alkali present. An alcoholic solution of glycerine containing two volumes of 30% glycerine, and one volume of 95% alcohol is then added to the solution under examination, two volumes of glycerine solution being used for one of the latter, and two drops of phenolphthalein, and the solution titrated with caustic soda (10 g. dissolved in a litre of water and decarbonated by boiling with lime) until a rose color just appears. The soda solution is standardized against boric hydrate (purified by crystallization and dried in vacuum over sulphuric acid) dissolved in an amount of water equal to that used to dissolve the borate, and treated with precisely the same amount of acid and glycerine solution, etc., as was used in the determination of the sample under examination.

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* *Comptes Rendus*, Vol. CXXVII., No. 20, Nov. 14, 1898.

BROMINE.

The total production of bromine in the United States in 1898 was 486,978 lb., of which 356,978 was in the form of liquid and the remainder in bromide of potassium. In 1897 the production of liquid bromine was 356,949 lb., and the total bromine production was 487,149 lb. The bromine business in the United States was without special feature during the year. The world's production of this substance continues in the hands of the Associated American Producers and the Leopoldshall-Stassfurt Convention of Germany, which divide the world's trade between them. Their agreement, which expired in the autumn of 1897, was renewed at that time for a further period of five years. At the same time a combination was effected among the European manufacturers of bromides, who immediately afterward raised the price of their products and since then have held it firmly at an advanced figure.

PRODUCTION OF BROMINE IN THE UNITED STATES.

Year.	Michigan.	Ohio.	Pennsylvania.	West Virginia.	Total.	Metric Tons.	Value.	
							Total.	Per Pound.
	Pounds.	Pounds.	Pounds.	Pounds.	Pounds.			
1894	29,116	146,501	100,623	103,304	379,544	172	$98,655	26c.
1895	30,280	152,360	104,647	107,567	394,854	179	102,662	26c.
1896	42,000	212,850	154,600	149,835	559,285	249	143,074	26c.
1897	(a) 147,256	124,972	116,967	97,954	487,149	221	136,402	28c.
1898	(a) 141,232	106,860	119,998	118,888	486,978	221	136,354	28c.

(a) Including the bromine equivalent of the product recovered as potassium bromide.

In the technology of bromine there were no important developments in 1898, although several new processes were described. Among these was one for the electrolytic recovery of bromine, patented by M. Nahnsen, of Magdeburg, Germany.* B. C. Hinman of New York patented a process† for the preparation of bromine compounds by treatment of a base such as caustic soda with bromine saturated with chlorine, the following reaction taking place:

$$Br_2+5Cl_2+12NaOH=10NaCl+2NaBrO_3+6H_2O.$$

If less chlorine is used a correspondingly less amount of sodium bromate is formed, sodium bromide being produced instead. This process has been devised in connection with a process for the treatment of gold ores.

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* German Patent No. 4,510 (Class 10), Aug. 10, 1898.

† English Patent No. 8,671, April 5, 1897.

In the application of the Hinman process to gold ores, the bromate and bromide of sodium coming in contact with an acid ore is decomposed with the liberation of bromine, which attacks the gold with the formation of auric bromide. The latter is removed by the passage of several water washes, and the solution is made to fall through a tower in which are rising air, steam and chlorine. The auric bromide is converted by the chlorine into auric chloride and bromine, the latter being volatilized by the heat of the steam and carried over into a tower through which is falling a solution of caustic soda, where it forms sodium bromide and hypobromite. Upon heating the solution the hypobromite passes into bromate, and the solution is then ready for the treatment of a fresh lot of ore, enough fresh bromide and bromate being added to supply any loss of bromine that may have occurred. The loss of bromine in this process is expected to be about 4%, although it is doubtful if so good a result will be attained in practice.

The recovery of bromine as proposed by Mr. Hinman is analogous to the process employed in the commercial production of this substance. In recovering bromine at Stassfurt in Germany the mother liquors contain from 0·15 to 0·35% bromine, and in strength are similar to the solutions which will have to be treated in the Hinman process. That the recovery of the bromine from these mother liquors, which are an otherwise waste product, is not a simple process is shown by the fact that when the bromine producers were engaged in a vigorous contest for the control of the market a few years ago, the lowest point quoted for the substance was 17·5c. per lb., which, it was commonly understood, was less than the cost of production.

The production of bromine in Germany in 1897, according to Paxmann in *Die Kali Industrie*, p. 31, was 490 metric tons; consequently Germany furnishes about two-thirds of the world's production.

CALCIUM CARBIDE.

The control of the calcium carbide business in the United States has been acquired by the Union Carbide Co. of Chicago, operating under the patents of the Electro Gas Co., which in May took over the works at Niagara Falls, N. Y., Sault Ste. Marie, Mich., and Appleton, Wis. Thereafter only the works at Niagara Falls were operated while arrangements were made for new works at that place and also at Sault Ste. Marie, the former to have an estimated daily capacity of 50 tons and the latter 100 tons. The domestic demand for carbide was very large during 1898 and in the latter part of the year the Union Carbide Co. refused to make any quotations on the substance for export, not having any surplus for that purpose. The range of prices was from $70 to $75 per 2,000 lb. in carload lots, or about the same as in the previous year. No statistics as to the production are available.

In Europe, on the other hand, the supply of carbide at the end of 1898 had become greater than the demand, though owing to the strict regulations as to transportation there were scarcities in certain disticts and countries. The over supply was attributable to the large number of new plants that had gone into operation, and when the output of those in construction comes on the market in 1899 a rapid decline in price is to be anticipated. At the end of 1897 carbide in wholesale lots was quoted in England at £18 10s. per 2,240 lb., not including packages, while at the end of 1898 the price had fallen to £14.

The New York fire department adopted strict regulations as to the sale and use of calcium carbide and acetylene. According to these all calcium carbide in transit through the city and on storage must be inclosed in hermetically sealed iron receptacles, and plainly marked "Calcium carbide: dangerous if not kept dry." No single package must exceed 100 lb. As to the sale of the carbide, not more than 20 lb., either in bulk or in cartridges, may be stored or kept in any building used for dwelling or mercantile purposes, and this amount can only be kept on a permit obtained from the fire department. This permit provides that all quantities in excess of 2 lb. shall be in tight metal packages, and kept elevated at least 6 in. from the floor, in a fireproof safe located above the street grade. The manufacture, transportation, storage, sale or use of liquefied acetylene is absolutely prohibited within the limits of New York. Provisions are made for the storage of calcium carbide in sealed receptacles in [illegible] buildings of fireproof construction, and the storage [illegible] department, and the entire [illegible] in the aggregate.

The number of works producing calcium carbide at present is said to be 29, as follows: United States—Niagara Falls; another works building at Sault Ste. Marie; Canada—St. Catharines; Great Britain—Foyers, Birmingham, Ingleton; France—La Bathie, [illegible], St. Beron, Notre Dame de Briançon, Froges, [illegible]; Germany—Frankfurt, Bitterfeld, Rheinfelden, Augsburg; Switzerland—Neuhausen, Vallorbes, Vernier, Vernayaz; Italy—San Marcel, Papigno, Terni; Spain—on the Ebro; Sweden—[illegible]; [illegible]. Besides the above 15 new carbide works are said to be in course of construction.

France.—According to United States Consul Thackara, writing from Havre, August 4, 1898, the wholesale price of calcium carbide in France at that time was [illegible] per metric ton, exclusive of the cost of packages, the latter amounting to [illegible] per iron drum, containing 50 kg., $1·19 per drum of 100 kg., and [illegible] per drum of [illegible] kg. The drums when empty are taken back by the manufacturers at the invoice price. The retail price of calcium carbide in France was [illegible] to [illegible] per 100 kg., not including package. The calcium carbide manufactured in France and Germany is guaranteed to give [illegible] liters of gas per kg.

Germany.—According to Frank H. Mason, United States Consul-General, writing from Frankfort, July [illegible], 1898, the demand for carbide in Germany has more than kept pace with the supply, and notwithstanding the number of new plants under construction or projected in Germany the supply is still obtained chiefly from Switzerland, especially from Neuhausen. Up to a year previous about the only manufacturer of carbide in Germany was the company at Bitterfeld, which employed steam power and on that account found itself unable to compete with the water power works and removed its plant to Neuhausen. The most important increase in the calcium carbide production of Europe is to be expected from the combination organized by the Schueckert Electrical Manufacturing Co., of Nuremberg, which in 1899 expects to have four works (one each in Spain, Bosnia, Norway and Switzerland) in operation, with an estimated annual capacity of 20,000 metric tons. Moreover the Aluminium Industrie Actiengesellschaft of Neuhausen has made plans to construct a large plant at Land-Gastein in Austria. The general opinion in the business is that carbide ought to be produced by well-managed water-power plants in Germany or Switzerland at a cost of $38 per metric ton. In 1898 calcium carbide could be purchased in large quantities at $8·09 to $8·33 per 100 kg. not including packing charges, the retail price being $10·25 to $11·09. Contracts for 1899 have been made at large reductions.

Switzerland.—The largest producers of calcium carbide in Switzerland in 1898 were the Aluminium Industrie Actiengesellschaft and the Electrochemische Werke Bitterfeld. There were several small producers and two large works in course of construction. The Swiss carbide was exported chiefly to Germany and England, while Bulgaria, Egypt, Tunis and Algeria took considerable quantities. The price at the works in 1898 was 40 fr. per 100 kg., excluding packages. The Swiss works have the advantages of excellent water power and an abundance of cheap and pure limestone, but coke is rather expensive, the best kind with 6 to 7% ash costing 35 to 40 fr. per ton. The raw material, however, plays but a small part in the cost of producing calcium carbide. The cost of an electrical horse power at Neuhausen is said to be 50 fr.

TECHNOLOGY OF CALCIUM CARBIDE.

There are two principal processes of making calcium carbide by electrical smelting—the intermittent and the continuous. In the former the coke and lime, finely ground and mixed in the proper proportion, are allowed to flow into and partially fill the crucible in which the electric arc has previously been struck. A partially fused mass, called an ingot in default of a better term, gradually builds up from the bottom, the carbon electrode being raised from time to time, automatically or manually, to suit the diminution of resistance due to the shortening of the arc by the rising level of the fusion. When the reaction has become complete the full crucible is withdrawn and another substituted, the change occupying only two or three minutes and being required only once in 10 or 15 hours. The crucibles are of metal and considerably larger than the ingot, which is surrounded by a mass of unreduced material protecting the crucible from the intense heat.

In the continuous process a stationary crucible lined with carbon is employed, with an electrode nearly as big as the crucible itself. A much higher current density is employed than in the intermittent process. Fine grinding of the raw material is unnecessary. There is complete fusion and the liquid carbide is tapped at short intervals, while there is no unreduced material. On the other hand, the output per e. h. p. per day is considerably less than in the intermittent or ingot process, and consequently the continuous process is only the more economical when power is comparatively cheap.*

Electric Furnaces.—The forms of electric furnace used for making carbide do not vary very greatly. The current must be made to pass between two poles, and whether these be vertical or horizontal, or whether one be a plate or the body of the furnace itself, does not appear to make any very radical difference. The most economical forms are the modifications of the Willson furnace used at Niagara, at St. Catherines in Canada, and at Foyers in Scotland; in this class of furnace the heat of the arc is kept in by the surrounding mixture of lime and carbon, so that very little waste of heat takes place. In the furnace used at Froges, on the other hand, one terminal is a carbon crucible or chamber into which the other pole is lowered, the result being that the carbide is so liquid that it can be run from the furnace. Other forms, such as the Bullier furnace used at Nôtre Dame de Briançon, and those used at Neuhausen and Bitterfeld, differ but little from the first class, so that the cost of production is more dependent upon the cost of the electrical energy used than upon anything else. According to W. S. Horry,† while there have been no recent changes in

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* Vivian B. Lewes, Cantor Lecture, 1898.

† In a paper read before the Northwestern Electrical Association, June, 1898.

the principles of calcium carbide smelting, the apparatus employed has been improved and continuous furnaces are now almost exclusively employed. Those used in the United States are large cast-iron drums, on the ends of which iron plates can be clamped to keep in the ores to be reduced. The drum is mounted on a shaft and can be slowly rotated by suitable gears. The electrodes are carbons fixed side by side, their points touching an imaginary horizontal plane, in which lies the axis of the furnace. The heat due to the resistance of the ores is developed between these two points. As the ore is reduced the resultant product lowers the resistance of the furnace and the ammeter indicator rises. When this occurs the furnace is slowly rotated, and by this movement the fused carbide is drawn down, away from the electrodes, and fresh material is also brought between them to be acted on in like manner. At Niagara a 500-h. p. furnace of this type is now being erected.

Efficiency of Calcium Carbide Furnaces.—According to Nicolai, for the production of 100 kg. of calcium carbide are required theoretically, 87·5 kg. of lime and 56·25 kg. of carbon. In practice the best mixture is 100 parts lime and 70 parts coke. V. B. Lewes gives* 100 parts lime and 68 parts carbon as a good working mixture, and states that from 0·3 lb. to a little more than 0·5 lb. of carbide can be produced by 1 e. h. p. hour. The mass which is formed in the furnace consists mainly of pure crystalline calcium carbide, but it is almost always surrounded by a crust in which the carbide contains a certain proportion of imperfectly converted constituents and which gives a lower yield of acetylene. If this crust is picked out the yield is perceptibly affected; in one case, where the furnace return was 0·504 lb. per kilowatt-hour, the amount after breaking and sorting was only 0·406 lb. One ton of carbide is obtained from 1·79 tons of the mixture of lime and coke. The larger part of the carbide put on the market at the present time contains only about 80% of calcium carbide. At Foyers it is produced of 87·92% purity. The actual yield there varies from 7·2 to 12 lb. per e. h. p. per day. Pictet calculates from thermal data that the average yield will not exceed 9 lb. of carbide per e. h. p. per day. Nicolai claims in the most favorable cases a yield of 51·98% of the raw material as carbide. E. A. Schneider considers that a well-conducted carbide plant should produce 3·5 to 4 kg. of carbide per e. h. p. during 27 hours. He also states that small carbide works with less than 2,000 h. p. are not profitable. The total cost of a plant utilizing about 3,000 e. h. p. should not be more than £24,000, provided the power is furnished by a plant already established.

Pictet's Improved Furnace.—Pictet proposes to obtain further economy by heating the charge before it enters the furnace. He constructs a furnace† resembling a cupola in appearance, only the furnace proper is in the center, and surrounded by an outer casing, leaving an annular space, in the bottom of which a coke fire heats the outer walls of the inner furnace. The latter is closed at the top and contracted at the bottom to form a funnel. The charge is heated and partially fused by the coke fires and the combustion of the carbonaceous matter in the charge, heated air being admitted through tuyères for

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* *Journal of Gas Lighting*, Nov. 30, 1897.

† English Patent No. 21,508, Sept. 20, 1897.

this purpose. The charge then falls into the contracted neck of the furnace, where it passes through an electric arc maintained between a pair of horizontal carbon poles. Complete fusion results, and the fluid carbide flows out through the bottom of the furnace into molds.

Nicolai's Furnace.—Nicolai says the lime must be burnt well. The coke should not contain more than 5% ash, and the material should be crushed as finely as permissible in practice, without causing too large a loss through dust. The present carbide furnaces present a series of difficulties. They utilize too little current, owing to the loss in the hot electrodes and the great resistance of the blocks of carbide. The quality of the carbide is unsatisfactory because it is too long subjected to the action of the current. At most carbide works the yield is only 3 kg. per h. p. hour. Preheating the material by the hot gases has proved disadvantageous. Repeated resmeltings have shown how the yield of acetylene falls off the longer the material remains in contact with the current. Nicolai has devised a furnace which is free from the above-mentioned difficulties. It consists of an annular metal trough provided with refractory electricity-conducting lining, and connected with the source of the current. The metal ring is placed horizontally and revolves around an axle. The mixture of coke and lime is fed into it by means of a screw conveyor. A suitable roll presses the mixture solid. An adjustable electrode of carbon carries out the smelting at one place. The carbide produced is stripped out of the ring by an obliquely placed knife. In this furnace it is claimed charcoal can be used.*

The Patin furnace, which is in use at the Puteaux factory, near Paris, works continuously, raw materials being introduced at the top and the finished carbide removed at the bottom. The furnace is rectangular in section. The extraction chamber beneath the actual furnace can be separated from the latter by means of a plate set in motion by a hand wheel. The foundations contain a piston which fits into the extraction chamber and can be moved up and down by a hand wheel. The electrodes are of carbon; their carriers are water jackets. At the beginning of the process, the electrodes, which slope downward, are made to meet in the axis of the furnace, and the piston is raised until it is a few centimeters below them. The furnace is then charged with lime and coke, the current is started, and the electrodes are gradually moved until they are the proper distance apart. When fusion has begun the piston is gradually moved downward at a speed proportional to the rate at which the carbide is being formed. When the piston has nearly reached the end of its downward stroke it is made to finish the stroke rapidly, and at the same time the plate is made to close the bottom of the furnace. A door in the side of the extraction chamber, which has hitherto been kept tightly closed by means of screw clamps, is now opened, and the carbide is withdrawn. The door having been again secured, the piston is pushed up and communication is again re-established between the furnace and the extraction chamber by withdrawing the plate. In the meantime the level of the material in the upper part of the furnace is kept constant by fresh supplies, which have been dried by waste heat, and are

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* *Stahl und Eisen*, 1898, XVIII., 727.

charged in while still hot. The consumption of the electrodes amounts to 1% of the carbide produced.

Preparation of the Charge.—G. Bower proposes to mix the lime and carbon with a binder—such as sugar, asphaltum, pitch, etc.—which in heating is converted into carbon, and then press the mixture into cakes. A special advantage claimed is the facility with which the gases evolved can escape between the cakes, which in turn, owing to their having been compressed, are better conductors. At Vernier and Geneva* the preparation of the raw mixture is performed mechanically and only one man's labor is required from the time the crude material is received. The coke used contains 5% ash, and the lime 99 to 99·5% CaO. Each furnace takes 500 e. h. p. (6,000 ampères at 57 volts), and is composed of a large cylindrical crucible, 59 in. in diameter, and 32 in. high. The electrode is made of compressed carbon, and is in six pieces, each 60 in. long and 5·2 by 9·2 in. in section. The total weight of the six carbons is over 850 lb. The crucible is fed from above by means of iron chutes.

J. Leeds, of Minneapolis, mixes† the lime with an equal weight of melted coal tar pitch, and heats the resulting mass in an ordinary retort furnace. The hard, coke-like product thus obtained is ground, and subjected to the arc.

New Proposals.—Efforts are being made in Germany to substitute strontium in place of lime, to cheapen the production of acetylene, by obtaining a valuable by-product in the form of strontium hydrate, which is largely used in the German sugar industry.

E. Jacobson proposes to obtain‡ carbides in the electric furnace from the sulphates or sulphides of the earthy alkalies. According to existing conditions, sulphur, carbon bisulphide or sulphurous acid are obtained as by-products. To obtain sulphur the process has to be conducted in a current of carbonic acid; if sulphur dioxide is desired air is employed. Carbon bisulphide is formed if the mixture is covered with coal and the reaction is conducted in a current of carbonic acid. How far in all these processes the carbide is applicable for the manufacture of acetylene is yet to be found out. Willson obtains§ carbonic acid and carbon monoxide as by-products. F. J. Bergmann, Neheim am Ruter, Germany, produces‖ calcium carbide in blast furnaces. Coke, or a liquid hydrocarbon, is fused with lime in a water-jacketed blast furnace, such as is used in lead and copper works, by means of a blast of oxygen. The latter is obtained by heating manganese dioxide in a suitable furnace, and then regenerating the peroxide from the brown oxide by a current of air at a temperature below red heat. The blast furnace is rectangular, and divided into three compartments, for coke. With liquid fuel it is cylindrical. The hearth is graphite in either case. Less oxygen is required with the liquid fuel.

Calcium carbide is an energetic reducing agent in metallurgical processes. H. N. Warren, by heating litharge and calcium carbide, obtained lead or a lead-calcium alloy, according to the proportions of the mixture; he obtained calcium alloys with tin, copper, iron, manganese, nickel, cobalt and chromium.

The storage and transportation of calcium carbide still present considerable

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* *Journal of the Society of Chemical Industry*, February, 1898.

† English Patent No. 32, 1898.

‡ German Patent No. 77,168.

§ United States Patent No. 492,377.

‖ English Patent No. 29,384, Dec. 11, 1897.

difficulty, and several improvements have been suggested in this direction, none of which, however, appear feasible. The coating of the carbide with oil or varnish is suggested; another proposition is to replace the air in the canister by carbonic acid. H. Herzfeld, at the congress of engineers of the carbide and acetylene industry at Frankfurt-am-Main, condemned* the practice of sealing carbide canisters hermetically, and stated that any possibility of pressure accumulation in transport packages should be avoided. He called attention to a canister used by the acetylene works at Augsburg-Oberhausen which is provided with a safety valve of simple construction.

"Carbolite," an Alleged New Carbide.—There was considerable discussion in the technical press in 1898 as to a process invented by Herman L. Hartenstein for the production of "carbolite" as a by-product in iron smelting, this being, it was claimed, a cheap substitute for calcium carbide, and of the same nature as the latter. In the patents it was described as calcium-aluminum-silicon carbide, represented by the formula Al_4C_3, SiC, CaC_2. According to an account furnished by the inventor it is manufactured by taking blast-furnace slag direct from the furnace, placing it in converters similar to those used in steel making, impregnating the molten mass with pulverized coke by means of a gas blast, and then turning the converters on their trunnions so that the mixture is brought in contact with carbon bars through which a powerful current of electricity is passed, generating the intense heat required to produce the carbides. The claims made for this process seem to be ill founded. Blast-furnace slag contains at the most only 50% of lime, and a carbide thus made would be a very poor substitute for pure calcium carbide in the evolution of acetylene. The silica in such slag would combine with the carbon through the agencies of the powerful electric current to form silicon carbide, or carborundum, which gives off no gas on the addition of water. The alumina under the same conditions would form aluminum carbide, which on addition of water evolves methane, CH_4, which is only faintly luminous, it being the same in substance as natural gas. The iron oxide in the slag would be reduced to iron carbide and would thereby add still more impurity to the "carbolite." From this it follows that the "carbolite" would be so mixed with non-gas-making impurities that it would generate only a small amount of illuminating gas as compared with pure calcium carbide. It should also be borne in mind that the production of these useless impurities (aluminum, silicon and iron carbides) costs electrical energy, and although some economy would be secured by beginning with an already heated product, it is doubtful if this would offset the waste of electrical power.

Analytical Methods.—In the usual methods of analyzing calcium carbide, the gas evolved from a given weight of the sample on treatment with water is collected and measured, corrections being made for temperature and pressure and for the amount of gas dissolving in the liquids with which it comes in contact. In order to avoid these calculations and the chance of error in the solubility allowance, H. Bamberger has devised a gravimetric method which is based on the same principle as the differential determination of carbon dioxide

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* *Chemiker Zeitung*, 1898, p. 903.

in carbonates.* He employs a flask holding about 400 c.c., fitted with a dropping funnel of about 200 c.c. capacity, and a drying tube filled with calcium chloride. From 50 to 60 g. of the carbide in small pieces are placed in the flask, and the funnel filled with a 15 to 20% solution of sodium chloride. The apparatus (about 500 g. in all) is weighed on a balance sensitive to 1 mg., and the salt solution is then allowed to fall drop by drop upon the carbide. After three or four hours, when about half the solution has entered the flask, and the whole of the carbide has been decomposed, the remainder of the liquid is rapidly added, the flask slightly warmed, and a current of air drawn through the apparatus. When cold, it is again weighed, and the loss in weight gives the amount of acetylene evolved. From this the proportion of calcium carbide in the sample and its yield of acetylene per kilogram is calculated, with the data that 1 kg. of pure carbide yields 40·625% or 348·9 liters of acetylene.

The impurities in the gas may also be determined by attaching to the outlet of the calcium chloride tube a train of bulbs containing oxidizing liquids. The solubility of phosphoretted hydrogen and sulphuretted hydrogen in the confining liquid renders impracticable their estimation when the yield of gas is determined volumetrically.

R. Fuchs and R. Schiff estimate the yield of acetylene from carbide by collecting the gas over water, protected from it by a layer of oil. The movements of the oil and water, however, cloud the glass walls. This may be obviated by use of a bottle (*a*) of about 20 liters capacity, graduated in divisions of 0·1 liter, and connected by flexible tubing with another bottle (*b*) of similar capacity. A concentrated solution of common salt saturated at the temperature of the room with acetylene is used as the confining liquid in the first bottle. The gas is generated from the weighed granulated carbide in a two-necked bottle (*c*), which is provided with a tap funnel and a tube packed with cotton wool. The level of the solution in *a* is read after *c* is connected with *a*; the cock in the funnel is then opened, and water is allowed to drop slowly on the carbide in *c* until all the gas has been evolved therefrom, when the level of the solution in *a* is again read, and the volume of gas at the prevailing temperature and pressure is thus found.†

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* *Zeits. f. Angew. Chemie*, 1898 (9), 196–198; *Idem*, 1898 (11), 243; *Journal of the Society of Chemical Industry*, April 30, 1898, p. 382; *Idem*, May 31, 1898, p. 494.

† *Chemiker Zeitung*, XXI., p. 875; *Journal of the Society of Chemical Industry*, May 31, 1898, p. 494.

CARBORUNDUM.

By E. G. Acheson.

The manufacture of carborundum in the United States during 1898 remained as heretofore in the hands of the Carborundum Co. of Niagara Falls, N. Y. The output for the year amounted to 1,594,152 lb. against 1,242,929 lb. in the preceding year. In 1898 a still further reduction was made in the price of the grain product, the latest quotations being 10 to 12c.; the price of powders remained 8c., as during 1897. About three-fourths of the production in 1898 was in grains, the remainder in powders. Consequently the average value of the product in 1898 is reckoned at 9·5c. per lb. against 12·3c. in 1897.

The use of carborundum in grains and powders as a substitute for emery and corundum made decided advances and it made a successful showing in all lines of work, with the single exception of belts, where the practice has been to mount emery with glue. The difficulty encountered in using carborundum in this field is the extreme sharpness and hardness of the particles, which causes them to be torn from the glue or cement mounting, as the result of the deep cuts they make in the metal being ground. Early in the year it was introduced into the granite mills at Aberdeen, Scotland. The polishers quickly appreciated its value as a substitute for emery, and at the close of the year they were ordering at the rate of about 2·5 tons per month.

The sales of vitrified wheels in 1898 were over 50% greater than in 1897. Early in the year a very valuable improvement was made in the method of binding wheels and other forms of goods. Prior to January, 1898, the wheels had not proved a success when used for grinding tools, saws, and steel work generally, but the changes that were then made in the binder worked such a decided improvement that those now made for these particular purposes are, perhaps, the most successful of any produced. The change consisted in materially reducing the amount of bond (porcelain) and introducing iron, which at the temperature of vitrification fuses and combines with the porcelain, and at the same time chemically attacks the surface of the carborundum particles, causing a distinct and firm union of the mass, with the smallest amount of binder, thus producing a wheel of great strength and splendid cutting qualities.

An effort was made by the Carborundum Co. to have the sandpaper and emery cloth manufacturers take up the manufacture of carborundum paper and cloth, but although some of them were disposed to handle the product to a limited extent, the negotiations were not successful, and as a result a very extensive and complete factory for the spreading of carborundum on paper and cloth was added to the plant of the Carborundum Co. This department was put into operation about October 1st, and the company is extremely well satisfied with the venture. Thus far the work has been principally directed toward the introduction of the paper into the shoe trade as a substitute for garnet and ruby paper. At the close of the year many prominent shoe factories had adopted the paper, to the exclusion of the garnet, their results showing it accomplished about twice the amount of work done by garnet paper, while the price was only about 60% higher. A few experiments made with the paper in the woodworking trade have encouraged the company to believe that in the near future it will be able to secure a large portion of the work in that line.

The use of carborundum in steel making was a subject of investigation by a number of steel manufacturers, and several of the leading steel casting companies have adopted it as a substitute for the ferrosilicon formerly used. There was sold for this purpose 110 tons in 1898.

Experiments on the use of a non-crystalline variety of carborundum—known as amorphous carborundum—as a refractory lining for furnaces, and for the manufacture of refractory bricks, crucibles, etc., were very successful, and indicate that carborundum may prove a very valuable material for the uses of the furnaceman and metallurgist. This amorphous material is produced in the carborundum furnace simultaneously with the crystalline, but in those portions more remote from the zone of high temperature. It is the first step in the conversion of the crude material into crystalline carborundum, and when sufficiently heated changes into the latter form. Unlike the crystalline, it is not chemically acted upon by molten iron, and having been formed at an extremely high temperature (probably more than 4,000° F.) is admirably suited for the purposes in view.

During 1898 a small factory was built at Niagara Falls, Ont., in which 200 e. h. p. is used in the manufacture of carborundum, the current being supplied by the Canadian Electric Power Co. The plant was built by the Carborundum Co. of the United States, which owns the carborundum patents in Canada. The company proposes to limit the work to the production of crude carborundum and the crushing and sizing into the various grains, as required by the trade. The manufacture of wheels and other forms required by the trade has been arranged for with the Hart Emery Wheel Co., of Hamilton, Ont.

CEMENT.

The condition of the cement industry in the United States in 1898 is shown in the subjoined table, in which the most striking feature is the enormous increase in the manufacture of Portland cement, especially in New York, New Jersey and Pennsylvania, continuing the expansion which began a few years ago. The causes which have led to the great development of this industry are described in detail in the paper by Mr. Lewis which is appended.

The increase in the production of natural rock cement was due almost entirely to the Indiana-Kentucky district, where production was stimulated undoubtedly by the low prices which were brought about by the fierce fight among the producers. The prices ruling in this district in 1898 were at the cost of production to many of the producers and in some cases were below it. The falling off in the production of this kind of cement in other districts is to be attributed to the increasing substitution of Portland cement for it, the superiority of the latter being now generally recognized, giving due consideration to the disparity in prices. Consequently a decrease in the production of natural rock cement is to be looked for in the near future. The production of slag cement increased from 40,000 bbl. of 400 lb., valued at $60,000, in 1897 to 157,662 bbl., valued at $235,721, in 1898.

PRODUCTION OF PORTLAND CEMENT IN THE UNITED STATES.
(In barrels of 400 lb.)

States.	1897.			1898.		
	Barrels.	Value at Works.		Barrels.	Value at Works.	
		Total.	Per Bbl.		Total.	Ber Bbl.
California	17,060	$51,180	$3·00	40,586	$92,130	$2·27
New Jersey	440,454	641,421	1·46	585,000	994,500	1·70
New York	398,489	649,469	1·63	529,820	900,694	1·70
Ohio	147,888	260,611	1·76	277,805	479,368	1·73
Pennsylvania	1,316,181	1,908,390	1·45	2,017,038	3,428,965	1·70
South Dakota	39,900	75.810	1·90	30,000	60,000	2·00
Texas	7,778	19,912	2·56	(*e*) 8,000	(*e*) 20,800	2·60
Other States (*a*)	63,153	118,112	1·87	96,337	191,649	1·99
Total	2,430,903	$3,724,905	$1·54	3,584,586	$6,168,106	$1·70

(*a*) Includes Arkansas, Illinois, Indiana, Maryland, Michigan and Utah. (*e*) Estimated.

PRODUCTION OF NATURAL HYDRAULIC CEMENT IN THE UNITED STATES.

(In barrels of 300 lb.)

States.	1897.			1898.		
	Barrels.	Value at Works.		Barrels.	Value at Works.	
		Total.	Per Bbl.		Total.	Per Bbl.
Illinois	500,000	$175,000	$0·35	531,000	$160,000	$0·30
Indiana and Kentucky	1,539,818	615,927	0·40	1,929,018	482,254	0·25
Kansas	140,661	56,226	0·31	180,000	67,000	0·37
Maryland	350,000	168,600	0·48	235,000	99,640	0·42
Minnesota	81,731	40,865	0·50	98,436	49,218	0·50
New York:						
Ulster County	3,372,209	2,023,724	0·60	3,245,225	2,103,554	0·65
Onondaga County	90,625	!51,106	0·56	122,300	62,700	0·51
Schoharie County	8,000	5,200	0·65	61,272	27,572	0·45
Erie County	581,000	203,350	0·35	605,000	211,750	0·35
Ohio	26,035	21,341	0·82	27,144	19,391	0·79
Pennsylvania	758,355	379,177	0·50	694,152	347,076	0·50
Virginia	12,384	7,430	0·60	5,491	3,430	0·62
Wisconsin	400,000	200,000	0·50	(e) 400,000	(e) 160,000	0·40
Other States (a)	29,755	28,104	0·94	27,040	26,410	0·98
Totals	7,890,573	$3,976,050	$0·50	8,161,078	$3,819,995	$0·47

(a) Includes Georgia, Florida and Texas. (e) Estimated.

CEMENT PRODUCTION, IMPORTS, EXPORTS AND CONSUMPTION IN THE UNITED STATES.

(In barrels of 300 lb.)

Year.	Production.				Imports.		Exports. (b)		Consumption.	
	Natural Hydraulic.	Portland.	Total Barrels.	Value.	Barrels.	Value.	Barrels	Value.	Barrels.	Value.
1893	7,445,950	673,989	8,119,939	$6,063,131	3,565,532	$3,470,169	143,682	$174,663	11,541,789	$9,358,737
1894	7,813,766	814,972	8,628,738	5,665,374	3,517,476	3,396,729	156,976	180,881	11,989,238	8,881,222
1895	7,694,053	998,745	8,692,798	6,027,374	3,996,527	3,873,123	123,437	131,541	12,565,888	9,768,956
1896	7,407,311	a 2,103,044	9,510,355	6,888,441	3,558,166	3,394,426	87,910	103,315	12,980,611	10,179,552
1897	7,890,573	a 3,294,537	11,185,110	7,760,955	2,787,766	2,688,122	79,583	103,389	13,893,293	10,345,688
1898	8,161,078	a 4,989,664	13,150,742	10,223,822	2,687,473	2,625,950	70,915	98,150	15,767,300	12,751,622

(a) Includes slag cement. (b) Includes re-exports of foreign.

IMPORTS OF CEMENT INTO THE UNITED STATES ACCORDING TO SOURCE.

Year.	Belgium.		British North America.		France.		Germany.	
	Short Tons.	Value.	Short Tons.	Value.	Short Tons.	Value.	Short Tons.	Value.
1895	141,775	$865,687	2,083	$18,894	4,567	$27,842	259,984	$1,737,442
1896	140,197	813,827	2,006	17,777	6,531	38,317	246,666	1,629,785
1897	109,531	649,675	981	9,311	7,545	44,574	221,856	1,467,845
1898	134,163	802,671	955	8,852	3,459	22,224	206,982	1,394,551

Year.	United Kingdom.		Other Countries.		Total Imports.		Exports. (a)	
	Short Tons.	Value.	Short Tons.	Value.	Short Tons.	Value.	Short Tons.	Value.
1895	161,377	$1,014,316	29.693	$208,942	599,479	$3,873,123	18,566	$131,541
1896	126,268	08,550	12,056	86,615	533,725	3,394,426	13,186	103,315
1897	68.869	451,256	9,380	65,461	418,165	2,688,122	11,937	103,389
1898	47,212	333,405	10,350	64,247	403,121	2.625,950	10,637	98,150

(a) Estimated from number of barrels reported at 1 bbl.=400 lb. Includes re-exports of foreign.

THE HYDRAULIC CEMENT INDUSTRY IN THE UNITED STATES IN 1898.

BY FREDERICK H. LEWIS.

Portland Cement.—The American Portland cement industry is at this time in an extremely interesting stage of development. With a very large production in 1898, and every plant taxed to its utmost, there was nevertheless a period of several months during the spring and fall when the demand for cement exceeded the supply, and the prospects are that a similar state of affairs will prevail this year. This is due not only to the greatly increased consumption of cement in this country but to a demand for it which is world wide and which is taxing the entire productive capacity of the United States and of Europe. Standard grades of European cements are commanding $1·50 to $1·85 per bbl. at British and Continental ports, and under these conditions the export to the United States has fallen off considerably. Not only so but the cements imported to this country during 1898 were many of them of inferior quality, distinctly inferior to the best grades of domestic cements now being manufactured. As a result of this inferiority the domestic cements actually commanded a premium in the New York market over a majority of the imported brands of cement during the greater part of the year. The condition of affairs as regards American production at the present time is shown by the following estimates of the output of the plants in operation in 1898, and projected.

CEMENT STATISTICS OF THE LEHIGH VALLEY DISTRICT. (BARRELS OF 400 LB.)

Company.	Kilns.		Estimated Annual Output—Barrels.		
	Now Installed.	To Build in 1899.	1898.	1899.	1900.
Atlas Cement Co.	28 Rotary.	Others. Rotary.	1,000,000	1,250,000	1,500,000
Vulcanite Cement Co.	11 Rotary.		225,000	425,000	500,000
Alpha Cement Co.	10 Rotary.	May build Rotary.	350,000	350,000	500,000
Saylors Cement Co.	11 Schoefer. 23 Dome.	8 Rotary.	275,000	315,000	625,000
Giant Cement Co.	62 Dome.	5 Rotary.	375,000	425,000	625,000
Lehigh Cement Co.	4 Rotary.	4 Rotary.	50,000	240,000	360,000
Bonneville Cement Co.	3 Rotary.	3 Rotary.	60,000	100,000	200,000
Hercules Cement Co.	7 Dome.	(?)	40,000	40,000	40,000
Nazareth Cement Co.	None.	6 Rotary.	None.	75,000	215,000
Lawrence Cement Co., of Pa.	5 Dome.	8 Rotary.	25,000	100,000	285,000
Total	164		2,400,000	3,320,000	4,850,000

New Construction in 1898 *and Proposals for* 1899.—In the Lehigh district the Atlas Cement Co. added seven rotary kilns to its plant at Northampton, Pa., during 1898 and had a capacity during the latter part of the year of 4,000 bbl. daily. This company expects to enlarge its plant still further during 1899. The Coplay Cement Co., at Coplay, added two Schoefer shaft kilns in 1898 and has in course of construction this year a rotary kiln plant containing eight kilns, which when completed will more than double the company's output. The American Cement Co., at Egypt, added six dome kilns to its plant in 1898, and has under construction five rotary kilns which will add 60% to its productive capacity of Portland cement. The Bonneville Cement Co., at

Siegfried, rebuilt its works with three rotary kilns in 1898, and expects to build three more this year and to lengthen the present kilns, increasing their capacity, and to be able to produce at the end of 1899 about 900 bbl. per day. The Lawrence Cement Co., at Siegfried's Bridge, began work in the summer of 1898 on a rotary kiln plant which is still in course of construction. It is not likely that this plant will cut much if any figure in production in 1899, but it should have an output of 1,000 bbl. per day in 1890. The Lehigh Portland Cement Co. at Ormrod, built a new rotary plant of four kilns in 1898, the buildings being designed for eight kilns. Four more kilns will be added this year, giving the company a production in 1900 of 1,000 bbl. a day. A plant was also under construction in 1898 at Nazareth, for the Nazareth Cement Co. This plant was expected to be making cement by the first of July, 1899. The Alpha Cement Co. added two rotary kilns last year, and is considering the building of a new plant adjacent to the old one, which, if constructed, will nearly double the capacity of the works in 1900.

In New York the Glens Falls Portland Cement Co. with six Schoefer kilns, the Empire Cement Co. with twenty dome kilns, the Genesee Cement Co. with twelve dome kilns, the Millen Cement Co. with sixteen dome kilns, and the Jordan Cement Co. with six dome kilns are estimated to have produced 420,000 bbl. in 1898, while their probable production in 1899 will be 460,000 bbl. The Glens Falls Co. proposes adding two Schoefer shaft kilns in 1899, which will increase the capacity 60%. The Genesee and Millen plants, at Wayland, controlled by the Wayland Cement Co., which have respectively 12 and 16 dome kilns, will probably have four additional dome kilns each.

In Ohio the Diamond Cement Co., at Middle Branch, had six Dietzsch kilns in operation in 1898, having rebuilt its plant which had been destroyed by fire in 1897, and expects to add two more in 1899. The Buckeye Portland Cement Co., at Bellefontaine, had two Dietzsch, four shaft and one new rotary kilns in 1898, to which two rotaries will probably be added in 1899. The Sandusky Cement Co., installed one new rotary in 1898, making a total of five, and is preparing plans for a large new plant at Syracuse, Ind. The Art Portland Cement Co., at Sandusky, operated two rotaries in 1898, to which another is to be added. The Castalia Cement Co. had four rotaries in operation. The reported production of these companies in 1898 was 277,805 bbl. In 1899 it is expected to be 410,000 bbl.

In Michigan the Michigan Portland Cement Co., at Coldwater, built in 1898 four rotary kilns, and intends adding 10 similar kilns to its plant and to build a second plant of equal capacity a few miles away, at Quincy, a total increase of 24 rotary kilns. This company is organized with large capital and seems likely to become one of the heaviest producers of Portland cement in the United States. The Bronson Portland Cement Co., had seven rotary kilns in operation in 1898, to which it intends adding three more rotaries in 1899 and to build another plant. The Peerless Cement Co., at Union City, operated 18 dome kilns up to October, 1898, when its works were seriously damaged by fire. The Michigan Alkali Co. is reported to have three rotary kilns in construction, with the idea of utilizing for making cement the calcium carbonate produced as a waste product. The total production of the Michigan works in 1898 was 81,500 bbl.

Chicago capital is expected to build a rotary kiln plant at La Salle, Ill.; it will, however, be constructed by German engineers from plans based on experimental work abroad. The Chicago Cement Co. has a plant of 10 shaft kilns nearly completed. The White Cliffs Cement Co., of Arkansas, has four Schoefer kilns and four more projected. The Portland Cement Co. of Utah operates one rotary kiln at Parley's Cañon, about 8 miles from Salt Lake City, using argillaceous limestone as raw material. The Western Portland Cement Co., of Yankton, S. D., has six Johnson kilns. There are also several rotary kiln plants projected in California and Texas.

The foregoing list of existing plants and estimates of additions thereto indicates a total domestic output for 1899 of about 5,000,000 bbl. of Portland cement. In 1900 the output of plants now existing and of those which seem certain to be completed in 1899 would be about 7,000,000 bbl.; and as there are a number of enterprises under consideration, some of which will certainly be completed during the present year and in operation in the spring or fall of 1900, it may fairly be estimated that the production during that year will reach 7,500,000 bbl. and perhaps more.

Considerable activity in the manufacture of Portland cement is reported in Canada. The present capacity of these plants is as follows: The Owen Sound Portland Cement Co., at Shallow Lake, in Grey County, Ont., has a small plant which it is planning to enlarge. The Rathbun Cement Co. has two Dietzsch kilns at Napanee Mills. The Beaver Portland Cement Co.'s works at Marlbank, Ont., has been bought by American capitalists. It has one dome kiln and one Hoffman kiln. The new proprietors are adding four rotaries. The plant of the Canadian Pacific Co., at Victoria, now operating in a small way, is reported to have secured capital to construct a very large plant with the idea of supplying the entire Pacific coast.

Technology.—This extremely rapid growth of the Portland cement industry is entirely logical. It is based upon the fact that a process of manufacture suitable to American conditions has now been developed. This is the rotary kiln process employing powdered coal for fuel. It cannot of course be claimed that the rotary kiln is a new process nor that it is essentially a new process in America, since it has been in use to some extent for 10 years past. As a labor-saving device this form of kiln has had much to commend it from the first, but it was handicapped for five or six years by the very expensive fuel which was employed, that is, crude petroleum. The cost of this fuel per barrel of cement was high, varying all the way from 28c. to over 40c. per bbl., depending upon the price of oil. By the use of powdered coal, however, this expensive fuel cost has been reduced. And at the same time the evolution of the mechanical features for handling materials in rotary kiln plants has been going on and these are now so perfected that the labor cost has been also much reduced. It is confidently believed that the cost of producing cement in America at the present time in the best equipped plants for that purpose is lower than in the English works and nearly if not quite as low as in German plants. This development of an entirely suitable process for manufacturing in this country is a sufficient reason for the very large development of the industry here, since it absolutely assures manufacturers of the domestic market and enables them to look forward confidently to an export trade at any time when the production reaches such volume as to make it worth while.

New Uses.—Side by side with this great growth in the industry is to be observed a widening field for the use of cement and particularly Portland cement. Cements of one grade or another are very rapidly displacing lime mortar for all kinds of masonry, besides finding a very large place in concrete construction in a great variety of applications. The indications are that there will be a rapid growth in the use of cement for monolithic concrete construction and that in this field Portland cement will be largely employed to the exclusion of Roman or natural hydraulic cements. In this matter of monolithic construction it is interesting to note that leading railroads in the Mississippi Valley, such as the Illinois Central Railroad and the Kansas City, Pittsburg & Gulf Railroad, are employing it for the construction of concrete bridge piers and in arch culverts. This is entirely logical, and the general use of concrete masonry in the Prairie States and in the great central valley may be expected as a natural economic development. A large amount of concrete has been used on the elevated railways in Chicago and for monolithic construction of sewers in Washington and Chicago. The Lake Superior Power Co., of Sault Ste. Marie, is about to build a very large water-power plant employing concrete masonry throughout for its dam and wheel pits.

It is evident that when it comes to the common use of this type of concrete construction for railways and for national, state and municipal public works the consumption of Portland cement will be sufficient to sustain an immense American industry in this product. Hence, while the growth of the Portland cement industry at this time is very rapid, there does not seem to be any likelihood of an overproduction or of a failure to find an adequate market for the product. The estimated production of 1900 is not likely to exceed 7,500,000 bbl., which is considerably less than the present English output and less than half that which Germany has produced in recent years.

Advantages of the Lehigh District.—It is of interest to call attention to the output of the Atlas Cement Co. given in the preceding table. This company, operating in Lehigh and Northampton counties in Pennsylvania, is now the largest producer of Portland cement in the world. And it may be remarked that this company was the first to use rotary kilns and contributed largely to the successful development of this process to its present economical basis of production. The table also emphasizes the great aggregation of plants, producing from 70 to 75% of the output of Portland cement, which are found in or near the Lehigh Valley in Pennsylvania and New Jersey. The raw materials for cement production in this locality are undoubtedly superior, and all things considered this district possesses unusual advantages for the economical production of cement. It can hardly be expected, however, that this great predominance of the Lehigh Valley will continue. The cost of transportation of a bulky and comparatively cheap product like cement will always favor the establishment of local plants near the larger centers of consumption; and we find on this account large works planned or in process of construction in the marl region of Michigan and Indiana, which are evidently attracted to that locality by the great markets afforded in the lake cities. With an export trade in view and the certainty of being able to supply cement to the republics of South and Central America, plants on the seaboard, or which are sufficiently near it to offer special advantages in ocean shipments, are receiving a good deal of attention, and it may be expected that a number of this character will be built during the next five years.

Natural Rock Cement.—The conditions in the industries producing cement of the natural or Rosendale varieties were not very satisfactory during 1898. Prices were generally lower, and in certain districts were certainly at times below the cost of production. It would seem evident from this that Portland cement has to some extent displaced natural cement, and it would not be surprising to find during the next few years a decline in the production of the latter. This would naturally be the case as concrete construction displaces masonry. While natural cements often answer very well for mortar in masonry, they are not suitable for rapid work in concrete construction, which is the condition under which much of this work must be done. In addition, the reliability of Portland cement, its capacity for surviving rough treatment and yielding good results in inexperienced hands is commending it to architects and engineers in much the same way that the softer grades of steel now so commonly employed for construction have commended themselves. This is leading to the very general requirement of Portland cement in specifications because of the confidence in results which this product inspires.

Plant.—In machinery and other plant nothing has been developed during the last year. The tendency has been to large machines and heavier types of construction in the entire equipment of cement works. The service is hard and continuous, and an investment in ample but economical power and in heavy machinery is found to effect a considerable reduction in the cost of repairs and renewals.

Chemical and Physical Tests.—In the chemistry of cement and in the theory of cement production nothing noteworthy has appeared during 1898. There is an evident disposition to tolerate the presence of magnesia in cement in larger quantities, and to hold that it may be disregarded in the calculation of the relative proportions of acids and bases in cement analyses. Mr. S. B. Newberry has taken very decided grounds on this point, declaring that he has made cements from compositions in which the proportion of lime by equivalents approached the theoretical quantity required to combine with the content of silica and alumina, that he has then added to these mixtures considerable quantities of magnesia, and that the results which he has obtained from the calcination of such mixtures have been quite satisfactory, the cement being sound under approved accelerated tests. Some experiments made by myself have given similar results at short periods. It is important, however, to remember that the injurious effects of magnesia, if any, are likely to be manifested at the end of a year or more, and are much more evident in mortars which are under water than those which are subjected to air exposure.

The American Society of Civil Engineers now has a very competent committee considering methods of cement testing. This committee has already done a considerable amount of work and will no doubt formulate a new system of manipulation for cement tests, which will prove very acceptable to both engineers and manufacturers.

An interesting series of comparative tests of different forms of briquettes, reported by Prof. Jerome Sondericker before the Boston Society of Civil Engineers, shows that the style of briquette which is in general use in this country is quite as satisfactory as any.

It is interesting to note here the composition of the various brands of Portland cement made in Russia, as shown in the following table taken from the Riga *Industrie Zeitung* (1897, XXIII., p. 153).

	Schmid.	Port Kunda.	Liphard.	Gluchoosersk.	Wysoka.	Noworossisk.
	%	%	%	%	%	%
CaO	62·1	62·82	62·50	63·00	64·85	66·33
MgO	2·1	2·09	1·75	1·04	1·21	0·57
K_2O+Na_2O	1·3	0·92	0·50	1·60	0·53	....
SiO_2	22·0	22·42	22·10	22·07	21·81	25·99
Al_2O_3	6·5	6·28	6·25	6·59	8·05	5·77
Fe_2O_3	3·2	3·62	3·70	3·41	3·67	
SO_3	1·1	1·29	1·20	1·53	0·16	0·55
Moisture	1·7	....	2·00	....	....	1·18
Hydraulic modulus	2·03	1·97	2·00	1·98	1·98	2·09

CHROMIUM AND CHROME ORE.

THE chrome iron ore industry of the United States, which for many years had been confined to California, came almost to a standstill in that State in 1897, when the production amounted to only 50 long tons and in 1898 the business ceased completely; but mining was resumed in 1897 in the Pennsylvania-Maryland region, which was formerly productive, in Chester County, Pa., about 2 miles northwest of Sylmar on the interstate line, and was continued in 1898. At this place chromite is obtained by washing the deposits of sand and gravel which have accumulated on the serpentine rock, the original source of the chromite.

PRODUCTION, IMPORTS AND CONSUMPTION OF CHROME ORE IN THE UNITED STATES.

Year.	Production.			Imports.			Consumption.	
	Quantity. Long Tons.	Value Per Ton.	Value.	Quantity. Long Tons.	Value Per Ton.	Value.	Quantity. Long Tons.	Value.
1894..........	3,285	$12·14	$39,980	3,886	$9·87	$38,364	7,171	$78,344
1895..........	1,553	10·81	16,795	5,230	15·84	82,845	6,783	99,640
1896..........	702	11·07	7,775	8,869	21·13	187.400	9,571	195,175
1897..........	150	10·33	1,550	11,566	16·11	186,313	11,716	187,863
1898..........	100	10·00	1,000	16,304	16·70	272,234	16,404	273,234

As in previous years the domestic supply of chrome ore was obtained chiefly from Turkey, although larger amounts are being imported from Canada, Newfoundland and New Caledonia. A good deal of the Canadian ore is of lower grade than the standard prescribed by the chemical manufacturers, and this is employed for the manufacture of ferrochromium and for use in furnace lining. At Pittsburg the stipulated grade of chrome ore for the latter purpose is 45% Cr_2O_3, but ore with as little as 40% is sometimes taken. In the manufacture of ferrochromium ores of even lower grade may be used, it is said.

THE WORLD'S PRODUCTION OF CHROME ORE. (IN METRIC TONS.) (*a*)

Year.	Bosnia.	Canada.	Greece.	New Caledonia (*b*)	New-found-land.	New South Wales.	Norway.	Russia.	Turkey. (*b*)	United States.
1893......	965	*Nil.*	1,820	242	*Nil.*	*Nil.*	*Nil.*	14,585	13,780	1,655
1894......	1,808	907	6,000	1,042	*Nil.*	3,083	*Nil.*	7,537	20,250	3,338
1895......	707	2,882	2,740	8,014	44	4,297	190	21,014	21,050	1,578
1896......	443	2,124	1,600	16,018	1,031	3,914	(*c*)	(*d*)	20,137	713
1897......	396	3,392	563	9,054	3,084	3,434	(*c*)	(*d*)	11,551	152

(*a*) Fom the official statistics of the respective countries, excet for the United tates, which are our own. (*b*) Exports. (*c*) None reported. (*d*) Statistics not yet published.

Canada (By J. Obalski).—In 1898 work was carried on as usual in the township of Coleraine, Province of Quebec, in the close vicinity of the Quebec Central Railway. The Coleraine Mining Co., which heretofore had been leasing part of its lands, covering 5,000 acres, has decided to work on its own account, and started developments about 1·5 miles from Black Lake station, where a mine has been equipped with a small steam plant. About 30 men were employed during the year. The mine, which is worked open cast, is now 90 ft. deep, and has several feet of chrome ore in sight at the bottom. The company will extend developments in this mine in 1899, and will also operate on the other side of Black Lake and near Caribou Lake. It has erected a 35-ton concentrating mill near Black Lake. It is well established that the quantity of ore through this district is very large, and though a part of it requires to be concentrated previous experiments have proved that this is easily done. Besides that by the Coleraine Mining Co., some work has been done by several other parties who have shipped their products mostly to Pittsburg and vicinity. The important chrome deposit near Lake St. Francis, owned by Leonard & Co., was very little developed in 1898. A company known as the Eastern Townships Chrome Iron Mining and Milling Co., has been organized with the view of concentrating chrome ore by custom work. A building has been put up 2 miles from Coleraine Station, at Chrome Siding, and preparations made for using the water of Caribou Lake, situated 100 ft. higher and 1 mile distant. The shipments from Quebec in 1898, mostly to Pittsburg and vicinity, were 1,805 gross tons of an estimated value, at the mine, of $25,000, part being of 50%, and the remainder of middle grade, about 45%. In 1897 the shipments were 2,340 gross tons. At the end of 1898 there remained on hand about 3,000 tons of middle and low grade, besides a much larger quantity of milling material. The prices were regularly good, as well as the demand, and the small shipment of 1898 must be accounted for by the reason that the miners were waiting for the concentrators to be built. With the new improvements it is expected that shipments of first-class ore will be larger in 1899.

Cuba.—According to Raimundo Cabrera, chrome iron is found in the district of Baracoa, where three mining claims have been taken up. The Valhourat mine shows mineral assaying as high as 58% Cr_2O_3. Other mines of this mineral have been located at Mayari and at Sagua de Tanamo, in both of which localities it is very abundant.

New Caledonia.—The export of chrome ore forms a large and increasing item in the mineral production of this island. It is widely diffused over the whole of the southern two thirds of the island, the chief shipping points being at the southern extremity and northward along the west coast. Bowlders of chrome ore are exceedingly plentiful in every part of the serpentine area. On several plateaus there are hundreds of acres covered several inches deep with a layer of granules varying in size from a large pea down to partridge shot, which assay from 20 to 35% Cr_2O_3. In the same vicinities are mounds of bowlders, piled up to sometimes 25 ft. high, not yet disintegrated. These agglomerations of granules weigh from a hundredweight or so up to a couple of tons.

In July, 1897, the price of chrome ore at Noumea was 80s. per 2,240 lb. on a 50% basis, with a variation of 2s. per unit up or down.[*]

The exportation of chrome ore from New Caledonia for the first nine months of 1898 was 5,562 tons, which showed a falling off in the rate as compared with the previous year, this being due to a slacker demand. The chrome ore deposits of New Caledonia occur near the surface, and are worked in benches like a quarry. Little adits, called prospecting tunnels, are driven here and there at random. The chrome ore is called "alluvial," but it is not really so. Beneath this alluvial is the "rock" or solid chrome ore. The alluvial is that which occurs in the decomposed rock near the surface. It is preferred by the consumers because it is separated more readily from the gangue than is the solid ore. It consists chiefly of small grains of chromite in decomposed serpentine. It is washed in long sluices similar to those employed in recovering gold from beach sand, and is generally dressed up to 51 or 52% chromic oxide.[†]

Newfoundland.—The shipments of chrome ore from Newfoundland in 1898 amounted to 724 tons of 2,000 lb., averaging 55% Cr_2O_3. This was entirely dressed ore, the product of the Halifax Chrome Co. at its mines at Port au Port.

MANUFACTURE OF METALLIC CHROMIUM AND CHROMIUM SALTS.

The manufacture of metallic chromium (which heretofore has been limited to Germany, where it has been made by the Elektrochemischen Werke, of Bitterfeld) has now been undertaken in the United States at Philadelphia, where there was a make of 930 lb. in 1898. Ferrochromium is made by three concerns—at Philadelphia, Pa., Brooklyn, N. Y., and Phœnixville, Pa. Chromic salts are made by the Baltimore Chrome Works of Baltimore, the Kalion Chemical Co. of Philadelphia, and the Mutual Chemical Co. of Jersey City. The last began the manufacture in 1898, using ore from New Caledonia. In 1899 the Anatron Chemical Co., of Elizabeth, N. J., will begin the manufacture of bichromates, using ore from Asia Minor.

The Goldschmidt Process.—The most important development in the metallurgy of chromium in 1898 was the process of reducing this metal from its oxide by means of alumina, invented by Claude Vautin, of London, and Dr. Hans Goldschmidt, of Essen, Germany.[‡] This consists in the utilization of the great heat which is developed by aluminum in the reduction of metallic oxides. The heat of formation of alumina is theoretically about 360,000 calories, and in certain experiments a temperature, estimated to be at least 3,000° C., has been attained, and by means of this high heat work can be done which heretofore has required the electrical furnace for its performance. So great is the temperature that the slag (consisting of pure alumina) is sufficiently fusible to allow the reduced metal to sink through it. The slag may be used for the reproduction of aluminum, or it may be marketed as an abrasive, being an artificial crystalline corundum, which it resembles in every respect. In reducing a metallic oxide it is unnecessary to heat the whole charge, it being sufficient to

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[*] H. W. Edwards, *Engineering and Mining Journal*, 1898.

[†] F. Danvers Powers, *Australian Mining Standard*, Dec. 22, 1898.

[‡] *Journal of the Society of Chemical Industry*, June 30, 1809; *Stahl und Eisen*, XVIII., 1898, p. 468.

start the reaction (by a special cartridge containing barium peroxide and magnesium ribbon) whereafter the local heat causes the whole charge to react, the heat passing gradually through it. The reaction can be kept up as long as desired by feeding into the crucible more of the mixture of chromic oxide and aluminum. Since no carbon is used in any form, metals free from that element are obtainable, while by having an excess of the oxide the product is not contaminated by aluminum. In the production of pure metal it is necessary to use pure aluminum, which is employed in a finely divided state. In the production of chromium the smelting is performed in a crucible lined with magnesia, an ingot weighing 25 kg. having been made in one smelting. It will probably be feasible to design a furnace for continuous operation. The cost of production is about 2 marks per kg., not allowing anything for the value of the slag. The chromic oxide which is employed as the source of the metal is obtained by reducing crystallized chromates with coal. Metallic chromium can be made by this process so as to compete with ferrochromium, and the demand for it is said to be large already. It is preferred to ferrochromium on account of its greater certainty of composition and freedom from carbon.

Iron-Chromium Carbide.—At the meeting of the Academy of Sciences in Paris, October 3, 1899, MM. Carnot and Gontal stated that by treating sesquioxide of chromium, iron and petroleum coke in the electric furnace they had obtained a double carbide of iron and chromium. It was in the form of needles of metallic appearance, and the color of nickel. The needles were brittle and had a sp. gr. of 7·22. In a current of air or steam they oxidized at 500° C.

CLAY.

THE manufacture of brick and other clay products is one of the most important branches of the mineral industry in the United States, the aggregate value of the output being surpassed only by those of coal and iron. This is not surprising when it is considered that brick is the most important element in all our permanent construction, and its use is increasing not only through the natural extension of building but also through the replacement of old wooden work with material of greater solidity and durability. The brick and clay production of the United States in 1897 and 1898 is summarized in the following table:

SUMMARY OF PRODUCTION OF BRICK AND CLAY WARES IN 1897 AND 1898.

Kind.	1897.			1898.		
	No. of M.	Value.	Per M.	No. of M.	Value.	Per M.
Common brick	4,659,238	$23,262,077	$4·99	5,049,308	$25,457,236	$5·04
Front brick	369,266	4,293,684	11·63	393,327	4,077,019	10·37
Fire brick (*a*)	287,359	4,538,619	15·79	347,275	4,163,103	11·99
Paving and vitrified brick	533,806	4,269,134	8·00	567,688	4,589,843	8·08
Other clay building material (*b*)	....	5,898,422	....	....	5,439,914	....
Sewer pipe and drain tile	....	7,416,051	....	....	7,493,428	....
Crude clay, stoneware and misc. man'f'res (*c*)	....	6,809,540	....	....	*e*7,250,000	....
Totals	....	$56,487,527	....	....	$58,470,543	....

(*a*) Not including silica brick. (*b*) Including terra cotta lumber, hollow building tile or blocks, roofing tile, floor tile, and all other clay building material. (*c*) including the value of common stoneware and various miscellaneous clay manufactures and crude clay used in pottery, for laying fire brick, in paper making, as burnt clay railway ballast, for the manufacture of gas retorts, glass pots, zinc retorts, etc. (*e*) Estimated.

A statistical investigation of the brick and clay industry is burdened with difficulties which are not easily overcome. The number of producers is very large, our list comprising upward of 8,000 names scattered through every State and Territory of the Union. Most of them are individuals, who operate on a small scale and in many cases irregularly. The ease with which a clay bed can be opened and a brickyard established, together with the small outlay of principal required, which makes the abandonment of a yard a matter of little consideration, combine to give this industry a transitory character and add to the labors of the statistician. The difficulty is increased, moreover, by

PRODUCTION OF BRICK AND CLAY BUILDING MATERIAL IN THE UNITED STATES IN 1897

(In Thousands.)

States.	Building Brick. Common.		Building Brick. Front.		Fire Brick. Alumina.		Paving Brick.		Other Clay Building Material.
	Quantity.	Value.	Quantity.	Value.	Quantity.	Value.	Quantity.	Value.	Value.
Alabama (e)	45,000	$236,250	....	....	400	$80,000	2,000	$20,000	$500
Arizona	3,867	21,733	200	$2,583	....	....	....	....	e500
Arkansas	23,000	138,000	e1,000	e8,500	....	....	2,765	27,650	500
California	a97,468	a571,240	....	....	e500	e10,000	e100	e1,100	....
Colorado	25,000	125,000	10,000	100,000	2,000	40,000	700	7,000	20,000
Connecticut	116,002	596,628	2,625	21,004	3,500	70,000	....	....	29,018
Delaware	3,000	18,000	192	9,583	....	....	383	11,021	....
Dist. of Columbia	37,160	193,232	2,640	28,142	....	....	774	6,966	e500
Florida	22,821	128,464	....	....	....	....	....	....	....
Georgia	94,814	423,211	4,479	51,233	1,933	29,000	902	20,880	38,667
Idaho	1,001	6,579	....	....	....	....	....	....	....
Illinois	457,199	2,024,328	34,735	374,169	8,266	82,745	111,749	907,450	e700,000
Indiana	157,225	719,474	11,243	109,476	809	8,136	39,277	212,660	350,000
Iowa (b)	117,881	594,271	9,899	71,301	239	4,630	49,762	358,848	....
Kansas (b)	19,548	104,257	1,948	10,241	100	1,000	18,378	132,222	e10,000
Kentucky	71,000	367,000	889	8,570	15,003	187,037	380	3,800	85,000
Louisiana	73,704	353,482	8,000	56,000	....	....	1,000	5,000	1,300
Maine	61,414	383,986	1,897	16,414	....	....	....	....	....
Maryland	94,644	542,680	6,084	114,431	19,772	296,708	6,597	49,741	36,636
Massachusetts	250,952	1,300,473	5,839	91,593	2,022	77,667	....	....	174,767
Michigan	99,250	496,250	2,000	10,000	....	....	3,500	35,000	e3,500
Minnesota	80,000	360,000	3,000	32,000	500	5,000	530	5,900	36,000
Mississippi	27,822	149,771	462	4,616	839	8,393	....	....	750
Missouri	171,702	844,855	7,776	61,140	27,836	362,602	31,579	286,188	122,705
Montana	16,115	91,760	1,711	28,228	1,675	36,916	....	....	....
Nebraska	51,501	331,904	899	7,050	....	....	1,313	9,743	3,569
New Hampshire	89,550	444,377	1,108	5,383	792	7,917	950	6,650	400
New Jersey (b)	283,678	1,260,342	26,979	519,077	13,827	256,652	575	7,600	2,718,600
New Mexico	250	2,500	....	....	....	....	....	....	....
New York	862,732	3,397,305	10,384	181,117	18,073	360,958	19,849	209,124	e642,273
North Carolina	55,761	285,428	843	7,946	....	....	1,746	18,996	2,754
North Dakota	10,550	63,300	....	....	....	....	....	....	....
Ohio	236,230	1,306,124	62,662	760,216	56,908	762,838	97,225	735,764	430,431
Oklahoma (d)	11,408	69,500	....	....	....	....	....	....	....
Oregon	8,586	51,991	519	11,000	71	1,786	....	....	....
Pennsylvania	463,267	2,936,090	96,086	1,151,246	104,082	1,741,862	104,509	865,395	e400,000
Rhode Island	28,000	175,000	....	....	....	....	4,000	48,000	12,000
South Carolina	24,094	112,094	688	5,625	1,250	4,125	625	4,375	1,200
South Dakota	2,440	17,525	80	1,600	....	....	....	....	....
Tennessee	64,829	349,942	1,837	17,114	2,300	35,184	9,438	84,815	23,000
Texas	74,409	451,916	e20,000	e160,000	e2,000	e21,500	e3,669	e32,691	13,458
Utah	12,040	71,250	3,037	28,442	1,200	24,000	....	....	....
Vermont (b)	9,742	50,518	....	....	....	....	....	....	....
Virginia	78,674	464,246	18,797	152,442	143	1,993	2,109	16,022	e31,624
Washington	15,501	108,507	163	2,117	33	....	3,000	30,000	3,000
West Virginia	23,370	127,852	1,825	18,252	1,141	15,970	14,422	108,533	3,259
Wisconsin	84,537	388,942	6,740	55,883	e200	e4,000	....	....	3,120
Wyoming (b)	500	4,500	....	....	....	....	....	....	....
Totals	4,659,238	23,262,077	369,266	4,293,684	287,359	4,538,619	533,806	4,269,134	5,898,422

(a) Statistics reported by State Mineralogist. (b) Statistics reported by State Geologist. (d) Includes production of Indian Territory. (e) Estimated.

the fact that some of the producers do not keep records of their work, and consequently are unable to make accurate reports.

The production of brick and clay building material by States in 1898 and 1898 is presented in the following tables. In these tables the columns "common brick" include only the ordinary red brick that is used generally in building. Under "front brick" we have grouped pressed brick of all colors. Under "fire brick," however, only what is properly termed alumina brick has been included, such silica brick as Dinas brick, which is also properly called "fire brick" or "refractory brick" being omitted, because it is not a clay product. "Other clay material" includes fancy or ornamental and enameled brick, roofing tile, terra cotta, terra cotta lumber, floor tile, hollow

PRODUCTION OF BRICK AND CLAY BUILDING MATERIAL IN THE UNITED STATES IN 1898.

(In Thousands.)

States.	Building Brick. Common.		Building Brick. Front.		Fire Brick. Alumina.		Paving Brick.		Other Clay Building Material.
	Quantity.	Value.	Quantity.	Value.	Quantity.	Value.	Quantity.	Value.	Value.
abama. (e)	50,000	$255,000	1,000	$11,500	4,000	$80,000	3,000	$30,000	$500
rizona	5,800	34,800	e200	e2,500	....	....	....	....	e500
rkansas	22,808	147,102	2,060	20,418	997	14,330	....	....	e500
lifornia	a100,102	a511,362	e4,000	e80,000	e500	e10,000	e100	e1,000	....
lorado (e)	29,224	146,120	7,306	73,060	2,400	48,000	750	7,500	20,000
nnecticut (e)	125,000	657,750	2,500	20,000	3,500	70,000	....	....	78,812
laware	3,539	23,130	....	....	....	....	208	2,080	....
st. of Columbia	34,580	197,791	....	....	....	....	100	900	....
orida	27,978	167,608	169	1,490	....	....	....	....	....
orgia	106,456	465,941	6,301	54,676	2,062	27,491	....	....	55,000
aho	6,467	38,692	2,456	24,560	....	....	....	....	....
inois	481,773	2,312,974	22,105	221,869	8,269	86,422	88,366	719,496	e750,000
diana	159,546	725,887	19,244	194,589	202	1,616	41,072	414,477	e370,000
wa (b)	178,513	1,049,222	6,722	54,752	2,760	5,525	38,478	313,551	22,006
nsas (b)	23,157	146,765	6,575	38,300	611	4,888	26,182	190,735	....
ntucky	77,624	319,250	90	900	17,180	251,517	1,899	18,629	e90,000
uisiana	81,052	408,609	....	....	....	....	....	....	1,500
aine	61,493	279,097	874	5,244	....	....	....	....	....
aryland	96,336	537,449	4,651	55,251	24,455	231,606	5,227	48,233	e38,695
ssachusetts	221,589	1,139,339	2,190	27,651	2,594	79,617	....	....	249,056
chigan	120,626	467,118	15,726	82,533	....	....	34	510	9,787
nnesota	120,797	571,417	5,917	65,482	....	....	169	2,028	39,989
ssissippi	37,452	201,257	416	2,915	....	....	....	....	e800
ssouri	121,832	530,913	9,439	79,097	11,703	146,738	22,558	182,243	e93,166
ntana (b)	41,541	272,805	625	17,132	2,363	91,866	....	....	....
braska	65,719	405,364	136	952	....	....	2,330	25,752	e5,000
w Hampshire	78,752	275,858	881	8,813	1,459	12,156	....	....	17,512
w Jersey (b)	344,387	1,578,983	35,983	824,820	21,620	461,127	200	2,800	1,910,857
w York	817,481	3,786,833	18,326	179,132	e19,000	e380,000	18,233	177,968	e625,000
rth Carolina	e61,337	e317,113	e923	e8,695	....	....	e1,921	e20,900	e3,029
rth Dakota	33,413	204,672	138	2,070	....	....	....	....	....
io	311,436	1,407,517	54,629	479,097	76,975	808,882	157,251	1,138,903	518,741
lahoma (d)	6,681	44,212	....	....	....	....	....	....	....
egon	10,049	64,499	367	9,325	....	....	....	....	....
nnsylvania	527,787	3,272,024	84,404	773,100	116,848	1,139,857	69,965	624,946	e460,000
ode Island	e30,000	157,800	1,000	8,000	....	....	4,000	48,000	12,000
uth Carolina	29,181	138,823	287	1,722	719	5,752	....	....	e1,200
uth Dakota	7,175	47,779	....	....	....	....	....	....	....
nnessee	74,536	371,843	4,095	28,643	e2,500	e20,000	146	1,119	....
xas	83,847	478,009	e22,000	e170,500	e2,000	e20,000	....	....	e15,000
ah	17,380	104,661	19,400	159,620	1,291	12,910	....	....	....
rmont	11,437	53,774	60	241	....	....	....	....	264
rginia	96,208	579,688	20,097	215,254	567	5,303	11,215	108,142	e40,000
ashington	11,799	73,985	....	....	....	....	....	....	e2,500
est Virginia	b7,250	b43,500	b350	b3,500	b20,500	b143,500	b74,264	b509,771	e5,000
isconsin	87,749	438,901	9,685	69,616	e200	e4,000	20	160	e3,500
yoming (b)	400	4,000	....	....	....	....	....	....	....
Totals	5,049,308	25,457,236	393,327	4,077,019	347,275	4,163,103	567,688	4,589,843	5,439,914

(a) Statistics reported by State Mineralogist. (b) Statistics reported by State Geologists; the wide variation the figures for West Virginia from those of 1897 is due undoubtedly to differences in the classification. (d) cludes production of Indian Territory. (e) Estimated.

uilding blocks, etc. All the clay material other than brick which is now
sed so extensively in the construction of fireproof buildings is included under
iis classification. In the preparation of the accompanying tables the statis-
cs, collected by the State Geological Surveys have been used so far as avail-
ble, the States of Iowa, Kansas, Montana, New Jersey and West Virginia
eing especially prominent in this line of investigation. There is apt to be a
iriation in the results of any statistical inquiry concerning this subject from
ear to year and State to State, not only between the results of two statisticians
r the same year, but also in the results of the same statistician in different
ears, owing to uncertainty as to the classification of certain products, not
ily by the statisticians, but also by the producers themselves in making their
turns. For example, a producer may sometimes be in doubt as to whether

his product should be reported as common brick or front brick, or perhaps as paving brick or fire brick. In general the distinction between common and front brick is sufficiently sharp, since there is usually a wide gap between their value per thousand, although even this is not uniformly the case. Between front brick and fire brick, however, there is no sharp dividing line, because there is now a good deal of pressed brick made of fire clay and used in the construction of huoses which is manufactured by concerns making brick for furnace construction, etc., out of the same clay.

In collecting statistics of the clay production other than brick and building material the difficulty increases, owing to the practical impossibility of obtaining an accurate account of the crude clay used for these purposes and the misleading character of returns based on the finished products. Owing to the multiplicity of kinds and wide range of character of these, it is unsatisfactory to foot up the values of the manufactured products; it would be useless to base statistics of the iron and steel industry upon manufactured products which differ so widely in values per pound as watch springs and anchors; it is equally useless in reporting the pottery industry to include common stoneware and fancy porcelain in one general statement as to the value of the product. The manufacturers of porcelain and chinaware, moreover, consume not only ball and slip clay and china clay of domestic production, but also a considerable amount of domestic flint and feldspar, together with large quantities of imported material. Since it is impossible to separate from the value of the pottery production the value of the imported material used therein, and also to obviate the misleading variability of the labor and capital component, we have based our statistics of the clay used in pottery and for certain other purposes where there are similar difficulties upon the raw material in so far as data could be obtained, but of necessity this statement is largely estimated.

Our report of crude clay therefore includes ball and slip clay, fire clay used for the manufacture of glass pots, zinc retorts, gas retorts, clay used for laying fire brick, clay burned for railway ballast, and fire clay and china clay used in paper making and for a multiplicity of manufactures, such as flower pots, crucibles and assay supply, tobacco pipes, etc., etc. In reporting the production of sewer pipe and drain tile the statistics are based on the manufactured articles, which are of a comparatively crude character. These statistics can only be classified satisfactorily as to total values, since there is no uniform measure as to the quantity. The largest part of the production of drain tile and sewer pipe comes from the State of Ohio.

Besides the production of fire brick reported in the preceding table, 33,449,000 silica brick, valued at $590,254, were made in 1898, against 20,233,000, valued at $323,561, in the previous year. Pennsylvania is the largest producer of this class of refractory brick.

THE CLAY INDUSTRY OF MASSACHUSETTS.

BY CHARLES L. WHITTLE.

THE use of clay in Massachusetts for the cruder purposes, such as bricks, and even for the coarser forms of pottery, dates back many years. The earliest plants naturally manufactured only common brick; but the middle part of the 18th century witnessed an advance in the establishment of small pottery works near Quincy. From this small beginning there have grown up many industries that use clay as their raw material. Their condition to-day is the more remarkable for the reason that such progress as has been made has been in the face of greater difficulties than most States have been obliged to overcome. Not alone have the native clays been used in this advancement, for it has been necessary, or at least expedient, to bring in certain clays from other States, or from abroad, to permit the manufacture of special lines of clay products, incurring extra transportation charges not borne by clay manufacturers in most other places in the United States.

In striking contrast to the prevalence of china clays in the States south of Pennsylvania, Massachusetts is particularly wanting in occurrences of this kind. Clays occurring as the decomposed remains of feldspathic dikes, that have not undergone water transportation, are only infrequently found here, at least in sufficiently large quantities to have any economic value. Several localities are known to have small deposits of this nature, but only in the town of Blandford has there been found china clay of commercial value. Deposits of this kind represent a portion of the old land surface, deeply decayed by the process of weathering, that existed before the glaciers carried away the greater part of all disintegrated material. By some fortunate conditions the Blandford kaolin was spared the fate that overtook nearly all such deposits that existed in New England prior to the glacial period.

As a result of glacial conditions, the workable clay beds of Massachusetts have a more general distribution over the State than would otherwise be the case. In the many lakes and ponds, now filled to meadows or marshes, there are commonly found small but workable deposits due to these conditions. They are scattered pretty uniformly over the central and eastern portions.

The great center of the brick industry in Massachusetts is situated about the city of Boston, in the towns of Medford, Revere, Cambridge, Belmont and Everett. Boston itself, though not possessing many beds that are operated at the present time, is the seat of several large industries that use both native clays and those brought from other parts of this country. Other important centers are situated in southeastern Massachusetts, near Taunton and Bridgewater; in the western part of the State, about Holyoke and Springfield; in the northeastern part, about Newburyport and Danvers and extending up the Merrimac River as far as Lawrence. At all these localities the principal product is common, unpressed brick, while at a few points only, outside of Boston and vicinity, are pressed and enameled bricks manufactured.

About Boston, however, large quantities of pressed brick are made both from local and other clays. The smaller plants still adhere to hand presses in the manufacture of the higher grades of brick; but the larger concerns use composite molds pressed by steam power. After the molds are filled they are placed on shelves suspended from long tramways which pass by the presses and then outside to the drying yards, where they are removed by the workmen. The bricks thus made are of a very good quality, uniform in size, with smooth surfaces, and find a market mainly in the cities comprising the metropolitan district. The manufacture of many varieties of fancy brick is carried on in South Boston, where the most important plant in the State is situated. The South Boston company turns out several grades of face brick, both of Roman and standard sizes, using for this purpose a mixture of New Jersey and Gay Head (Martha's Vineyard) clays. The product, consisting of pale yellow and brown bricks, with various effects produced by iron oxides, derived from common pyritic impurities that exist in many of the seashore clays, and the addition of manganese dioxide, will compare favorably with similar bricks from New Jersey and elsewhere. Another concern in the same city produces white, glazed bricks and tiles also made out of New Jersey clays, for which there is a growing demand. Large quantities of these enameled tiles have been used in the Boston subway and in the Southern Union Station just completed. In the town of Somerset an enameled brick similar to that used in the subway is manufactured.

The brick industry, which is able to command a great and growing market, like the metropolitan district of Boston, has already so strongly intrenched itself that its dislodgment is not to be apprehended. It is true that the introduction of terra cotta fronts affords the strongest ground for apprehension that the demand will materially lessen, and it is a fact that the growing use of fancy, mottled face brick is to a slight extent restricting the sale of the native pressed and the better varieties of common bricks. A cheapening of the methods of extracting and working some varieties of building stones has also led to a somewhat more severe competition in the business; but to offset this there is taking place a rapid enlargement of the local market.

It is worthy of note that not only is there a rapid growth in the demand for plain, white, enameled bricks and tiles, but the demand has led to such improvements in the quality of the production that the local market is no longer dependent on foreign manufacture. Not only are plain, undecorated tiles receiving attention, but a growing appreciation of the possibilities of faience ornamentation is leading to much artistic improvement along lines of architectural terra cotta, for interior and exterior use as well as for mantels, wainscots, art tiles and panels. The Grueby Faience Co. is successfully building up a business furnishing enameled terra cotta after specifications submitted by architects, or by themselves, as to designs and color effects. Work of this kind has never been attempted before in this country. Rooms recently decorated in this manner deserve the highest recommendation not only as regards detail of design, but for the harmonious and artistic color effects produced. While it cannot be said at present that there is an extensive market for products of this kind, an effort is being made to educate the people in the artistic merits of this imperishable and long-neglected form of interior decoration. With an awakening of interest in such clay products, giant strides in the purely art side of the industry may be looked for. In fact, Massachusetts, in common with many States, is recognizing the worth of plain undecorated terra cotta as shown by the large number of buildings ornamented with this material which have been built within the last two or three years and which are in course of construction at the present time.

A branch of the ceramic art that has attained great excellence and prominence is the manufacture of terra cotta fireplaces and mantels, possessing effects variously styled gray, Pompeian and old gold. The local demand for such clay products as terra cotta chimney tops, fire brick, gas retorts, locomotive fire-box linings, ashlar blocks and tiles of various kinds is in part supplied by products manufactured in Massachusetts. In Taunton several plants are engaged in the manufacture of stove linings for the local market.

Massachusetts has perhaps made a new departure along the line of bas-relief art tiles, soda fountains, lamps and stoves. The introduction of the dry press process has permitted a more economical and reliable method of procuring articles of this type. Native clays are not used for this purpose; the potter's clay is brought from Perth Amboy, the kaolin from Brandywine Summit, Pa., and the quartz from Cheshire, Mass. Beautiful light and shade effects are made possible by a scientific molding of bas-relief patterns, whereby the glaze to be added is allowed to gather in different thicknesses corresponding to the tone desired. The production of soda fountains has assumed large proportions and their artistic merit and serviceable nature, combined with relative cheapness, has led to their introduction in effective competition with older firms using fancy marbles and travertines. The company manufacturing this line of clay products (Low Art Tile Co.) also makes a great variety of artistic pottery, such as coffee heaters, vases, jardinieres and teapot stands, using the latest and most improved methods of manipulation. A large establishment has grown up, reporting orders sufficient to keep the whole plant employed to its fullest capacity.

Among the various uses of clay, there should be mentioned the manufacture of a low grade of bricks from bowlder clay, or hard pan. In the towns situated on Cape Cod, a limited quantity of clay is used by the Massachusetts Highway Commission as a binder in the construction of State highways, where, for economical reasons, it is found advisable to build roads out of the sand covering that region. This use may prove only temporary. Another new departure consists in the manufacture of fireproofing out of common brick clay in the city of Revere. This product is finding a ready market, and its successful introduction in buildings going up in Boston at the present time will probably encourage others to go into the business. Since the native clays have thus been shown to be adapted for the manufacturing of fireproofing, a new field has been opened for those engaged in their exploitation.

The paucity of certain types of potter's clay in Massachusetts prevents this line of the industry from making such rapid strides as otherwise would be possible, and those concerns that are in operation are able to retain a foothold only by reason of the somewhat isolated character of the market about and including the city of Boston, the importation of foreign clays, or especial advantages attained by the manufacture of novel or exclusive products. There can be no doubt that, unless other clays are discovered, this branch of the fictile industry will never attain the importance possessed by many establishments in other States; or unless those who are in the business keep well to the front, or a little ahead in the excellence and artistic merit of their productions. It is to be noted that this fact appears to be appreciated, and affords a basis for the belief that the strong position occupied by the few who are in the business will be maintained and strengthened in the future.

Since the Centennial Exhibition in Philadelphia and the World's Fair, the interest in all branches of the fictile arts has been rapidly on the increase, and it is manifest that from now onward great progress will be made. Already this country leads the world in the excellence of its decorated tiles and panels; and this too in spite of the fact that our efforts in this direction have been limited to something like twenty years. Massachusetts stands foremost in this advancement, and perhaps no single firm deserves more credit than the Low Art Tile Co., of Chelsea, which has brought its unique productions up to a high order of excellence by its own unaided efforts.

With the growth of interest in art tiles and panels there has also taken place a rapid development in art pottery. While the Rookwood ware has been attaining its present unsurpassed artistic merit in Cincinnati, and the Belleek ware has been successfully fabricated in Trenton, Massachusetts has made nearly as great advances but along somewhat different lines. This progress has been brought about, notwithstanding the unreasonable and disloyal prejudice against American pottery, by individual effort and perseverance, and by the support of certain public-spirited citizens who have become patrons of the art.

Japanese "crackle" ware is now successfully made by the Dedham Pottery Co., which is the successor of the Chelsea Keramic Art Works, founded in 1886. This ware has a gray body with underglaze decoration in blue. The great demand for the ware is for plates, although many forms of vases, etc., are made. In an effort to produce the famous Chinese "Sang de bœuf," the company now produces a ware, which, if not a duplicate of this Oriental color, is nevertheless very beautiful. Vases of this ware are made in different shades of red which have in some cases a sheen of irridescent colors, and in others there is a mottled effect of this kind. The most artistic vase is one of deep rich red resembling ox-blood but having a certain liver-brown cast and possessing a faint irridescent mottling.

In connection with architectural terra cotta and art tiles, the Grueby Faience Co., of Boston, is now producing a unique and beautiful ware which is attracting attention not only in this country but in Europe. This is an enamel ware made in many plain colors and fashioned by hand on a potter's wheel, the relief decoration being added afterward. The color which gives prominence to the ware is a green tone simulating closely a remarkably rich copper bronze. The enamal has a dull mat finish which heretofore has never been successfully produced as a result of direct firing.

THE PRINCIPLES OF FIREBRICK MANUFACTURE.

By Charles Ferry.

A complete chemical analysis of a specimen of firebrick or fireclay is a valuable means for determining not only its refractory nature, but also its adaptability to a particular class of work. As the scientist applies refractory material of different chemical composition to practical test, he becomes convinced that the chemistry of clays and their burned products furnishes the most reliable and extensive data as a guide to the brick manufacturer.

If we investigate the problem from a chemical standpoint it appears that each element, even if present as the smallest trace, either works to the advantage of the refractory material or has a tendency to depreciate its value. If we make up a number of test lots of firebrick which differ in composition only with respect to one element, and put them into service in the same furnace, we shall see that the presence of this particular element in varying quantities has a decided effect upon their refractory property. This is not merely a theory; practical tests have proved it to be a fact. Before beginning to work intelligently and successfully along this line, however, it is necessary to know the relation which each chemical constituent bears to fireclay, and how any change in the combination is going to effect the efficiency.

I think that we are apt to expect too much of chemistry as applied to refractory material, for it is true that the chemical composition does not, and indeed cannot, be made to indicate the exact temperature at which a given specimen of brick or clay will succumb to the action of heat; but, on the other hand, if we join with these chemical facts such physical data as it is possible to gather by a study of the structure of the brick or clay we are in a position to draw reliable conclusions.

Firebrick and fireclays are composed of kaolinite, combined water, fluxing impurities and quartz (sand). These terms are used collectively and comprise all of the elements which enter into clay; but to discuss the absolute merits of a brick or clay mixture we must consider each chemical substance separately, because the fluxing bases which include ferrous and ferric oxides, lime, magnesia, and the fixed alkalies, all capable of forming fusible silicates, differ in their ultimate effects. Small laboratory furnace tests—even those for determining the supposed absolute temperature at which brick mixtures or clays will fuse, are of less practical value than a complete chemical analysis of the refractory in question.

The various methods for estimating the heat-resisting power of brick or clay in a direct manner are an attempt to arrive at absolute results; but if any firebrick manufacturer allows himself to be guided by them alone in classifying and adapting his various grades of firebrick to the demands of the trade, he will find that practical results and the conclusions which he draws from his small heat tests will often fail to harmonize. If I were obliged to forego either chemical or physical data, I should sacrifice the latter with much less hesitation than the former; in other words, I believe that the chemical analysis is the more important key to success to the firebrick manufacturer.

Now let us consider each constituent which usually appears in an analysis of fireclay, and a few of the facts which such a report may tell. In the first place I will tabulate several analyses of different grades of firebrick which have been brought to my attention:

Substance.	A.	B.	C.
	Per Cent.	Per Cent.	Per Cent.
Combined silica (Si_2O)	51.10	23.16	11.62
Alumina (Al_2O_3)	44.27	20.10	10.10
Ferric oxide (Fe_2O_3)	0.22	0.37	0.47
Lime (CaO)	0.30	0.21	0.11
Magnesia (MgO)	0.12	0.13	Trace.
Alkalies (Na_2O+K_2O	2.72	1.06	0.95
Quartz (sand)	1.91	55.24	77.07

The first two substances are reported separately, but in the aggregate represent the quantity of kaolinite present in the original clay, in which the "combined" silica, alumina and water are united chemically, forming hydrous aluminum silicate, the mineral kaolinite or china clay.

The water of constitution is evolved when a green brick is burned in the kiln, causing it to shrink. This tendency can be anticipated by observing the percentage of water present according to chemical analysis. Many brick mixtures check or become cracked in the kiln as a result of too great shrinkage, which occurs with clay mixtures of too high a tenor of combined water, but after some experience the manufacturer can determine from a chemical analysis of the clay if it requires the addition of anhydrous material (sand or biscuit) to bring it down to a normal shrinkage, which has been found to produce satisfactory wares, and can thus obviate the difficulty. These troublesome features relative to shrinkage are more characteristic of the plastic clays than of the rock or flint clays.

Kaolinite is one of the chief components of fireclay. Its silica, unlike that of quartz, being combined already with a base, does not unite as easily as quartz with other metallic oxides. Consequently, while kaolinite and quartz both have extremely high heat-resisting power, they give very different results when subjected to chemical action. The quartz (sand), on the other hand, has no shrinkage, so far as can be observed in practice; it is extremely refractory; and in some furnaces gives better results than a pure kaolinite brick. When subjected to extreme changes of temperature the quartz expands and contracts evenly, thus making it possible for the manufacturer to produce a brick which will endure severe physical abuse without any degree of disintegration. Of all plastic clays those highest in kaolinite will shrink the most in the kiln; in fact, it is very difficult to give clay brick of this nature such thorough heat treatment that they will not show a further tendency to shrink when in service. It is easy to see how this molecular contraction, which is likely to continue indefinitely subsequent to its firing in the kiln, may become the source of trouble.

In air-dried clays the percentage of water, free and combined, depends upon (1) its natural physical condition, or the state to which it has become disintegrated, and (2) its chemical composition. The clays highest in kaolinite are the most generously supplied with combined water, and often, by virtue of their extreme fatness, are capable of carrying the greatest quantity of hygroscopic or uncombined water. The air-dried, highly silicious clays, with their small percentage of alumina, are charged with the least amount of moisture. The volume of water required for converting these two extreme varieties of fireclay, fat and silicious, into a sufficiently soft condition to be fit for pugging and molding into brick follows the same law; the fat, aluminous clay will absorb far more than the silicious clay needs to bring it to the same consistency. As a rule, firebrick mixtures are compounded from more than one grade of raw clay, and the manufacturer depends upon his pug mill for blending them into a uniform mixture.

Air-dried clays that have been ground can be mixed more effectually than those semi-dry. Clay mixtures are never pugged in a condition that can be even regarded as liquid; however, when very soft, they are just beginning to approach a liquid, and we are able to see illustrated there an exaggerated tendency of the particles to stick together. This clinging effect is very characteristic of those substances possessing an affinity for water. When the clay has been satisfied in its thirst for water of combination, theoretically it has no affinity for more, but practically it greedily gathers and persistently retains large percentages of water while it is being manipulated in the factory. Its ability to carry so much hygroscopic water, mechanically mixed, seems to impart to the mixture its clinging nature. The only perfect way, then, in which two plastic, ground clays, radically different in composition, can be made to distribute themselves evenly in mixing is to have them beaten up in a thin, watery state.

This discussion serves to illustrate the advantage of pugging clay mixtures in a soft condition. While the results obtained show that mixing ground, air-dried clays is a more effectual plan for blending their constituents uniformly than to pug the raw lumps of clay semi-dry, the mixture is quite as much in need of a final and thorough pugging, in a soft condition, to complete the commingling of its finer particles and to bring into play the real strength of its kaolinite bond. The three general and important results obtained by the addition of water to clay mixtures are: (1) Disintegration; (2) distribution; (3) development of plastic bond. In the preparation of a brick mixture consisting of plastic clays, water is not added for the purpose of developing any chemical changes, but merely to soften the lumps and render the molecular structure of the mass more compatible. As the hard, compact lumps absorb the water their molecules find it more and more easy to move upon one another, until finally they are held together only by their plastic nature and mutual thirst for more water.

There is really no small test by which the user of firebrick is able to determine from two samples submitted for inspection the brick best suited for his purpose. Theoretical tests are of little significance in comparing two varieties of firebrick, unless the conditions are identical, and such as will measure the duration of time for which each will answer satisfactorily. Even if it were possible to adopt as refractory material a brick cut out of granite, we should find at once the scheme impracticable, since such brick could not be cut easily with the mason's trowel. Again, the consumer would complain (and with good reason) if the manufacturer should supply him with brick so friable as to crumble. Firebrick may be faulty in either of these respects, and one is as serious to the mason as the other. The general rules for the production of firebrick from any fireclay are similar, and the real source of an error in the burned ware may be found in the character of the materials used and the methods of factory manipulation.

Usually the manufacturer uses only two general refractory constituents in his brick mixture: (1) Raw fireclays; (2) non-shrinkable refractory substances. In developing this mixture there are two important points to be observed: That the wares produced shall be perfect in appearance and physical condition, and that they shall be composed of substances which yield brick of very refractory nature. To make a firebrick of satisfactory appearance is not a difficult problem. To be sure that the brick mixture is refractory, the manufacturer must compound it from only the purest clays, and especially those best suited to the conditions in the furnaces where the brick will be placed. The production of firebrick perfect in physical character is a matter of great moment to the firebrick manufacturer, and I shall explain how that valuable physical property, toughness, may be imparted to it, or how it may be sacrificed.

Raw clay usually carries so much hygroscopic and combined water that if it be made into brick and then baked in the kiln it will be taken out badly checked and cracked; besides this, we shall observe that it has suffered a very heavy shrinkage. There is a shrinkage in the kiln, however, which every firebrick mixture will stand without showing any tendency to crack or check. This problem is solved by careful study of the shrinkages of each clay entering into the composition of the brick mixture, and unless this list embraces several clays extremely dissimilar with respect to their principal constituents the question merely resolves itself into the possible combinations. There are two general varieties of clay from which firebrick are made, viz., hard or rock clays, and plastic or sedimentary clays. The purest plastic fireclays are found in the large deposits of the State of New Jersey. The shrinkages of the hard clays are not applicable to the plastic clays. In either case, however, there are varieties which carry a high percentage of combined water, which, if not driven off, is likely to lead to practical difficulties.

Some of the hard clays are immediately disintegrated when calcined, the action resembling that of some granites when exposed to fire. As a rule, they have to be reduced to a powder to permit the water of constitution to pass off freely. The soft, plastic clays, on the contrary, are able to evolve their water without disturbance of their molecular structure. The effect of such treatment, in either case, is to destroy the plasticity and harden the clay into a stone-like mass. If we compare calcined clay made from hard rock clay and that made from plastic soft clay the following differences will be observed: The fragments of rock clay are extremely hard and so dense that they are almost non-absorbent when placed in water, while the calcined plastic clays are rather hard but quite porous.

Similar methods are used for manufacturing firebrick from either of the two varieties. The raw and calcined clay (the latter being ground to a moderately fine condition) are mixed with enough water to convert them into a soft mass. The pugging is continued for a considerable length of time, until the mixture has been thoroughly effected and all the plasticity of the raw clay has been brought out. This stock is molded into bricks, which are dried to a stiff condition and afterward repressed, after which all of their hygroscopic water is driven off in a hot tunnel or dryer. The final treatment is to burn the brick in the kiln. The chemical constituents of both the plastic sedimentary and the hard rock clay are: (1) Refractory substances, viz., kaolinite (silicate of alumina), and quartz (anhydrous silica); and (2) fluxing impurities; lime, magnesia, ferric and ferrous oxides, and fixed alkalies.

The manufacturer depends for the solidifying of his ware upon the action of the plastic kaolinite, and the fluxing impurities. The green firebrick, which is practically free from fluxing impurities, and is dependent wholly upon its plastic kaolinite for producing a strong bond throughout its structure, possesses that property of toughness which the fluxing impurities cannot impart. The hard fireclays have little or no plasticity, and for them there is no choice; they must depend upon the bond given by fluxing impurities. The firebrick so united are not tough in the sense that they expand and contract successfully without disintegration. On the contrary, they seem to assume a character attributable only to the presence of fusible silicates, acquiring a hardness which is apt to appear brittle when subjected to sudden and extreme changes of temperature.

The brick made from the purest of the New Jersey plastic fireclays, which do not contain 1% of fluxing impurities, will burn smoothly, and yet very gradually fuse over the entire exposed surface, while the brick made from the hard rock clays will begin to chip off after the furnace is cooled down a few times and leave a jagged wall. This tendency of the brick to waste away rapidly without disintegration, as a result of inability to expand and contract, is more serious than a slight fusion of the brick on the surface. We find that "tough bond" exists most prominently in the firebrick made from the best plastic clays, in which it is due to the plastic kaolinite.

Referring again to the two principal constituents of a firebrick mixture, calcined clay "biscuit" and raw clay, there are two chief points in which they differ when made from the plastic and the rock clays respectively: (1) Of two clays containing the same percentage of kaolinite we find that the plastic clay is much more finely divided than the rock clay, and therefore is "fatter" (or more sticky): (2) Comparing the "biscuit" made from the two clays, the plastic (New Jersey) clay yields a slightly porous mass, while the rock "biscuit" is too dense to absorb water, having become almost vitrified.

In mixing and pugging these two substances in the shape of soft mud the tendency of the plastic clay "biscuit" is to absorb its raw fat clay to a very slight degree, by virtue of its little porosity, and thus is formed a strong adhesion between the two parts when in the green state, which imparts a firm union to the mass burned in the kiln. Kaolinite is the base of the clay and its plastic constituent, although the sticky nature depends wholly upon the fineness to which the material has been disintegrated. As the sedimentary or New Jersey fireclays have been thoroughly sorted by suspension in water, this property has been developed in them to the greatest possible extent.

Experience in the chemical laboratory teaches us that the rapidity with which a dry insoluble substance goes into solution in an acid depends in a great measure upon its fineness of division. In all probability the same law holds good in the case of raw clay; not that it can be made to go into solution in water, but when the clay is the finest water is best able to bring out its greatest kaolinite bond. In other words, the finest clays in a "soft mud" brick mixture, with the same amount of pugging, can be more readily absorbed by the porous biscuit than the coarser rock clays. A mason sometimes asks if a hard, vitrified brick "will take" cement, he having learned that cements cannot cling with any great strength to non-absorbent masses. It is just as true when applied to the relation between the raw clay and "biscuit" of a fire brick mixture; the former part must have its plastic agent to give and the latter should be sufficiently porous to receive it.

The term "high silica brick" is so often misused, especially in the trade, that the phrase arouses an incredulous feeling as soon as it is mentioned. Perhaps many people are cognizant of the fact that fire brick are being sold for "high silica brick" which, on the contrary, are as basic as any fire-clay brick can be made. Appended are four recent analyses of brick sold as "high silica."

Substance.	A.	B.	C.	D.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Combined Silica (SiO_2)	51 10	15·80	11·62	23·16
Alumina (Al_2O_3)	44·27	13·71	10·10	20·10
Quartz (SiO_2)	1·91	69·45	77·07	55·24
Lime (CaO)	0·30	0·22	0·11	0·21
Magnesia (MgO)	0·12	0·17	Trace.	0·13
Ferric oxide (Fe_2O_3)	0·22	0·43	0·95	1·06
Alkalies (Na_2O+K_2O)	2·72	0·53	0·47	0·37

There is nothing mysterious about the manufacture of "high silica brick." The term in itself implies not only a large excess of SiO_2 in the brick, but the more of this constituent that is introduced into the mixture, without sacrificing the necessary physical character of the brick, the more efficient will be its life; that is to say, if the use in question actually calls for fire brick containing a large preponderance of silica. Beyond a certain point, however, silica renders brick mixtures very short and unmanageable in the process of manufacture, and likewise makes the burned product more and more friable. A "high silica brick," therefore, cannot endure very much abrasion.

THE FIRE BRICK INDUSTRY IN GERMANY.

The manufacture of fire brick and refractory material in Germany was described by Dr. Verwer at the meeting of the Chemische Gesellschaft zu Aachen, November 9, 1898. The best fire clays are found in the Westerwalde, in the Eifel, in the Pfalz, in Silesia and in Prussian Saxony. Besides fire clay, kaolin, clay slate, quartzite, etc., are found in these districts. The refractory products are divided into acid or basic according to their tenor in alumina and silica. The type of the basic is chamotte brick, which is made of clay and chamotte (burned and ground clay); these contain normally 34 to 36% alumina. Higher tenors in alumina are attained by the admixture of Bohemian clay slate (Thonschiefer), kaolin and bauxite. The German bauxite is useless for this purpose on account of its large content of ferric oxide, wherefore French bauxite is employed. The type of the acid brick is Dinas brick, containing 98% silica. However, the German Dinas brick usually contains a considerable amount of clay and is really only "half acid." Bauxite bricks with 70% alumina, magnesite bricks and carbon bricks are made by certain works for special purposes.

THE MANUFACTURE AND USE OF PAVING BRICKS IN THE MIDDLE WEST.

BY H. FOSTER BAIN.

Character of Clays Used.—The clays used in the manufacture of paving brick are all of the sedimentary type. The purer clays which are derived from the decomposition of feldspathic rocks are not fitted for this work. It is the impure clays, those in which the kaolin is mixed with a relatively large proportion of fluxing impurities, which are in demand. The clays from different strata in the same pit are commonly not used separately, but are suitably mixed, as also are clays from different places. In general, three classes of clays have been used for this work: (1) Surface clays; (2) fire clays; (3) shales.

Surface Clays.—The brick laid at Bloomington, Ill., and noteworthy as being among the first used for pavers in this region, were made from surface clays by the simple soft-mud process. They were burned in up-draft kilns, by which process only a relatively small proportion of No. 1 pavers can be obtained. The brick first laid here would not be judged as much more than a hard-burned builder according to present standards, but they gave 20 years of excellent service. In Omaha a number of streets have been paved with similar brick made from black prairie soil scraped off the farm land at points between that city and Lincoln. At Decatur, Ill., similar brick have been burned. In general, however, the surface clays are not considered so well adapted to this purpose as the older beds, and practically paving brick are now made only of shales and fire clays, worked by the stiff-mud process. Shale is much the more common material.

Fire Clay.—Fire clay brick were among the earliest used in this country for paving. Usually large blocks, instead of bricks, were at first made. Charlestown, W. Va., has the credit of first using these blocks. Similar blocks, shipped down the river from this same region, were laid in Cincinnati some years ago. Fire clay brick are still being manufactured and used for pavers to some extent. It is obvious, however, that pure fire clay cannot be economically used for this purpose, as fire brick, even of a low grade, sell for more than pavers. It is the impure fire clays which are properly used and in most cases even they are mixed with shale. Since in the manufacture of pavers it is necessary that the brick be brought to the temperature of incipient vitrification and held there until a certain amount of annealing takes place, fire clays are poorly adapted to the work. Their peculiarity is their low percentage of fluxing constituents, so that they do not fuse to any appreciable extent at the low temperatures which motives of economy in fuel dictate for burning pavers. Fire brick as ordinarily manufactured have but little bond and are of relatively low strength. It is this as much as anything which makes the absence of laminations so important in them and necessitates the thorough pugging and expensive hand work employed in their manufacture. Paving brick, if they are to be widely used, must be manufactured at low cost, so that even if as good a brick could be made of the fire clay it would be unavailable.

When the fire clay is used in connection with shale, the fire clay acts much as does grout in fire brick. It forms little flecks of unfused material throughout the brick, outlining, if it be in sufficient quantity, the skeleton of the latter. In and between these bits of fire clay is packed the more fusible shale clay. The bits of imperfectly crushed shale of course to some extent act with the fire clay, but they are better bonded to the surrounding shale clay. This framework of unfused material may be a distinct gain or weakness to the brick. It depends to a large extent upon the relative shrinkage and fusibility of the two sorts of matter. If the fire clay and the unbroken bits of shale have less shrinkage than the matrix they may be conceived as resistant cores around which the other material shrinks, compensating to some extent the shrinkage of the latter and preventing the opening of minute cracks and the development of lines of weakness. If, however, these particles have a higher shrinkage they tend to pull away from the surrounding matrix. In any event too large a proportion of such inert matter intensifies that heterogeneity of structure which it is the object of the brick maker to avoid. In practical work the fire clays are mixed usually in small proportions, and are taken rather because they are in the pit and it is difficult to throw them aside than because of any good or bad qualities which they may inherently possess. They are normally somewhat impure and really grade by degrees into the surrounding shale, so that it is not always easy to say which is impure fire clay and which is shale.

Shale.—This is a somewhat indefinite term. As used by the geologist it mainly refers to structure. Shales stand between clays and slates. The cleavage planes which are the marked characteristic of the latter are absent in a clay and imperfectly developed in a shale. These planes are due to pressure and accordingly usually indicate a period of burial below a considerable mass. It follows that they are normally more abundant in the older clays. Their presence, may, however, be due to other causes, and may indicate disturbance rather than age. In ordinary instances they are approximately horizontal and are parallel to the original planes of sedimentation. They are not associated with any particular kind of rock or clay so far as chemical composition is concerned, nor are they necessarily associated with any other physical properties. They are best developed in the clays which have suffered the most compression, as is shown by their perfection in the slates overlying coal beds. In a sense they indicate a dense clay; but this is less important, since in pugging a clay its original structure is destroyed and it is left for the brick machine to give it new form. Shale structure is apt to be the more perfectly developed the finer the grain of the clay. This is probably the essential point of the well-known relation between shales and good brick material, since it is known that, within the limits of practice, the finer the material is ground the better the brick. It is sometimes insisted that only shales are capable of yielding good pavers; but this seems to be carrying the matter too far, as it is quite possible to have an exactly similar clay, both in composition and texture, without the development of the cleavage planes which mark it out for a shale.

The boundary line between shales and slates is equally difficult to draw. Van Hise has shown that the same process which changes the soft plastic clay to the moderately hard cleavable shale, if continued, will transform the latter into the very hard thin-splitting slate or even into a crystalline schist. The development of cleavage planes is brought about or accompanied by an actual movement of the particles of the mass so that their least axes lie normal to the pressure. Deformations of the particles in the same direction also occur and eventually chemical change creeps in and new minerals are developed. As a result of these changes slates and schists are unfitted for use in the clay-working industries. They become hard to crush and often fail to show plasticity even after very fine grinding. A very pronounced cleavage is accordingly a bad indication, and in any case the shaly character is probably more of an accidental variation usually marking a good clay than anything inherently of value.

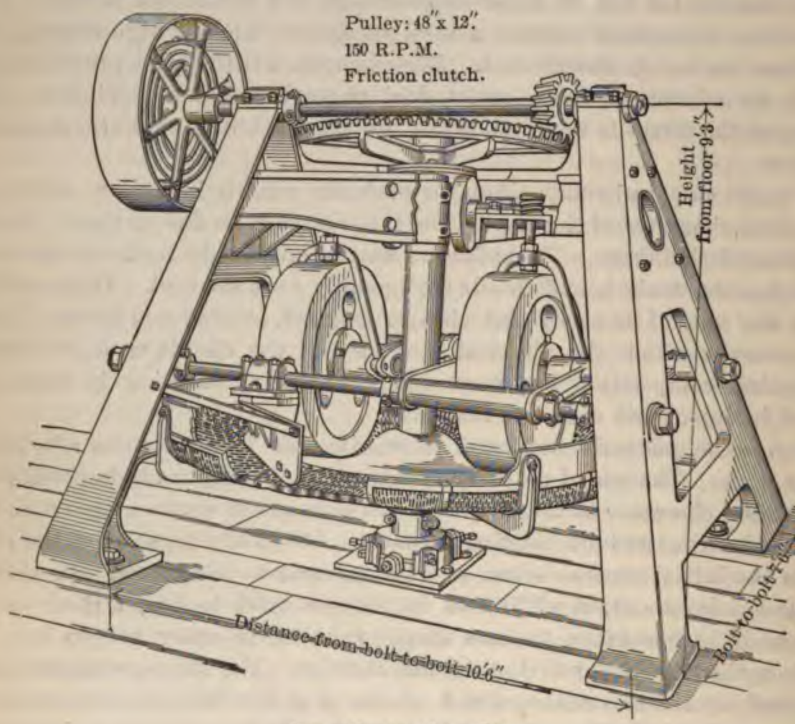
For manufacture into paving brick a clay or shale should have a relatively high percentage of fluxing impurities, the alkalies, lime and iron. Surface clays are usually too low in these constituents, and are higher in silica than is desirable. Often they contain a total of fluxes which is high enough, but these fluxes are poorly distributed. For example, a little lime properly combined is an advantage, but a great deal uncombined is a detriment. The character of the fluxes is very important, as some act with and others against each other.

Next to the chemical composition, or probably equally important with it, is the physical character of the clay. The latter should be fine grained, plastic, and of limited shrinkage. These factors control largely the methods by which the clay may be worked, and this in turn mainly fixes the cost. It is essential that the clay should be easily and cheaply pugged, molded and dried. These are processes in which the physical character of the clay is most important. The chemical composition has more influence during burning in fixing the points of incipient and complete fusion.

Mining.—The methods employed in mining the clay are those adapted to open pit work. The use of plows and scrapers is usually made impossible by the indurated character of the material. In the smaller works simple pick and shovel work expedited by blasting is usual. When the capacity of the plant warrants the initial expense steam shovels are used to advantage. In general it requires a daily output of 70,000 to 100,000 brick to keep a steam shovel busy, and it is important to have steady running in order not to lose the interest on the investment during the idle months. The clays are usually readily handled by the steam shovel and where, as at Des Moines, the clay shows marked variation in horizontal distribution the steam shovel is an important aid in getting a thorough mixture.

Transportation.—The clay as dug is loaded into small cars of the size and type common in coal mines and hauled to the crusher. This is usually done by tail rope since in most instances it is necessary to hoist the clay so that it may feed down easily to the crusher. Where there is no such necessity mule power is often employed. The pit and the factory are ordinarily close together as it rarely pays to haul crude clay any considerable distance.

Crushing.—The clay as it comes from the pit is dumped into bins, from which it is fed either automatically or by hand into the crushers. In a large majority of the recent plants the latter are what is known as dry pans. These consist of a circular revolving pan usually 9 ft. in diameter with a chilled iron bottom. In this pan are two heavy wide-surfaced wheels called mullers. The mullers are 48 in. in diameter and have a face of 10 or 12 in. They are made of cast iron with a removable 4-in. tire of chilled iron, and weigh about 400 lb. per inch of surface. As is shown in the accompanying engraving, these mullers rest on the solid portion of the base of the pan and the clay is carried under them by the revolution of the latter. The centrifugal force throws the clay out so that it drops through the gratings and is collected for screening. The gratings are made in reinforced cast plates. The openings are slots set radially, and are usually a little larger than the screens so that a portion of the clay comes back to be reground. In grinding shale, for example, the dry pan slots may be $\frac{1}{8}\times 2$ in. and the screens $\frac{3}{32}\times\frac{3}{4}$ in. In the better-built machines a portion of the weight of the mullers is supported on the cross beam by the Lambing or some other anti-friction device, so as to prevent heating of the step below the machine. A spring takes up the jar when the mullers pass over a bit of rock or other obstruction and suitable shields direct the clay under the mullers. The capacity and power needed vary considerably with the character of the clay. With a moderately hard shale the pan makes 25 to 30 revolutions per minute if the clay is damp, and less if dry. Under such conditions 20 to 30 h.p. would be needed, and clay for 35,000 to 60,000 brick could be prepared. Where, however, the clay is not suited for this work, as is the case



THE EAGLE DRY PAN.

The pan is 9 ft. in diameter. Each roller weighs 4,500 lb. The total weight of the machine is 30,000 lb.

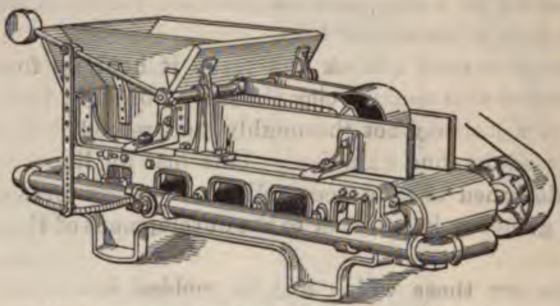
with loess clays, which are always slightly damp unless artificially dried, the same machine may only prepare clay for 20,000 to 30,000 brick. In some such cases the clay seems to merely mash down and not to disintegrate. In shale crushing where the pan is given constant service new screens are needed each season, and new tires on the mullers must be furnished, at a cost of about $125, every four or five years. The whole machine well mounted on an iron frame costs $900 to $1,000, and about one-third less with a shorter-lived wooden frame.

The dry pan is excellently adapted for crushing such material as is ordinarily used in making pavers, since it has the merit of handling a considerable variety of clay and doing it rapidly. While in many cases the clay should be crushed finer, that is usually a matter of the way the pan is used. As the clay comes from the dry pan it runs through screens, the tailings going back to be re-crushed. The screens are revolving trommels or shaking screens. They may be either slotted or punched sheets or woven wire. In general they pass material from $\frac{1}{16}$ to $\frac{1}{4}$ in. in diameter. It is probable that finer grinding and closer screening would in many cases improve the quality of the brick. The screened material goes to the supply bins.

Pugging.—In order to mold a brick and make it hold its form while being handled, it is essential that the dry clay should be brought to a plastic condition. Plasticity is a property not thoroughly understood, but in general the plasticity is increased by fine grinding. This has its limits and in practice the grinding is designed also to accomplish a thorough mixing of the clays. None the less the grinding is essential to the development of the latent plasticity of the shale.

Plastic materials are those which may be molded into shapes which they will hold. When a mass holds a certain shape it does so because of the cohesion or adhesion between its component particles. This depends, other things being equal, upon the number of points at which the individual particles touch each other. In coarse-grained material the air spaces are large and unequally divided, and the particles touch at relatively few points. In fine-grained material the air space is much more evenly broken up and the points of contact between the individual particles are far more numerous. If these particles be placed in a certain relationship to each other, there being more points of contact, the mass holds its shape better.

The action of water in adding to the plasticity of clays is due to the force known as surface tension; which, in liquids, causes them to expose at all times the least amount of surface possible. The water spreads over the surface of the individual particles, and striving to reduce its own surface, pulls the particles together. If we imagine two particles surrounded by a very small amount of water, it will be evident that the latter acts much in the same manner that a rubber band would in drawing them together and holding them there. If, however, the quantity of water be conceived to be larger, it will be seen that the particles will swim freely in it. The importance of getting the right mixture of water and clay is accordingly evident. This is a matter often left entirely to the judgment of the man in charge and is something difficult to control. In some of the newer works there is an automatic machine, illustrated herewith, which regulates the amount of water flowing into the pugging machine according to the amount of clay. The latter falls into the hopper at the top and is carried forward to the pugger by an endless belt. On this belt runs a loose wheel which is geared to the valve in the water pipe. If a large amount of clay runs out the wheel is necessarily lifted while the clay runs under it. The conditions are such that this lifting of the wheel opens the valve and allows a corresponding outflow of water. When the belt is not carrying clay the wheel runs directly on the belt surface and the water valve is closed. It is only necessary to determine the correct percentage of water and clay and set the machine accordingly. This machine has not yet been widely introduced, but as it does away with the services of one man and improves the quality of the brick, it will certainly be rapidly adopted. It sells for $150. The automatic mixer makes it possible to use hot water in mixing the clay. This is a distinct advantage, as the brick dry more quickly and the capacity of the dryer is correspondingly increased.



COOK AUTOMATIC CLAY TEMPERING MACHINE.

The total weight of the machine is 375 lb. The distance between the foundation bolt holes, center to center, is 24 in. lengthwise and 18 in. endwise. The driving pulley is 8 in. diameter, 4½ in. face, and makes 80 r. p. m.

The water and clay are thoroughly mixed in what is known as a pug mill. This is a narrow horizontal box in which is a horizontal revolving shaft set with short blades arranged to cut through the clay or to knead it and at the same time feed it forward. The thoroughness of the pugging is controlled by the length of the box and the speed at which the clay is forced through it. In ordinary cases the pug mill is 8 to 10 ft. long, but in some of the newer plants 18-ft. mills have been installed. It is a serious though common mistake to make the pug box too short. The quality of the brick depends to a very considerable degree upon the perfection of its work. It should yield a thoroughly homogenous plastic clay.

Molding.—As paving brick are usually made, the brick machine and the pug mill are combined. The clay is fed forward from the pug mill and by means of an augur is forced out in a continuous bar through a die which gives the form to the brick. This bar is cut off in suitable lengths or widths according as end-cut or side-cut bricks are being made. In other cases the clay is conveyed from the pug mill to a second box from which the augur carries it through the die.

Augur machines have the merit of large capacity—25,000 to 35,000 brick per day—but have the disadvantage of introducing laminations in the brick. As the clay bar moves forward through the die the friction on the sides retards the outer position of the clay, and if the pressure is equal over the whole cross-section the center of the bar moves fastest. This makes the planes of lamination planes of sheering, and breaks the brick up into a core surrounded by a more or less distinct series of layers of clay. It is clear that such a laminated structure is particularly bad in a paving brick. The latter should act as a unit. If the laminated structure be developed there are not only open cracks subjecting the brick to destruction by frost, but planes of weakness which greatly lower the strength of the brick. Laminated brick always break much lower when tested than brick which show no laminations. To avoid the latter with an augur machine requires the careful adjustment of the machine to the individual clay and constant watchfulness on the part of the brick-maker. Clay mixed too soft or too stiff will laminate. Improper feeding may also produce this result. In general, however, the best means of preventing the lamination is to have the augur as far back from the die as working conditions will allow.

Repressing.—In recent practice the green brick as they come from the machine are fed into a repress. This is a steam press with a vertical plunger forcing a blank into a die and shaping up the brick. The merits of repressing have been considerably discussed. It is claimed on the one hand that it gives a denser, firmer structure and overcomes the laminations by firmly pressing the clay into form. On the other hand it is maintained that the pressure is not sufficient to give the clay a new structure, but merely enough to disturb the old one and that the chief value of the repressing is the better form which it gives the brick. It is certainly true that the pressure is very slight. One may finger mark the green brick before they go to the press and find the marks distinct after the repressing. It may well be doubted whether such slight pressure adds materially to the density of the brick, though it adds about $1 to the price. This is a source of profit as the process costs only about 25 to 35c. per 1,000. In some cases repressing results in actual injury to the brick. In this process it is necessary that the brick should be oiled and this outer coating of oil occasionally prevents thorough drying.

Prof. Edward Orton, Jr., has shown by an admirable series of experiments* that end-cut brick make, in the case of the clay tested, better pavers for being repressed; and that, with the same clay at least, the side-cut brick were the poorer for being repressed. None the less there has been a distinct reaction in favor of side-cut brick and in Des Moines these are now the favorites. At Oskaloosa, Iowa, end-cut brick are made without repressing and an excellent result is obtained. One of the Des Moines plants is now marketing a very good side-cut, unrepressed brick.

Drying.—The brick as they come from the repress are stacked on iron

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* *Proceedings Eleventh Annual Convention National Brick Makers Association*, 1897, p. 266.

frame cars and rolled into the dryer. The latter is ordinarily a single large, low room or a series of parallel tunnels through which dry hot air, either taken from the cooling kilns or heated by live or exhaust steam, circulates. As it becomes charged with moisture drawn from the brick it is drawn off through a stack. The temperature of this room or series of tunnels is about 150° to 175° F. at the cooler end and 200° to 225° F. at the warmer. The car of brick is rolled in at the cooler end and gradually moved forward as other cars are shoved in back of it, so that the brick passes through the dryer in from 36 to 48 hours. The brick as they go into the dryer are heavily charged with moisture. When they come out they should be thoroughly dry. In losing their moisture they become firm and hard and shrink considerably. This shrinkage is ordinarily about 5%. The philosophy of the shrinkage seems to be that the decrease in the amount of water increases the surface tension of that left behind and draws the clay particles closer together. If the water were simply driven off there would be a decrease in weight but not in bulk, whereas there is a decrease both in volume and weight. If the clay has not been homogeneously mixed the shrinkage is unequal and the brick become warped. This close packing of the clay gives it greater strength, and Wheeler has shown that its tensile strength becomes occasionally higher than that of some cements.* In the case of paving brick clays he found it to be 100 to 200 lb. per sq. in.

Burning.—From the dryer the cars of brick are run on temporary tracks into the kilns, where they are stacked for burning. Proper burning is the most essential part of the manufacture of pavers, and much time and attention have been given to the development of suitable kilns. Nearly all work is done now in down-draft kilns of some type. The up-drafts do not produce a large enough percentage of No. 1 brick to make their use economical, though they themselves are cheaper; and it is the kilns which constitute the most expensive part of a modern brick plant. Continuous kilns have not so far proved successful in the burning of pavers. Where a plant works continuously and where, as in the more recent plants, the heat from the cooling kilns is drawn off by a fan and turned into the dry house or kilns just set, the difference in the cost of working with a continuous and a series of intermittent kilns is after all not very great.

There are several different types of kilns in use. In appearance they divide themselves into the round and the rectangular. The round kilns are relatively small, holding 30,000 to 40,000 brick. They are fired by furnaces arranged in the sides and opening into the kilns near the top. The heat is drawn down through the brick, through a loose bottom and out, usually through a center stack doing duty for several kilns arranged around it. More rarely each kiln has its own stack or stacks built in its walls. These kilns are built of ordinary brick with a dome roof and are bound with iron bands. The furnaces are necessarily built of fire brick and the more exposed parts of the lining may be of the same material.

The rectangular kilns are much larger, of 150,000 to 250,000 capacity, and are built with either solid or open bottoms. In the latter case they are of

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* *Missouri Geological Survey*, Vol. XI., p. 85.

essentially the same construction as the round kilns, though each kiln usually has its own series of stacks. The solid bottom kilns are a recent improvement and have the advantage that wagons may be driven directly into the kilns to load. The heat in these kilns is drawn off by side flues with openings near the bottom of the kilns. The direction of the heat currents is regulated by dampers and also by the amount of space left between the brick. Thus if this space is wider near the center of the kiln, the stronger draft is found at that point.

In burning pavers the color is not important, so it is unnecessary to water-smoke with wood. Coal is used from the first and the cheapest slack coal is well adapted to the work. The long flame of the gas coals is an advantage. In the Swift, Dewhurst and other kilns of similar type the burning is done by gas evolved in coking the slack coal and then the coke is raked down into the lower fire box and itself burned. By this process it is possible to burn economically a fine slack coal which otherwise would drop through the grates. At Des Moines these kilns are used, burning about 1,000 pavers per ton of steam coal. As the latter, a mixture of slack and pea coal, is the cheapest coal on the market, this is very good work. In general the slack-burning furnaces save from 50 to 75c. per thousand in the cost of burning. The ordinary cost of burning pavers is in the neighborhood of $1·75 per 1,000.

The brick as they come from the machine carry about 20% of water, measured by weight. As they pass through the dryer they lose about 9·5 to 10% in weight, or a little over 5% in volume. When they are put in the kiln they still carry from 8 to 10% weight of water, so that firing must be very gentle at first until that portion which is mechanically mixed with the clay is driven off. This is the process known as water-smoking and lasts from one to two days. When the brick have ceased to steam the furnace is closed down and heavy firing begins. It is obvious that driving the water out of the brick must from the first be carefully done. If too great a heat be applied at first either in the dryer or the kiln, the outer portion of the brick dries so rapidly as to protect the moist central portion and in the later high heats of the kiln, the brick burst. When the brick are thoroughly dry the heat is rapidly raised to the point of incipient vitrification which for the Des Moines clays averages about 1900° to 2200° F., and in general ranges from 1600° to 2300° F. The kiln should be held at this heat for some hours and then cooled slowly. Burning, including water-smoking, ordinarily takes from six to ten days, and cooling from six to twelve. In the burning there is a shrinkage in volume of nearly 9%, and in weight of about the same. Beyond the point when a brick is incipiently vitrified there is a continued though slight shrinkage in volume, amounting to as much as 1·5%, but, as has been pointed out by Mead, there is no corresponding decrease in weight.* This additional shrinkage indicates the physical changes resulting from the complete vitrification of the material. Mr. Mead's figures are of considerable interest, and for greater convenience in comparison are given below, calculated in each case as percentages of the original volume or weight, the percentage of loss by volume being given first, and that by weight immediately

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***Transactions of American Society Civil Engineers*, 1893, Vol. XXIX., p. 664.

following in parentheses: In drying, 5·03 (9·64); in burning, 8·84 (8·97); in vitrification, 1·41 (*nil.*).

If an ordinary building brick be broken the individual grains of sand or clay can be easily rubbed or broken off. The bond which holds them together is very slight and the brick may be easily crushed. If a well-burned paving brick be broken the individual grains, except the uncrushed particles, cannot be so distinguished. The whole has a dull stony luster. If a brick slightly overburned be examined it will be found to have a semi-vitreous luster which reflects the light from the fracture planes. If a thoroughly vitrified, overburned brick be examined it will show a shrivelled, fused, unusually dark and scoriaceous fracture plane. The best brick are burned merely to the second stage and it is as important to avoid over as under burning, since the overburned brick are not tough. They break easily, though so far as abrasion, distinct from impact is concerned they would usually rank higher. With many clays there is but little range of temperature between the point of incipient and complete vitrification, and since it is impossible to regulate the heat of any kiln very narrowly, a considerable number of clays are shut out from any large use as pavers by this fact. The difference in temperature between the two points should be 300° to 400° F.

The burning of the brick usually depends entirely on the judgment of the foreman. Trial pieces are occasionally set where they can be taken out and tested. In many cases, however, the time consumed and the burner's guess are the main elements in determining when to open a kiln. At some of the plants the burning is measured by the shrinkage, and when the top courses have dropped a certain number of inches the kiln is cooled. This is a simple and effective test. Seger cones are not much used at these Western plants except in experimental work.

When the brick have been sufficiently burned the kiln is cooled, great care being exercised not to allow cold air to reach the brick too soon. In the earlier work the importance of slow cooling was underestimated in the desire to get full use of the kilns; but the days when brick are loaded out so hot as to set the railways cars on fire have passed and the larger plants now use every effort to improve the quality of their brick by slow annealing.

Testing Paving Brick.—The extended use of paving brick is so new that the technology of the industry is still in process of development. Necessarily the methods of testing are also tentative. The brick first made and used would fall far below present standards; and in reading a series of specifications one is struck with many curious inconsistencies. For example one engineer carefully specifies that the brick "Shall be hauled to the point where they are to be used and there dumped to the ground," while a neighbor specifies that the brick "shall not be dumped." In everyday work a good inspector, particularly after some experience with the brick of a given plant, can form an excellent judgment of their quality by their appearance and behavior under a chipping hammer. In order, however, to have something capable of formulation in writing engineers have applied certain other tests.

In the main the crushing and transverse strengths, rate of absorption, and resistance to impact and abrasion have been the things sought to be measured. The first two tests may be made in the ordinary testing machine according to the methods used in testing building brick and stone. There has been little discussion as to the methods to be adopted in making these tests, but considerable as to their value. The rate of absorption varies greatly with the dryness of the brick at the beginning of the test and with the continuance of the latter. These facts have not always been kept in mind. The impact and abrasion test is made by loading the brick into a foundry rattler either alone, with blocks of stone or iron, or with certain amounts of foundry shot. The results vary with the length and diameter of the rattler, the speed at which it is driven, the length of the test, the percentage of empty space in the rattler and the character of the charge. With all these variations it is evident that the methods adopted must be uniform or no two series of tests are comparable.

In view of considerable doubt as to the value of many of these tests and the lack of uniformity in their application, the National Brick Manufacturers' Association in 1895 appointed a committee to investigate the problem and to recommend a series of standard tests. This report, which was published in 1897, is an exceedingly valuable one. A very elaborate series of tests were made and the results carefully sifted and summarized. The commission recommended the abandonment of all tests save that of the rattler as unnecessary and in some instances misleading. It gave careful specifications for the rattler test and for the test of transverse strength.

Uses of Paving Brick.—While pavers are mainly used (as their name implies) for street paving, there are important secondary uses. Gutters, sidewalks and buildings may all be made of second-class pavers. Chemical tanks and sewers require the better grade. The use of pavers for both sewers and streets is an excellent combination and makes it possible to clean the streets by the hydraulic process. Brick not classed as either No. 1 or No. 2 pavers command ready sale as "builders."

In its use as a paving material brick has some of the advantages of each of the other paving materials and in a wide range of situations it has combined advantages over all other materials. Brick pavements are laid in two styles—one course and two course. The latter plan is adopted where a good foundation would otherwise be difficult or too expensive. When two courses are used the lower (which may be of No. 2 grade) is laid flat, and a sand cushion separates the top course from it. The foundation is rolled cinders or gravel, concrete, or old telford or macadam.

Illinois is the leading Western State in the manufacture of pavers. The most important works are located at Galesburg, where there are several plants, the leading ones being the Purington and the Galesburg Brick and Terra Cotta Co. The Galesburg manufacturers have had the advantage of a more uniform clay than many of their competitors are forced to use, and the further advantage of an early start in the business. The Springfield Paving Brick Co., of Springfield, is also one of the larger concerns of the West. There are a couple of plants at Streator, the larger one being the Barr Clay Works. The

Alton Paving Brick Co., and the Ottawa Fire Clay & Brick Co., have furnished considerable brick, mainly for the local markets. A considerable amount of paving brick have been made by the Spruck Brick Co. near Peoria, and at Sears, Bradolph, Jacksonville, Champaign and Urbana some have been burned. At Cuba and London Mills some of the earliest brick used in this region were burned, and at Virden, John Utt & Sons have manufactured. At this point a 25 ft. bed of clay is mined at the depth of 225 ft. This is quite unusual, as pit clays are ordinarily used in this work. At Decatur, Bloomington and Danville surface clays are used.

Iowa ranks second among the Western States as a paving brick producer. The main industry is centered at Des Moines, where there are now five plants engaged in the business. The pioneer was the Des Moines Brick Manufacturing Co., which, beginning in 1891, enlarged its plant from time to time until in the season of 1898 it operated a plant of 30,000,000 annual capacity. In December, 1898, the plant was burned, but is to be rebuilt. The Iowa Brick Co. and the Flint Brick Co. have large and well-equipped plants built in 1893. The Capital City Brick & Pipe Co. and the Merrill Brick Co. have smaller plants. At Burlington a large amount of excellent brick have been made by the Granite Brick Co. At Oskaloosa there is a thriving plant, the Oskaloosa Paving Brick Co., mainly supported by the local trade. At Ottumwa there is a similar concern likewise named after its home town. At Buffalo, near Davenport, the Davenport Paving Brick Co. has manufactured several million pavers. At Fort Dodge the Fort Dodge Clay Works and the Fort Dodge Brick & Tile Co. have gone into business. In former years some brick made at Lehigh, near this place, also came into the market. More recently the Lehigh firm has been mainly concerned with building and face brick. At Council Bluffs local material is used in two-course work. At Boone the Boone Paving Co. and Boone Clay Works have made a small number of pavers, and at Sioux City the Sargents Bluff & Sioux City and the Northwestern Sewer Pipe Works have each furnished pavers for local use. Neither is now in the business though the plant formerly operated by the latter firm has passed into the hands of the Lower Brick Co. and after being thoroughly overhauled is to be operated as a paving brick works. In general the tendency of recent years has been for the Des Moines and Galesburg plants to supply the bulk of the shipping trade. The smaller plants can manufacture brick suitable for local use, but at present do not seem to make much headway in the general market.

Missouri.—Prof. H. A. Wheeler describes some 14 plants in different parts of this State. The output in each case is small and the industry may be said to be poorly developed since work began. This is the more remarkable in view of the excellent local markets, good shipping facilities and pre-eminence of Missouri in other lines of clay goods. At St. Louis, where so many millions of building and fire brick are annually made, a recent visit failed to locate a single large plant devoted to pavers.

Kansas.—The main industry in this State is centered at Atchison and Topeka, though there are small plants at other points. At Atchison the first paving was done with Galesburg brick, but for many years all the work has been done with local material, and a good shipping trade is being built up. At Leavenworth the Atchison and St. Joseph paving brick were used as early as 1894, but it is expected to begin work with local material.

Nebraska.—The paving at Beatrice and a considerable amount at Omaha has been done with brick of local manufacture. At Nebraska City a plant has shipped pavers, some of the product going east as far as Creston.

Future Development.—The development of the industry has now gone far enough to allow some inferences as the future to be fairly drawn. It has been shown that different cities and different streets require widely differing qualities of brick. Material which could not be used on streets of heavy traffic may be excellently adapted for use at other points. When traffic is not too heavy and where concrete is expensive two-course work may be used to advantage. There are very few brick yards which cannot, with proper care, produce brick suitable for the lower course, and a considerable number can furnish brick suitable for top-course work either over brick or concrete. The number, however, which can furnish such brick suitable for use on streets of heavy traffic is limited, and often local and imported brick may be with advantage used together. At Council Bluffs, for example, some recent work has been done with local brick for the lower and Des Moines brick for the top course. In general, with the cheapening price of cement, the one-course pavement is found to be more and more used. This will limit the field so far as local manufactures are concerned, as it cuts off their market for seconds. The manufacturers of pavers located in or near good cities have no trouble in this particular, as their seconds bring top prices as building brick. The large manufacturers have also all the advantage which comes from ample capital, specialization, and thorough knowledge of the business. Under such circumstances large centers of manufacture are sure to develop, and some of them are even now foreshadowed. Aside from the considerations mentioned, suitable clay, local support, cheap coal and good shipping facilities are essential to success. The amount of coal necessary and the bulkiness of the product make the last two especially important. The experience of the past year, however, and the outlook for the present seems to indicate that too much specialization may be disastrous. The manufacture of building and paving brick should be carried on together and both trades should be cultivated. The brick which cannot be used as pavers make excellent builders, either "rock-faced" or plain, but it is well to remember that this is one of the rules which will not work both ways.

KAOLIN: ITS OCCURRENCE, TECHNOLOGY AND TRADE.

BY T. C. HOPKINS.

Mineralogical.—The word kaolin is said to be a corruption of the Chinese word "kauling," meaning "the high ridge," the name of a hill in China where the material is mined. Kaolin, as now used in the trades, refers to the commercial forms of the mineral kaolinite. This mineral, like most others, does not occur in nature in a pure state in commercial quantities. The pure kaolinite is chemically a hydrous silicate of alumina with a formula Al_2O_3, $2SiO_2$, $2H_2O$, which equals 46·5% SiO_2, 39·5% Al_2O_3, and 14% H_2O. This is the essential base of nearly all commercial kaolins, as in fact it is of all clays. The common impurities which occur with the kaolinite are silica, iron, alkalies, lime, magnesia and titanium. If the iron is absent and the alkalies are in small quantities and the product white or nearly so, it will pass in the markets for kaolin or china clay. As it varies from this composition or from the white color it passes into one of the other numerous varieties of clay. The most common mineral impurities are quartz, feldspar, mica, calcite, magnetite, limonite, pyrite, apatite and hornblende.

Some writers use the term kaolin for the crude material as found in nature and china clay for the refined product obtained from it. Thus the material goes into the washer as kaolin and comes from it as china clay. This usage while common is not universal and is liable to lead to confusion.

There are several other minerals closely related to kaolin in composition and physical character, so closely related in fact that they might, for the most part at least, be classed as varieties rather than as separate minerals. Some of these varieties have considerable local interest, some are of commercial importance, but most of them are of interest only to the mineralogist and are unknown to the tradesman. The chemical relations of these minerals to each other and to kaolin are best shown by tabulating them as follows:

CHEMICAL COMPOSITION OF KAOLINITE AND RELATED MINERALS.

	Formula.	Al_2O_3	SiO_2.	H_2O.		Formula.	Al_2O_3	SiO_2.	H_2O.
		%	%	%			%	%	%
Kaolinite.....	Al_2O_3, $2SiO_2$, $2H_2O$	39·5	46·5	14·0	Montmorillonite..........	about Al_2O_3, $2SiO_2$, H_2O, but varies.	......	......	......
Halloysite....	Al_2O_3,$2SiO_2$,$2H_2O$aq	36·9	42·5	19·6					
Newtonite....	Al_2O_3, $2SiO_2$, $5H_2O$	32·7	38·5	28·8	Allophane....	Al_2O_3, SiO_2, $5H_2O$	40·5	23·8	35·7
Rectorite.....	Al_2O_3, $2SiO_2$, H_2O	42·5	50·0	7·5	Collyrite	$2Al_2O_3$, SiO_2, $9H_2O$	47·8	14·2	38·0
Cimolite......	$2Al_2O_3$, $9SiO_2$, $6H_2O$	23·9	63·4	12·7	Schroetterite.	$8Al_2O_3$, $3SiO_2$, $30H_2O$	53·1	11.7	35·2
Pyrophyllite..	Al_2O_3, $4SiO_2$, H_2O	28·3	66·7	5·0	Indianaite....	Al_2O_3, $2SiO_2$, $2H_2O$	39·5	46·50	14·00

Any one of the above is likely to occur in small quantities associated with kaolinite or allied minerals, but the only ones known to occur in commercial quantities are indianaite (see under Indiana), halloysite and newtonite. Halloysite is mined in Kentucky, South Carolina and Georgia.* There are numerous other sub-varieties, such as smectite, bole, lenzinite, gumberite, samoite, etc., which are mostly local in their occurrence and importance.

Kaolin as a rule has very little plasticity, sometimes none. Sometimes the

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* *Engineering and Mining Journal*, Oct. 1, 1898.

ball or plastic clays are classed as kaolin, which is a confusion of terms. Kaolin is frequently rather pulverulent, and sometimes contains minute crystals and clusters of crystal plates, hexagonal in outline and resembling mica plates in thinness. The crystal forms for kaolinite which have been measured approximate those of biotite (Dana). The lack of plasticity in kaolin is commonly ascribed to the presence of these crystal plates, and it is claimed that if the material can be ground fine enough to break up the crystals it will become plastic. However this may be theoretically, the crystals are so small that practically it is not possible to render the commercial kaolin highly plastic by grinding or any other known process.

The most common non-essential constituent in kaolin is free silica or quartz, present sometimes in angular fragments of considerable size, sometimes as sand grains and sometimes in extremely fine particles. Fortunately this most common impurity is one of the least injurious; in fact, if the percentage is constant it may be no injury for pottery use, as it is one of the substances added to the mix in making pottery. In the residual kaolins mica and feldspar are frequently so abundant that it is necessary in nearly all cases to separate them by some process before the kaolin can be used. In some places nature has performed this process and the washed and refined kaolins have been deposited in sedimentary beds.

Source.—Kaolinite is a secondary or derivative mineral, its source being aluminum bearing minerals, the most common of which are the feldspars, which are probably the origin of nearly all the commercial deposits of kaolin. But clayey soils and more impure clays containing kaolin may be derived from any one of the numerous alumina and silica bearing minerals.

The feldspars consist of silicate of alumina combined with the silicates of potash, soda or lime, or a mixture of two or all of these bases. The meteoric waters containing carbonic and other acids attack this compound and slowly dissolve the alkalies at probably the same time that some water of crystallization combines with the silicate of alumina. In this process there is some silica in excess which may in part remain as finely granular quartz or under favorable conditions may enter into new combinations forming other silicates.

Mode of Occurrence.—The kaolin which occurs in the position of the original feldspar is called residual kaolin. Frequently it happens that this residual material is carried away by the streams and deposited as sediment in a distant locality. It might then be called transported or sedimentary kaolin. The residual kaolin is likely to have considerable quantities of fragmental crystalline quartz, mica and undecomposed spar, with smaller quantities of other minerals, while the transported kaolin is likely to contain iron oxide, lime carbonate and other impurities intimately diffused through the mass. Hence of the two classes the first is likely to furnish the purest grade of kaolin, as the impurities are nearly all in coarse particles which can be separated by washing. Although in natural condition the second is purer kaolin such impurities as do occur are likely to be of a nature that will not wash out.

The residual kaolin deposit will take the form of the feldspar body from which it is derived, which is generally that of a vein or dike. In most cases the rocks in which the kaolin deposits occur are disintegrated to a great depth; and it is difficult, sometimes impossible, from the small opening made, to determine either the kind of rock in which the kaolin occurs or the size of the deposit. The common method of removing the material adds to this difficulty.

The transported kaolins occur in sedimentary beds like other clays which have been deposited as sediment from water. These vary greatly in thickness and extent, but are generally of greater lateral extent than the residual kaolins.

Mining.—The common method of mining kaolin in the United States is by means of vertical shafts 25 or 30 ft. in diameter lined with small pieces of lumber about 3 in. thick, 10 or 12 in. wide, and 2 ft. long. The ends are cut on a bevel so that when the pieces are laid end to end they make a many-sided polygon roughly approximating a circle. This makes an exceedingly strong wall that will resist the great pressure from the clay and at the same time can be extended to an indefinite depth and then readily removed when desirable. The walls are extended downward by building on below, and when the shaft is sunk as deep as desirable the timbers are removed from the bottom upward and the hole filled with waste as the timbers are removed.

In some places the clay has been mined from open pits, and in a few instances in underground galleries by using heavy timbers, but in most cases where the kaolin does not have a solid rock roof, or an excessively heavy loose covering, the first method described will be found to be the most economical and safest, and it is the one most commonly used.

Washing.—In nearly all cases it is necessary to wash the kaolin before using it, the object being to remove the coarse material and the foreign impurities as far as possible. While this process is carried on by different methods, they are all based on the same principle, that of flotation, in which all the material is thrown into water, and the clay particles being finer and lighter than the foreign admixtures, remain longer in suspension. Hence it is only necessary to increase the length of the troughs through which it is carried or decrease the rate of flow, or both, to get the required degree of fineness in the kaolin, and remove practically all the foreign ingredients. The law that the carrying power of water increases with the sixth power of the velocity is true conversely.

One method used at many of the Pennsylvania and Delaware kaolin washing plants is to feed the crude material with a current of water into a common log washer, consisting of a horizontal beam anywhere from 10 to 25 ft. or more in length, revolving in a horizontal rectangular or semi-cylindrical trough of about twice the diameter of the beam. The beam contains numerous short arms or knives which serve to break up the lumps, stir up the material, and at the same time move it slowly to the opposite end of the trough from that in which it is thrown. The current of water carrying the clay passes from the log washer into a long trough or series of troughs arranged for convenience so as to lead the water through a labyrinthine course. The length traversed by the current in these troughs and the rate of flow may be varied to suit the character of the material used and the grade of kaolin required. The greater part of the coarse sand and the larger particles are dropped either in the log washer or close to it and sand wheels are used to remove this and prevent the troughs from being clogged. The finer sand and the mica flakes are deposited in the long series of troughs, which are commonly about 700 ft. in length. These troughs are opened and the settlings scraped out at intervals.

The kaolin carried in suspension by the water flowing through this long zigzag channel is run into large vats or settling tanks. After standing for some time the clear water is drawn off and the mud is pumped into a filter press and squeezed by hydraulic pressure. The presses consist of a series of flat frames, sometimes iron, sometimes wood, strung on a central iron pipe. Bags of heavy cloth are placed in the spaces between the frames and connected with the central pipe which is connected with the pump.

The kaolin comes from the filter press in large cakes either round or square. These are sometimes coiled in a loose roll, sometimes simply doubled over, and are further dried either by storing them in racks where they are exposed to the air for several weeks, or put on a floor or in a tunnel heated by steam or hot air.

With the cheaper grade of clays it is customary to dispense with the use of a filter press and the clay is either dried in the settling tanks or transferred to a drying floor directly from the tanks.

There are objections to this method of washing kaolin, because the decrease in the current is not uniform and as the sand is deposited it narrows the channel and hence increases the current, resulting in some of the sand being carried over into the settling tanks. A modification consists in sending the current through a series of troughs, of which each successive one has double the capacity and twice the number of divisions; that is, the trough next to the washer would overflow through two openings into two troughs and each of these into two others. By this method it is only necessary to extend the series of troughs to get a finer grained material, and by varying the size and number of the troughs all the material can be separated into any number of grades desired, which may be an advantage when the sand is utilized. In some places boards on hinges are placed in the troughs to increase or diminish the rate of the current by lowering or raising the board.

Another method of washing used to some extent in this country is to put the clay into blungers instead of the log washer, where it is thoroughly disintegrated and stirred up into a slip which is run off through troughs to settling tanks. One of the most modern plants in Pennsylvania has nine blungers, from which the material is run into 12 large cypress settling tanks where the coarse material settles and is used for making bricks. The kaolin slip is carried thence into 10 large cypress tanks, whence it is pumped into the filter presses. The clay is removed from the press to the dry floor, heated by the exhaust steam from the engine and located close to the filter presses.

A method used to some extent in Europe is to have a horizontal cylinder closed at both ends within which revolves a horizontal shaft bearing iron arms. The water and clay are charged at one end and pass out at the other, much of the coarse sand remaining behind. This necessitates stopping the washer to remove the sand. They are of limited capacity and do not appear to be any marked improvement over the log washer which they resemble in principle.

There is evidently much room for improvement in the methods of preparing kaolin for the market, as now practiced in the United States. The washing plants are usually rather crude affairs, and are operated under the disadvantages of limited capital for original installation, insufficient working capital and small capacity. Proper attention is not always given to the different requirements for special uses and adapting the treatment to them, or closely classifying the products in marketable grades. There is plenty of good material, but it needs careful preparation to bring top prices. For turning out high-grade kaolin, as for the paper trade, it is usually easy to get rid of grit by elutriation and settling in the washing troughs, vats, etc., and iron can be avoided by selection of material. The chief trouble is often the presence of almost microscopic plates of mica, which are suspended in the washing process and sometimes cannot be eliminated in this way. This defect is remedied in part in one of the large Pennsylvania refining plants by passing the material through a very fine-meshed silk netting. It is claimed that this process gives a superior quality of clay for the paper trade.

The cost of mining and washing kaolin varies considerably according to the character of the deposit worked, some deposits having a heavy overburden while others have none, while some are troubled with water and others are not, etc. One concern in Pennsylvania, which produces 40 tons per day employing 65 men, mines and refines its product at a cost of $2 to $2·50 per ton of refined product, including interest on capital invested. The yield of china clay from crude kaolin varies widely in different localities. Some of the larger producers in Pennsylvania and Delaware give 40% as an average, but the South Mountain clays yield a higher proportion. In any case the yield is governed to some extent by the degree of refinement required; thus for some uses the presence of fine silica is not objectionable and consequently this mineral is not removed, while for other purposes it must be removed, which consequently lowers the percentage of yield.

Such kaolins as can be marketed at a good price in their natural state, without washing, of course have a decided advantage, as in the case of some of the fine French clays. For china and stoneware, indeed, the presence of feldspar and quartz is not necessarily a detriment, if these minerals are in fine particles and contained in uniform proportions, evenly disseminated throughout the mass; so that a very good kaolin may actually contain less than 50% of pure clay substance. Unfortunately these natural high-grade kaolins, requiring little or no preparation beyond selection, are not common.

Uses.—The two most important uses of kaolin are for the manufacture of chinaware and paper. For the former purpose, a white color, which means freedom from metallic oxides, is an important property. As a rule the purest kaolins make the best china clays, yet a clay may have a high percentage of silica and still be a valuable china clay, providing it has a uniform composition and the silica is in a finely divided state.

The paper trade uses about the same quantity of kaolin as the china trade. The essential qualities of a good paper clay are white or nearly white color, slaking or falling apart readily in water, remaining in suspension in the pulp, and freedom from grit. The clay is to give weight to the paper and to give a good surface for printing. Much of it is used in the manufacture of wall paper, where it forms a body for the colors. The properties imparted to a paper by kaolin are opaqueness, whiteness, increased softness on the surface, and an increased absorbing power for printer's ink, thereby allowing clear impressions of the type to be run off rapidly. It also has an affinity for aniline dyes that adds to its value in the production of tinted paper of the palest shades.

The percentage of clay used in paper manufacture varies with the grade of the paper. Clay is superior for this purpose to talc, gypsum and other mineral fillers, inasmuch as it remains better in suspension in the pulp, which is an important requirement. A good deal of the paper clay, however, has a creamy tint and does not give so white a body as do some of the other minerals which are used for this purpose.

Considerable quantities of kaolin are used in the manufacture of encaustic tiling and for white and enameled bricks, where it serves the same purpose as in china ware, namely, to furnish a white body to the ware.

Prices.—The largest pottery manufacturers are located at Trenton, N. J., and East Liverpool, Ohio, and prices for china clays are commonly governed by them.

One of the largest producers in eastern Pennsylvanja gave $8 per ton f. o. b. as the value of his kaolin in 1898. In the South Mountain region the prices were $10, $8 and $7 f. o. b. for grades Nos. 1, 2 and 3. These are about the maximum prices, and would make the average price at East Liverpool and Trenton from $10 to $11 per ton.

The usual price of the South Carolina paper clay at the mines is $5 per cask (2,100 lb.). The selling price in New York is $8·20.

DISTRIBUTION OF KAOLIN IN THE UNITED STATES.

Alabama.—Kaolin (halloysite) is mined near Sulphur Springs, De Kalb County, and Stevenson, Jackson County. It is used for the manufacture of pottery. Analyses and the results of physical tests of the Alabama kaolins were given in the *Engineering and Mining Journal*, Sept. 24, 1898, p. 369. They all burn white, lack plasticity, and are moderately refractory.

Arkansas.—Large deposits of kaolin occur in Pulaski, Saline, Pike, and Ouachita counties, but they have not been developed.

Delaware.—Kaolin has been mined at several localities in Delaware. The most productive locality is Hockessin, near the Pennsylvania state line, where at present there are three companies mining and washing kaolin for the market. The material occurs in irregular deposits in the schistose rocks, probably enlarged portions of former pegmatite veins which are now changed to kaolin. The mines are located on the northeast, east and west sides of the village of Hockessin. Some of the material is mined from open pits without any support to the sides, but most of it is got through the common circular shaft by using temporary removable timbering. The material is refined by the use of the log washer, longitudinal troughs, settling vats, hydraulic filter press, and drying in the air in racks under open sheds. The output of one of the largest plants is now 40 to 50 tons per day, but only recently it is said to have been 100 tons per day. Two grades of kaolin are produced—Nos. 1 and 2. The pure white kaolin is very tender or "short," so that it is necessary at times to mix some of the more plastic cream-colored clay with it to keep it from falling to powder when dry. There are several smaller mines in the near vicinity of Hockessin, but the output of all of them is handled by one of the three companies mentioned.

There is a kaolin mine and washing plant at Mermaid post office, several miles west of south from Hockessin. The plant has been in operation for five years, and ships about 10 tons per day. The refined kaolin is hauled four miles by wagon to the nearest railway point. The company once manufactured some fire bricks from the material, but that industry has been abandoned. The kaolin veins have a general N.E.-S.W. course, but, as at Hockessin, they are so variable in width that the workable portions are separated into irregular pockets. The methods of mining and washing are the same as at Hockessin.

Florida.—The kaolins of this State are of superior quality.* The chief deposits are situated in Lake County along Palaclakaha Creek. The deposits are apparently of sedimentary origin, lying in pockets in a micaceous sandstone. They have an overburden of 2 to 6 ft., and occur in beds 15 to 30 ft. thick, which on washing yield 25 to 50% of refined kaolin. Another authority states that all the so-called kaolins shipped from Florida are really ball clays and are shipped from Edgar, a few miles from Palatka on the St. John's River. The deposit is apparently the bed of an old lake and contains much sand. It is washed and bleached before shipping. The clay particles are said to be so fine that a considerable percentage passes through the canvas sacks in the filter press and is lost. Kaolin was discovered near Lake Eustis in 1889, and occurs in considerable quantities in Citrus, Lee and Hernando counties. The analysis shows it to approximate closely pure kaolin, and it is one of the best grades in the market. The following analysis of Florida kaolin was made by Johnson & Sons, London, England: Silica, 46·11%; alumina, 39·55; water, 13·78; iron, 0·35; sulphur, 0·07; magnesia, 0·14—total, 100%.

Georgia.—Kaolin (halloysite) is taken from two mines near Butler, Taylor County, the larger of which is said to ship 50 to 100 tons per day.† Kaolin is said to occur also in Cherokee, Pickens, Heard, McDuffie, Columbia and Richmond counties, but is not mined.

Indiana.—Near Huron, Lawrence County, there is a large deposit of white clay known as indianaite, and classed by Dana as a variety of halloysite. The following analysis shows it to be nearly a pure kaolin: Silica, 44·75%; alumina, 38·69; water, 15·17; iron oxide, 0·95; lime, 0·37; magnesia, 0·30; potash, 0·12; soda, 0·23. Microscopic examination shows it to be crystalline throughout. The upper half of the deposit is mainly composed of a massive snow-white clay, which has a smooth, unctuous feel. The lower half varies in color from a pale buff to a rich brown. The refractory properties of the clay are of the highest. The deposit apparently thickens further back in the hill, and is

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* "Florida Kaolin Deposits," by C. J. Memminger, in *Engineering and Mining Journal*, May 12, 1894.

† *Engineering and Mining Journal*, Oct. 1, 1898.

said to extend over a large area. This clay was first mined about 20 years ago and sold to a company in Cincinnati for use in making high-grade pottery. Later the property was sold to the Pennsylvania Salt Manufacturing Co., which for 10 years mined annually about 2,000 tons of the clay, which was used for making alum cake. The deposit has not been worked since 1891. In the area worked over the kaolin varied in thickness from 4 to 11 ft., averaging about 6 ft.

Kaolin appears in Martin, Greene and Owen counties, and has been opened and worked to a very limited extent in a few places. Like the Lawrence County kaolin, these deposits all consist of transported or sedimentary kaolins, and in places are pure enough for use without washing.

Maryland.—Kaolin is reported in Harford, Cecil, Anne Arundel, Montgomery, Howard, Carrol and Baltimore counties.

Massachusetts.—There is a large deposit of kaolin at Blandford that has been mined for the past ten years and used for the manufacture of white brick. The kaolin has been derived from large veins of coarse pegmatite, and is clearly the product of decomposition *in situ* in pre-glacial times. Following is an analysis of Blandford kaolin:* Silica, 52·03%; alumina, 31·76; water, 15·55; magnesia, 0·54—total, 99·88%. There are traces of iron, lime, soda, and potash.

North Carolina.†—North Carolina has several producing mines and numerous localities where a good quality of kaolin is known to occur but which has not yet been put on the market. Two miles north of Sylva, in Jackson County, is a kaolin vein which has been worked to a depth of 50 ft.

Analyses and pyrometric tests of the North Carolina clays are given in the following table:

COMPOSITION.

	1	2	3	4		1	2	3	4
	%	%	%	%		%	%	%	%
SiO_2	44·08	45·70	53·10	63·10	Insoluble residue	5·75	3·19	....	41·08
Al_2O_3	36·26	40·61	33·06	23·33	Total fluxes	2·99	4·75	2·47	5·11
Fe_2O_3	1·86	1·39	1·18	2·97	Clay substance	94·21	96·81	83·39	58·92
CaO	0·43	0.45	0·38	0·15	Quartz	5·75	0·07	14·98	35·27
MgO	0·20	0·09	0·08	0·09	Feldspar		3·12	1·58	5·81
Na_2O, K_2O	0·50	2·82	0·83	1·90					
H_2O (hygroscopic)	3·07	0·35	....	0·75					
H_2O (combined)	13·56	8·98	11·32	7·65					

(1) Washed kaolin from 2 miles south of Sylva; (2) washed kaolin from near Webster; (3) crude, very white, kaolin, near West's Mill; (4) washed white kaolin, burned buff, from near Troy.

PHYSICAL PROPERTIES.

	1	2	3	4		1	2	3	4
Specific gravity	2.31	2.43	2.31	2.34	Shrinkage, drying, %	8	6	6	3
Incipient fusion, F°	2,200	2,300	2,300	2,100	Shrinkage, burning, %	4	4	6	9
Vitrification, F°	2,450	2,500	2,600	2,300					
Viscosity, F°	2,700	2,700	2,700	2,500	Tensile strength, air-dried, lb. per sq. in.	15	20	15	10
Water to make paste, %	40	42	31	......					

Near Webster, in the same county, is a large deposit of kaolin in a pegmatite dike said to have a maximum thickness of 300 ft., which, however, includes

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* The analysis is by Dr. L. M. Norton. A more detailed description of the deposit is given by Prof. W. O. Crosby in *Technology Quarterly*, Vol. III., No. 3, August, 1890. Reference should be made also to the paper on "The Clay Industry of Massachusetts," elsewhere in this volume.

† *Bulletin* No. 13, of the North Carolina Geological Survey, "Clay Deposits and Clay Industry in North Carolina," by Heinrich Ries, describes the different clays of this State.

large masses of quartz and country rock. The clay is mined in pits sunk 80 to 125 ft. deep. The kaolin burns to a white body and is well suited to the manufacture of china ware. Another kaolin deposit, about half a mile northeast of Webster, has been opened recently.

Near West Mill, in Macon County, is a deposit of kaolin which is said to burn to a pure white color, thus making it a desirable china and porcelain clay. White and gray kaolins, both of which burn to a buff color, have been opened recently at a locality four miles west of Troy, in Montgomery County, while recently a promising bed of kaolin has been exploited near Bostick's Mills, 14 miles north of Rockingham, Richmond County. It burns to a dense body with a yellowish tint.

*Pennsylvania.**—There were seven kaolin washing establishments in operation in Pennsylvania, one of which was temporarily idle during the summer of 1898.

The Pennsylvania kaolins may be divided into two quite distinct and well-marked groups. The Delaware and Chester County kaolins, with probably one exception, are residual deposits, formed by the decay of pegmatite veins. They have had an established reputation in the pottery trade for many years. The South Mountain kaolins are sedimentary deposits resulting from the disintegration of an argillaceous slaty rock. They have been on the market a shorter time than the residual kaolins, but have made a rapidly increasing output.

One of the oldest kaolin producing localities in the State is that near Kaolin post office, in Chester County, where the mineral has been produced since 1839. At the present time there are two companies operating mines, each turning out from 40 to 50 tons of refined kaolin per day. Nearly all the kaolin from this locality is shipped to the large potteries in East Liverpool, Ohio, and Trenton, N. J. Both of the companies make a partial use of the by-products. One company screens the sand, pulverizes it, and sells it for ground flint to the potteries. The other company uses the sand for making silica brick for furnace use. But despite the use made of the sand there is a large accumulation of it at both works. The kaolin vein is about 120 to 150 ft. wide, with at least one parallel vein, and perhaps others. It is all mined from the circular shafts with the temporary timbering previously described. The deepest shaft sunk was 130 ft., but most of them are less than 100 ft., generally 80 to 100 ft., as it is claimed that the expense of hoisting more than 100 ft. is too great to pursue with profit. As all the openings are filled with waste material except the one in which the work is being carried on, there are no rock exposures except the kaolin in the bottom of the one or two shafts in actual operation. Several large bowlders of coarsely crystalline granite or pegmatite have been left on the surface in one place, and this is the only solid rock in sight at the mines. The kaolin is a decomposed pegmatite vein in the mica schists. The apparent course of the vein, as shown by the openings made by the two companies, is N.E.-S.W., but neither the width nor the linear extent can be accurately determined from the openings made. Near the surface the kaolin has a buff or

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* The description of the kaolins of Pennsylvania is based on field work by the writer during the summer of 1898, under the direction of the authorities of the Pennsylvania State College, and is here published, in advance of the college report, with the consent of the president of the college.

yellow tint due to the admixture of yellow soil from other rocks. The yellow color extends several feet below the surface, although in a few places the white clay is exposed.

The Hockessin, Del., kaolin mines and works are but two or three miles south of east from the kaolin mines described above, and are closely related to them in character and mode of occurrence.

Kaolin mines have been operated for many years near Brandywine Summit, in Delaware County, and clay of high grade produced in large quantities. The mines are controlled by two companies, both of which have ceased to wash kaolin. One of them is now producing ground feldspar in large quantities, and the other is manufacturing ornamental bricks from the low-grade kaolins that were thrown aside when high-grade kaolin only was sought. The reason given for abandoning (in 1895) kaolin mining and washing is that the best kaolin was exhausted from near the surface, and the water entered in such abundance that it was not profitable to work the deeper parts of the bed, especially at the present low prices of kaolin. Kaolin from this mine had an excellent reputation in the markets.

About a mile southeast of Elam post office, and 2 or 3 miles southeast of the Brandywine Summit kaolin mines, is a deposit of white clay which was formerly mined and used in part for making fire brick and some of it shipped to distant points in the crude state for use in lining furnaces. The clay is apparently a sedimentary deposit, and is probably the material from some nearby kaolin vein which was deposited here in a recent geological period. More thorough exploration might show kaolin of good quality. The reports given of former workings would indicate the presence of a large body of refractory white clay.

About 1½ miles north of Glen Loch Station, on the north side of Little Chester Valley, is a kaolin mine and washer that have been in operation for four years, except during the summer of 1898, when they were temporarily idle. The mine is near the contact of the Chester Valley (Ordovician) limestone and the North Valley Hill (probably Cambrian) sandstone. The Glen Loch kaolin bears a remarkable similarity to the white clay deposits of the South Mountain.

Probably the most extensive deposits of white clay in Pennsylvania are to be found in the South Mountain region in Cumberland, Franklin, Adams and York counties. The most extensive developments have been on the Gettysburg & Harrisburg Railway, south of Mt. Holly, and on the Hunter's Run & State Belt branch railway. The occurrence of white clays in this region has been known for a long time, but it was only a few years ago that any attempt was made to put them to commercial use. There are a great many large iron ore mines in which the ore is associated with the white clay. In nearly all these iron mines, which were worked extensively years ago, the white clay was found sometimes underneath the ore and sometimes overlying it. In the latter case it was necessary to remove the clay to get the ore, and it was dumped out as so much waste material, just as the ore is thrown aside at the present time where it interferes in the removal of the clay. In the reports of the Second Geological Survey of Pennsylvania bearing on this region there is mention of the white clays at nearly all the iron mines described, but there is no intimation of any possible commercial value for them.

The first use made of the clay was about six years ago, when the Penn Tile Works were established at Aspers, about the same time or shortly after the brick works at Pine Grove began operations. About three years ago two clay washing plants were established on Mountain Creek, between Pine Grove and Mt. Holly Springs. More than a year ago a clay washing plant was started at Latimore, and in 1898 another large plant started at Mt. Holly Springs. There is one party prospecting for a good body of the clay near Mt. Holly, and another clay washing plant is soon to be put in operation at or near Dillsburg.

The works at Pine Grove manufacture ornamental bricks of different kinds. The white clay together with some colored clay is used simply as a bond, the body of the material being a black slate for reddish-colored bricks, and a greenish talcose schist (soapstone) for light-colored ones. The slate is brought to the works by railway from the slate quarry, about 3 miles southwest of the works; the soapstone is hauled by wagon from the quarry, a quarter of a mile from the works, and the clay is brought by rail from Laurel Station, 3 miles away. By different mixtures of these materials this company manufactures a variety of ornamental bricks of different colors, shades and sizes.

At Henry Clay Station, on the Hunter's Run & State Belt Railway, about 3 miles below Laurel, near the large Henry Clay iron mine, is a clay washing plant well equipped and capable of turning out 25 to 30 tons of refined clay per day. It has been in operation about three years. The clay is mined in shafts 90 to 100 ft. deep on the mountain side, a mile or more east of the works, from which point it is hauled by wagon to the works on the railway.

At Crane's Siding, on the same railway, one mile above Hunter's Run Station, is a clay refining plant which has been in operation three years and ships about two carloads (50 to 60 tons) per day. The clay is obtained at the long-since abandoned Crane iron ore mine, where it underlies the ore, but at a high angle on the steep south slope of the South Mountain. The mining is done in underground galleries, and the clay is run out on tramcars and tipped on a platform, from which it is carted to the works. The refined clay is shipped to wall paper manufacturers and potters.

Near Latimore post office, on the south side of South Mountain, 6 miles southwest of Dillsburg, the nearest railway point, a clay washing plant has been recently constructed and has washed several carloads of clay, but has not yet put any on the market. Part of the clay is mined near the works and part is brought from another mine, 2 miles or more north of the works. It is rumored that another plant will soon be established at or near Dillsburg.

The last company to begin operations in the South Mountain district has located its plant at Mt. Holly Springs, and obtains its clay from Upper Mill Station, on the Gettysburg & Harrisburg Railway, and from the Henry Clay mine on the Hunter's Run & State Belt Railway. From the first point the clay is moved to the works on wagons; from the second it is taken on railway cars. The company has leased clay at other points in the vicinity, but is only working these two deposits at present. This company has one of the largest and best equipped clay working plants in the State. It has facilities for washing and refining the clay in large quantities, and utilizes the by-product from the refined clay to make white brick for building purposes. The company began operations in the summer of 1898. The machinery was started in July, and the first kiln of brick was burnt in September. It expects to refine clay for the wall paper and pottery trades and use the screenings for high-grade white brick.

The South Mountain clays are sedimentary, unlike the kaolins of southern Chester County. They are probably of Cambrian age, apparently occurring near the contact of the South Mountain sandstones and shales of supposed Cambrian age and the Cumberland Valley limestone of Lower Silurian age. The strata are so broken, crumpled and disintegrated that it is not possible in many places to locate accurately the position of the clay beds in geological series. In so far as they can be located they appear to be at or near the top of the supposed Cambrian rocks.* They are closely associated with extensive deposits of iron ore. There are a great many large iron ore mines that were operated extensively a few years ago, and in all these mines that occur in or close to the mountains the white clay exists in large quantities, and in many places the clay is stained in streaks, bands and irregular patches by yellow, brown and red iron oxides. In some places the ore overlies the mass of the clay, and in other places it underlies the clay, which is the result of the folding of the strata.

The developments at the present time, in fact the developments 20 years or more ago, were and are sufficient to show that enormous quantities of beautiful white clay occur in this region. The works erected during the last few years show that the commercial importance of the clay is now established, and the developments will be still greater as their quantity and quality become better known. Owing to the association of iron ores with the clay large quantities of it will be found in all the localities that are stained with the iron oxides, and the company with sufficient capital to use these lower grade clays along with the higher grade ones will have the advantage, as in nearly all localities much low grade clay must from necessity be handled in working the best grades.

White clays in the Cambrian rocks are not peculiar to the South Mountain locality, but occur in other districts as well. Similar clay has been found, but only slightly developed, in rocks of the same age, on Neversink Mountain on the outskirts of Reading, where some one has opened a deposit of white clay resembling that at South Mountain. The Glen Loch mines, described above, may be near the same horizon.

A white clay, somewhat similar in character but in smaller quantities, occurs with some of the iron ores in the Nittany Valley. This is probably at a higher horizon but may have the same source.

It is possible that a careful search in the same formation in other parts of Pennsylvania would reveal similar clays in varying quantities. Similar clays are mentioned in the South Mountain near Easton, but so far as known have had no commercial use as yet. They also occur in Tennessee and at other points along the Great Valley of the Appalachians.

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* The so-called Chickies or Hellam quartzite, supposed by the geologists to be Cambrian because it unconformably underlies the Trenton limestone. By some writers it has been called the Potsdam sandstone.

CHEMICAL COMPOSITION OF PENNSYLVANIA KAOLINS.

No.	Locality.	SiO_2	Al_2O_3	Fe_2O_3	CaO	MgO	K_2O	Na_2O	H
		%	%	%	%	%	%	%	%
1	Brandywine Summit.....	67·71	20·53	7·78	0·39	0·04	0·29		3·
2	Brandywine Summit.....	46·26	36·25	1·64	0·19	0·32	1·69	0·85	13·
3	Glen Loch................	51·90	31·29	....	Trace.	1·52	4·01	2·99	8·
4	Mt. Holly Springs, crude.	84·05	9·44	0·28	0·23	1·35	2·37	0·28	2·
5	Mt. Holly Springs, refined	73·80	17·30	0·35	....	1·18	2·49	0·20	4·
6	Henry Clay Mine, refined.	73·30	17·43	0·37	0·02	1·28	2·99	0·17	4·
7	Chestnut Hill............	67·10	20·10	3·90	0·10	0·70	2·00		5·

MINERALOGICAL COMPOSITION OF PENNSYLVANIA KAOLINS.

No.	Locality.	Clay.	Feld-spar.	Sand.	Water.	No.	Locality.	Clay.	Feld-spar.	Sand.	W
		%	%	%	%			%	%	%	
1	Kaolin	82·75	0·59	4·00	12·66	3	Crane Siding......	84·47	0·19	10·96	4
2	Henry Clay Stati'n	56·75	0·54	37·17	5·54	4	Latimore..........	49·77	1·31	45·52	3

South Carolina.—South Carolina contains several kaolin deposits of comm cial importance, some of the best paper clays coming from this State. T most productive locality is in Aiken County, near the center of the State, ab 130 miles from Charleston. The clay is a sedimentary deposit, and occurs beds 4 to 12 ft. thick, covered with 6 to 14 ft. of overburden. It is rema ably free from grit, some of it pure white and some yellowish white in co The clay is mined by negroes, packed in casks, and shipped by rail Charleston and thence by steamer to New York. The selling price in N York is $8·20 per cask (2,100 to 2,300 lb.). The price at the mines is ab $5 f. o. b., but inferior grades can be bought as low as $3·75. The actual for labor and material is $2·50 f. o. b. Kaolin is said to have been wor to some extent at other localities. The annual output from the South Caro mines is about 27,000 tons.

Tennessee.—Kaolin has been found but not developed at Tullahoma, Pi Flats, Loudon, Smithville, and Cookville.

Texas.—Kaolin was worked formerly near Austin, but the production never large, and no pits are known to me to be operated in this State at prese

Vermont.—Associated with the limonite and manganese ores in Vermont considerable deposits of kaolin, which lie in a narrow belt of Tertiary drift n the western border of the State. The kaolin has been worked at several poi along this belt. During 1897 mines were operated at Monkton and Benni ton, producing in all about 3,000 tons. The material lies near the surface is mined by hand at small expense. It is shipped by boat and rail to vari points, and most of it is used in paper making.

COAL AND COKE.

The year 1898 was one of great prosperity in all parts of the civilized world, and there was consequently a large increase in the production of coal. In the United States this was especially large. The details of the American production are given in the following tables:

TOTAL PRODUCTION OF COAL IN THE UNITED STATES.
(In tons of 2,000 lb.)

States.	1897.			1898.		
	Tons.	Value.		Tons.	Value.	
		Totals.	Per Ton.		Totals.	Per Ton.
Bituminous:						
Alabama	5,893,771	$5,186,518	$0·88	6,466,741	$5,496,730	$0·85
Arkansas	850,000	926,500	1·09	a1,134,064	1,202,108	1·06
California	87,449	196,255	2·24	135,795	334,999	2·47
Colorado	3,501,563	4,692,094	1·34	4,125,206	7,219,110	1·75
Georgia	196,268	147,479	0·75	240,000	156,455	0·65
Illinois	a20,072,758	14,472,529	0·72	a18,599,299	14,567,598	0·78
Indiana	4,439,489	3,551,591	0·80	5,435,896	4,729,230	0·87
Indian Territory	a1,343,376	1,813,558	1·35	a1,458,098	1,822,622	1·25
Iowa	4,523,494	5,098,104	1·13	4,117,339	4,759,967	1·16
Kansas	3,672,195	3,931,707	1·07	3,800,405	4,193,160	1·09
Kentucky	3,247,542	2,403,181	0·74	3,492,243	2,479,493	0·71
Maryland	4,411,924	3,353,062	0·76	4,621,618	3,142,700	0·68
Michigan	152,850	236,917	1·55	178,035	263,492	1·48
Missouri	a2,429,388	2,684,757	1·10	a2,838,152	3,148,862	1·10
Montana	a1,607,637	2,881,481	1·79	1,450,471	1,902,252	1·31
Nebraska	500	1,750	3·50	500	1,750	3·50
New Mexico	a729,653	1,189,334	1·63	a863,583	1,416,890	1·64
North Carolina	20,611	28,855	1·40	6,144	8,602	1·40
North Dakota	120,000	120,000	1·00	124,526	124,526	1·00
Ohio	12,465,533	9,598,460	0·77	14,053,829	10,680,910	0·76
Oregon	100,929	235,377	2·33	65,871	183,543	2·79
Pennsylvania	54,674,272	36,631,762	0·67	64,247,859	38,548,715	0·60
Tennessee	2,888,849	2,329,534	0·81	3,084,748	2,340,346	0·76
Texas	615,540	991,019	1·61	726,133	1,103,722	1·52
Utah	506,455	607,746	1·20	571,417	583,084	1·02
Virginia	1,418,746	879,622	0·62	1,640,000	984,000	0·60
Washington	1,489,815	3,325,480	2·23	1,988,288	3,301,978	1·66
West Virginia	13,950,000	9,067,500	0·65	16,499,840	9,715,754	0·59
Wyoming	2,744,960	3,431,200	1·25	3,181,905	4,006,756	1·26
Total bituminous { Sh. tons.	148,155,567	120,013,372	0·81	165,208,025	128,419,354	0·78
Met. tons	134,405,849	....	0·89	149,875,737	....	0·86
Cannel·						
Kentucky { Sh. tons.	56,511	153,145	2·71	49,889	134,700	2·70
Met. tons	51,266	....	2·99	45,259	....	2·98
Anthracite:						
Colorado	64,097	150,628	2·35	48,831	134,285	2·75
Pennsylvania	52,581,036	85,707,089	1·63	52,799,774	81,311,652	1·54
Total anthracite { Sh. tons.	52,645,133	85,857,717	1·63	52,848,605	81,445,937	1·54
Met. tons	47,759,351	....	1·80	47,943,940	....	1·70
Grand total coal { Sh. tons.	200,857,211	206,024,234	1·03	218,106,519	209,999,991	0·96
Met. tons	182,216,466	....	1·13	197,864,936	....	1·06

(a) Fiscal year.

TOTAL PRODUCTION OF COKE IN THE UNITED STATES.

(In tons of 2,000 lb.)

States.	1897.			1898.		
	Tons.	Value		Tons.	Value.	
		Totals.	Per Ton.		Totals.	Per Ton.
Alabama	1,381,252	$2,417,191	$1·75	1,541,250	$2,620,125	$1·70
Colorado	320,738	882,029	2·75	445,925	1,226,294	2·75
Georgia	33,474	42.478	1·27	50,000	63,500	1·27
Illinois	3,000	4.950	1·65	3,000	4.950	1·65
Indiana	7,000	11,200	1·60	1,521	2,662	1.75
Indian Territory	(*a*) 28,600	64.350	2·25	(*a*) 34,810	78,323	2·25
Kansas	10,000	20,000	2·00	10,000	20,000	2·00
Kentucky	32,285	53.916	1·67	21,394	30,593	1·43
Montana	(*a*) 66,182	440,637	6·66	70,235	529,825	7·54
New Mexico	1,500	3,375	2·25	(*a*) 2,275	5,119	2·25
Ohio	95,000	228,000	2·40	100,000	250,000	2·50
Pennsylvania	8,533,291	15,786,588	1·85	10,671,920	21,663,998	2·03
Tennessee	368,769	667,656	1·81	394,545	710.181	1·80
Utah	25,000	56,250	2·25	28,327	127,809	4·51
Virginia	320,000	448,000	1·40	525,000	703,500	1·34
Washington	25.000	87,500	3·50	62,720	219,520	3·50
West Virginia	1,494,000	1,942,200	1·30	1,916,482	2,184,789	1·14
Wyoming	23,800	47,600	2·00	18,393	64,375	3·50
Total coke — Short tons	12,768,891	23,203,920	1·82	15,897,797	30,505,563	1·92
Total coke — Metric tons	11,583,862	....	2·00	14,422,387	....	2·12

(*a*) Fiscal year.

UNITED STATES PRODUCTION OF COAL, 1894–98. (IN TONS OF 2,000 LB.)

Year.	Alabama.	Arkansas	California.	Colorado.	Georgia.	Illinois.	Indiana.	Indian Territory	Iowa.	Kansas.
1894	4,381,295	*a*723,860	94,754	2,994,028	354,111	*a*17,113,576	*a*3,065,394	1,072,542	*a*3,776,373	3,611,214
1895	5,680,410	900,671	80,115	3,582,393	414,310	17,735,864	4,312,084	1,228,440	3,995,836	3,190,843
1896	5,747,698	889,750	70,649	3,371,633	246,012	*a*19,786,626	*a*4,068,124	1,235,333	4,115,000	3,191,748
1897	5,893,771	850,000	87,449	3,565,660	196,268	*a*20,072,758	4,439,489	*a*1,343,376	4,523,494	3,672,195
1898	6,466,741	*a*1,134,064	135,795	4,174,037	240,000	*a*18,599,299	5,435,896	*a*1,458,098	4,117,359	3,860,405

Year.	Kentucky	Maryland	Michigan	Missouri.	Montana.	Nebraska.	New Mexico.	North Carolina.	North Dakota.	Ohio.
1894	2,967,195	3,101,082	....	*a*2,383,322	688,780	....	323,721	13,157	*b*65,000	11,902,678
1895	3,207,770	3,479,499	....	2,283,081	1,104,854	....	367,442	15,386	79,850	13,683,879
1896	3,128,818	4,062,716	85,465	*a*2,420,147	*a*1,157,722	....	655,370	14,970	94,500	12,912,608
1897	3,304,053	4,411,924	152,850	*a*2,429,388	*a*1,607,637	500	*a*729,653	20.611	120,000	12,465,533
1898	3,542,132	4,621,618	178,035	*a*2,838,152	1,450,471	500	*a*863,583	6,144	124,526	14,053,829

Year.	Oregon	Pennsylvania.		Tennessee.	Texas.	Utah.	Virginia	West Virginia.	Washington.	Wyoming	Totals.
		Bituminous.	Anthracite.								
1894	42,509	41,867,188	51,828,405	2,589,664	469.904	453,601	925,837	*a*10,559,926	1,183,660	2,224,135	169,867,781
1895	65,918	51,813,112	58,660,657	2,319,720	499,668	530,713	1,083,229	12,355.113	1,184,619	2,197,914	189,053,390
1896	90,715	50,273,656	48,074,330	2,663,714	585,000	503,243	1,018,237	13,509,964	1,295,313	2,273,928	187,657,250
1897	100,929	54,674,272	52,581,036	2,888,849	615,540	506,455	1,418,746	13,950,000	1,489,815	2,744,960	200,857,211
1898	65,871	64,247,859	52,799,774	3,084,748	726,133	571,417	1,640,000	16,499,840	1,988,288	3,181,905	218,106,519

(*a*) Fiscal year. (*b*) Estimated.

EXPORTS OF COAL AND COKE OF DOMESTIC PRODUCTION. (IN LONG TONS).

Year.	Anthracite.		Bituminous.		Totals.		Coke.	
	Quantity.	Value.	Quantity.	Value.	Quantity.	Value.	Quantity.	Value.
1894	1,440,625	$6,359,021	2,195.716	$4,970,270	3,636,341	$11,329,291	42,819	$154,828
1895	1,470,710	5,937,130	2,211,983	4,816,847	3,682,693	10,753,977	117,293	424,174
1896	1,350,000	5,925,506	2,276,202	5,072,818	3,626,202	10,998,324	151,062	553,600
1897	1,298,768	5,836.730	2.399,263	5,326,761	3,698,031	11,163,491	173,034	546.066
1898	1,350.948	5,712,985	3,152,457	6,699,248	4,505,405	12,412,233	199,562	600,931

IMPORTS OF COAL AND COKE INTO THE UNITED STATES.

Year.	Coal.					Coke.		
	Anthracite.	Bituminous	Totals.			Long Tons.	Metric Tons	Value.
	Long Tons.	Long Tons.	Long Tons.	Metric Tons	Value.			
1894	90,068	1,244,330	1,334.398	1,355,748	$4,063,580	27,591	28,032	$66,463
1895	141,337	1,229,109	1,370,446	1,392,373	3,970,761	26,448	26,871	71,366
1896..........	101,689	1,246,991	1,348,680	1,370,259	3,725,832	43,372	44,066	114,713
1897..........	3,281	1,276,963	1,280,244	1,300,728	3,432,154	35,193	35,756	98,558
1898..........	3,149	1,269,775	1,272,924	1,293,291	3,571,600	41,185	41,844	142,334

PRODUCTION AND CONSUMPTION OF COAL IN THE UNITED STATES. (IN LONG TONS.)

Year.	Production.	Imports.	Total Supply.	Exports.		Consumption.	
				Domestic.	Foreign.	Tons.	Metric Tons.
1894....	151,674,804	1,334,398	153,009,202	3,636,341	877	149,371,984	151,761,936
1895.............	168,797,669	1,370,446	170,168,115	3,682,693	3,440	166,481,982	169,145,694
1896.............	167,551,116	1,348,680	168,899,796	3,626,202	5,204	165,268,390	167,912,684
1897.............	179,336,796	1,280,244	180,617,040	3,698,031	5,297	176,913,712	179,744,331
1898.............	194,737,963	1,272,924	196,010,887	4.503,405	2,890	191,504,592	194,568,665

The activity in bituminous coal mining was general in all parts of the United States, and the prosperity was unchecked by any important labor difficulties, such as the great strike in the West in 1897. There were several small strikes, the most noteworthy being that in the neighborhood of Pana and Virden, in Illinois; but these were confined to local districts and caused no general cessation of production. The chief cause of these difficulties is probably the rapidity with which the trade has developed, the result being an excessive competition which not only keeps the price low but compels frequent stoppages in the working of the mines. In most cases the nominal mining rate is sufficient to give the miner fair wages if his work is steady, but in many of the more important districts the miner is not sure of more than half time.

In explanation of the table showing the imports and exports of coal and coke it may be remarked that nearly all the anthracite and a large part of the bituminous coal exported goes to Canada. However, the increase in the exports in 1898 was due to the shipment of considerable quantities of Pocahontas and New River coal to South America, where an opening developed through the scarcity of Welsh coal caused by the strike in Wales. The imports of coal into the United States are received chiefly on the Pacific coast, which derives part of its supply from British Columbia and Australia.

Alabama.—The fear of interruption of ocean traffic by the Spanish war in the early part of the year led to an increased demand for Alabama coal from the Gulf ports. Later in the year the iron and steel industry called for increased supplies, and consequently there was an increase in the coal production.

Colorado.—Labor difficulties reduced the production in the northern lignite field, which supplies Denver with much of its fuel, but in the bituminous and semi-bituminous field there was a large increase. The output in 1898 was 2,085,034 tons of bituminous, 1,462,493 tons of semi-bituminous, 577,679 tons of lignite, and 48,831 tons of anthracite, making a total of 4,174,037 tons.

Illinois.—The decrease in the production of coal in this State in 1898 was due partly to labor troubles and partly to the increased use of Ohio and West Virginia coal in Chicago and other manufacturing centers. The increased consumption of Missouri coal in St. Louis, which formerly drew almost its entire supply from southern Illinois, also affected the latter region.

Kentucky.—The coal production of this State in 1898 was the largest in its history. At the end of the year 95 companies were working 128 mines, employing a maximum force of 8,408. The tendency of the operators is more and more toward machine mining, 43% of the product being obtained in this way in 1898, against 41% in 1897, 30% in 1896, and 26% in 1895. There are 171 machines in use, of which 122 are operated with compressed air, and the remainder with electricity.

Ohio.—The coal production of Ohio increased in 1898, notwithstanding the fact that some of the large operators in the Hocking Valley were compelled to shut down for several months, owing chiefly to the competition of West Virginia. However, this stoppage was compensated for by greater activity in the closing months of the year.

Pennsylvania.—This State continued to be the largest producer of bituminous coal. The increase in production in 1898 was due to the prosperity of the iron industry, and the consequent heavy demands for coal and coke. The anthracite production did not change materially from 1897, notwithstanding the larger demand for this kind of coal from Western cities. As in previous years, the chief market for anthracite was in the Atlantic States, and principally for domestic use. The gradual displacement of anthracite by bituminous coal for steam purposes continued in 1898.

West Virginia.—The large increase in the coal production of this State in 1898 was due to the general business activity and the natural advantages possessed by the West Virginia operators, which, combined with the excellent quality of the coal, have enabled them to undersell all other Western producers. The West Virginia operators pay a lower mining rate than elsewhere, but on the other hand the miners generally have steadier work and are thus able to make better wages. During the great strike of 1897 the West Virginia mines were unaffected, and their proprietors took advantage of the situation to extend their market, and the new trade gained thereby has been largely retained. In 1898 West Virginia made large shipments to Michigan and Minnesota through Lake Erie ports, and also held a good deal of the trade in Chicago and other Northwestern cities in competition with mines much nearer the point of consumption.

Other States.—Nearly all of the States east of the Mississippi, other than those mentioned above, made increased outputs in 1898, which was due in most cases to the increased demand for coal for manufacturing purposes, and this was especially marked in the second half of the year. West of the Mississippi the same conditions of prosperity prevailed, although the mines of Missouri, Kansas and Iowa did not increase their production so much as in the Eastern States. On the Pacific coast Washington made an important increase, but the outputs of Oregon and California were small, no important new sources of production having been discovered.

COAL PRODUCTION OF THE CHIEF COUNTRIES OF THE WORLD. (IN METRIC TONS.)

Year.	Africa	Australasia.					Austria-Hungary	Belgium.	Canada.		France.	Germany.
		New South Wales.	New Zealond.	Queensland.	Tasmania	Victoria.			Alberta, British Columbia	N. Brunsw'k, Nova Scotia.		
1893	748,578	3,330,939	691,548	268,200	43,893	93,202	26,548,604	19,407,251	925,298	2,556,085	25,738,073	95,426,153
1894	1,015,653	3,730,829	745,000	275,036	31,052	174,407	26,905,490	20,534,501	1,327,250	2,168,340	27,416,905	98,805,702
1895	1,402,182	3,500,000	760,000	275,000	33,000	204,000	27,250,000	20,414,849	1,403,911	1.782,732	28,236,039	103,957,639
1896	1,787,908	3,972,068	805,537	377,322	44,286	230,187	33,676,411	21,252,370	1,138,682	2,259,168	29,190,000	111,471,106
1897	2,003,174	4,453,728	854,164	364,142	43,210	240,053	35,939,416	21,492,446	1,178,010	2,383,567	30,798,000	120,474,485

Year.	India.	Italy,	Japan.	Russia.	Spain.	Sweden	United Kingdom.	United States.	All Other Countries	Totals.
1893..	2,570,332	317,249	3,370,178	7,613,724	1,531,810	199,933	166,971,440	159,919,176	1,500,000	518,105,087
1894..	2,620,910	271,294	4,329,397	8,763.340	1,659,264	213,634	188,277,525	154,211,308	1,600,000	543,811,506
1895..	2,650,000	250,000	4,843,936	9,098,477	1,774,560	205,000	194,350,604	177,595,679	1,750,000	580,186,311
1896..	3,909,582	276,197	5,100,005	9,377,551	1,852,947	225,848	198,487,040	170,242,657	*e*2,000,000	597,676,866
1897..	4,128,137	314,222	*e*5,500,000	*e*9,750,000	2,019,000	224,343	205,373,631	182,216,466	*e*2,000,000	631,750,204

(*e*) Estimated.

North America.—The output of coal increased in 1898 both in Nova Scotia and in British Columbia. A further increase may be expected in 1899.

The production of coal in Mexico in 1898 was 367,193 metric tons, of which 80,950 were converted into coke, producing 39,443 tons. The production of coal in 1897 was 359,070 metric tons, of which 63,215 were converted into coke, producing 31,877 tons. The production of coal in 1896 was 253,104 metric tons, and of coke 27,215. The only coal measures worked successfully in Mexico so far are those at Sabinas, near the northern border of the Republic, and far distant from the centers of Mexican trade and commerce. In 1898 a new discovery was made near Jiquilpan, on the south shore of Lake Chapala (State of Michoacan), and investigations of the district between Yurecuaro on the Mexican Central Railway and Ameca in northwestern Jalisco revealed the existence of considerable quantities of lignite. These coal measures, which are probably of Tertiary age, are overlain by igneous rocks. It has been ascertained that two seams from 2 to 50 in. thick cover an area of 30 miles square. In quality this lignite is said to compare favorably with that mined in Montana.

South America.—Argentina and Chile are the only coal-mining countries of South America. The output of Chile is the more important, but it is not sufficient for the domestic consumption, and large amounts have to be imported. The imports of coal into Chile are shown in the following table, reported in long tons, for which we are indebted to Jackson Bros., of Valparaiso:

Year.	Steam Coal.						Smelting Coal.		
	Hartley.	Orrell.	Other Classes.	Australian.	North American.	Total.	English.	Australian.	Total.
	Tons.	Tons.	Tons.	Tons.	Tons.	Tons.	Tons.	Tons.	Tons.
1893..........	171,473	21,540	42,328	127,118	....	362,454	15,223	23,152	38,375
1894..........	208,639	13,082	104,132	206,586	....	532,439	20,120	52,476	72,596
1895..........	189,119	16,412	157,498	227,014	....	590,043	22,809	41,988	64,797
1896..........	151,547	9,254	115,024	290,073	....	565,898	14,889	38,186	53,075
1897..........	126,909	....	52,979	296,173	....	476,061	7,646	50,172	57,818
1898..........	134,177	8,591	100,480	270.085	10,608	523,941	8.599	42,763	51,362

Europe-Asia.—The strike of the coal miners in Wales, which lasted until September, greatly curtailed the output of this part of Great Britain; but owing to the large demand production was very heavy when work was resumed. The Scotch and North of England fields were active throughout the whole year. An interesting development in the industry was the discovery of coal in Kent by boreholes. The total production of coal in Great Britain in 1898 was 202,042,243 long tons, against 202,119,196 in 1897. The exports of coal and coke of all kinds amounted to 36,546,152, against 36,096,918 in 1897.

France produced 32,439,786 metric tons of coal in 1898, which was an increase of 5·5% over 1897. The northern basin furnished 63% of the total.

Belgium produced 22,075,093 metric tons of coal in 1898.

Germany in 1898 produced 96,279,992 tons of bituminous coal and 31,648,498 tons of lignite, a total of 127,928,490 metric tons.

Russia produced 10,250,000 metric tons in 1898, against 9,750,000 in 1897, the increase being derived chiefly from the Donetz basin and from the mines of western Siberia, the output of Poland remaining about the same as in the previous year. Russian railways were prohibited in 1898 from using foreign coal, which of course stimulated the domestic production. However, the Russian coal miners are obliged to compete with the cheap oil residuum from the Caucasus, which in many parts of the empire controls the fuel market. In Siberia the most important development of 1898 was the systematic attempt of the government to open the beds of lignite, anthracite and bituminous coal that were discovered several years ago in the Ossuri district, near Vladivostok. According to Russell L. Dunn excellent bituminous coal, superior to Japanese, has been proved to exist within easy reach of tidewater. Many of the seams are thick and can be worked cheaply. At the present time the output is small, only two seams being worked.

THE ANTHRACITE COAL TRADE OF THE UNITED STATES IN 1898.

The conditions governing the anthracite coal trade have been fully explained in previous volumes of THE MINERAL INDUSTRY. The production and shipments in 1898 and three years previous are shown in the following table:

PENNSYLVANIA SHIPMENTS AND PRODUCTION OF ANTHRACITE COAL.

Companies.	1895.		1896.		1897.		1898.	
	Long Tons.	Per Cent.	Long Tons.	Per Cent.	Long Tons.	Per Cent.	Long Tons.	Per Cent.
Philadelphia & Reading	9,905,059	21·3	9,019,533	20·9	8,395,411	20·2	8,219,814	19·6
Lehigh Valley	7,360,454	15·8	6,749,128	15·6	6,425,227	15·4	6,885,577	16·5
Central of New Jersey	5,335,785	11·4	4,999,003	11·6	4,730,860	11·4	4,626,386	11·0
Delaware, Lackawanna & Western	6,129,261	13·2	5,627,533	13·0	5,690,684	13·7	5,795,540	13·8
Pennsylvania Coal Co.	1,828,382	3·9	1,728,972	4·0	1,777,841	4·3	1,854,515	4·4
Delaware & Hudson and Lackawanna	6,151,148	13·3	5,835,621	13·5	5,646,853	13·5	5,613,186	13·4
Other lines	9,801,388	21·1	9,217,693	21·4	8,970,988	21·5	8,904,733	21·3
Total shipments	46,511,477	100·0	43,177,485	100·0	41,637,864	100·0	41,899,751	100·0
Used and sold at mines	3,255,803	......	3,022,424	......	5,309,490	......	5,242,904	......
Total production, long tons	49,767,280	......	46,199,909	......	46,947,354	......	47,142,655	......
Total production, short tons	55,739,353	......	51,742,898	......	52,581,036	......	52,799,774	......
Total production, metric tons	50,563,556	......	47,939,098	......	47,698,512	......	47,896,937	......

There are few published reports as to the present cost of mining and marketing anthracite coal, but figures are available for some of the companies. Mining costs the Delaware & Hudson Canal Co. $1·34 per ton, Lehigh Coal & Navigation Co. $1·31, Lehigh Valley Coal Co. $1·90, Philadelphia & Reading Coal & Iron Co. $1·71. It costs the Delaware & Hudson 41c. per ton additional to market the product, and the Philadelphia & Reading $1·13. It appears from these figures how small is the possible margin of profit to the anthracite producers. In fact, the Philadelphia & Reading Coal & Iron Co. reports the running expenses nearly equal to receipts, leaving nothing to pay interest on its liabilities, while the Lehigh Valley Coal Co.'s product costs it more than it receives for it. However, most of these concerns transport their own coal and realize their profits in its carriage. The Pennsylvania Coal Co., which confines itself strictly to mining, obtains low rates of freight from the railways, and pays regular dividends. This company, however, has the exceptional advantage that its coal lands were bought many years ago at a small cost, and it is not obliged to pay heavy royalties or interest on extravagant first cost of property, as many of its competitors do.

The general course of the anthracite trade in 1898 resembled that in 1897 and 1896. The quotations f. o. b. points on New York harbor at the beginning of the year were: Broken, $3·40@$3·50; egg, $3·75@$4; stove, $3·85@$4; chestnut, $3·50@$3·80. A cold wave in February stimulated consumption. The producers then got together and made out a price list as follows: Broken, $3·50; egg, $3·75; stove, $4; nut, $3·75. The companies kept faith in restricting production, so that by the end of February stocks were less than at the corresponding time in 1897. In March the weather was mild and demand dull, but the companies decided not to make any concessions in price, and the list established in February was maintained throughout the year. The small sizes used for steam were high during the winter, pea selling in March at $2·60 f. o. b. New York and buckwheat at $5·25. The output for the first quarter of the year was about 8,535,800 tons, compared with an estimate by the companies of 7,500,000 tons.

In April buying was stimulated by the cold wave and the panic caused by the beginning of the war with Spain. May opened with heavy shipments to seaboard points, contracted for during the war scare. This demand was satisfied by June, in which month the Eastern trade was dull; but movements up the Lakes were heavy, freights being low (20c. per ton was quoted from Buffalo to Duluth). Shipments to the West were limited, however, by the large stocks left on hand in Chicago territory by the mild winter. The total production of anthracite coal during the first half of the year was about 16,191,420 tons, against an estimate of 15,500,000; while the average net price f. o. b. New York was as follows: Broken, $3·36; egg, $3·64; stove, $3·87; chestnut, $3·60; pea, $2·21; buckwheat, $1·81; rice, $1·55.

Demoralization of the market began in July. One or two companies which had large stocks at Western points, brought over from 1897, sold at low figures (about $1 below the Chicago circular price). Other companies not only sold old coal at this figure, but contracted to supply freshly mined at the same price. This action became general, and the low prices caused thereby remained established for several months. August and September showed larger sales, although the warm weather restricted buying in the East; but in the West the trade was active, the companies finding it easier to sell coal at low figures there than to restrict production at the mines. Eastern consumers, appreciating the weakness of the producers' position, remained out of the market, awaiting lower prices, and proved correct in their forecast that these would come. By November 12 the top price for stove coal f. o. b. New York was \$3·60, while much was selling for less than \$3·50, the other large sizes being correspondingly low, particularly broken. The total output for the first nine months of the year was 28,003,830 tons, against an estimate of 26,500,000. By this time the companies had abandoned all idea of restricting the output, and coal was mined as fast as it could be shipped. A peculiarity of the trade was an unexpected scarcity of chestnut. In the East the anthracite market was prevented from utter demoralization by the coming of winter, December being unusually cold and the consumption of anthracite correspondingly heavy. Prices, however, remained low, though in Boston there was an advance of 25c. in December on account of the blizzard of November 28, which destroyed many vessels and interfered with coastwise traffic. In the West the closing of lake navigation and the continued good demand also led to an improvement in prices. Finally the market took an upward turn in New York, and at the end of the year an advance of 15 or 25c. was under consideration by the companies. The output for the second half of the year was 25,432,100 tons. The average selling price f. o. b. New York was: Broken, \$3·20; egg, \$3·42; stove, \$3·54; chestnut, \$3·43; pea, \$2·10; buckwheat, \$1·77; rice, \$1·51. The year was free from labor troubles, and the only thing that kept down production was the united action of the large companies during the first six months. In June and July many collieries did not work more than five days a month. When restrictions were removed operations were generally carried on full time, and during November and December the mines were busier than for several years.

During the first half year the so-called Anthracite Association was formed among the chambers of commerce of several towns in the coal region, with the idea of obtaining lower freight rates to New York and Philadelphia, doing away with alleged discriminations, and thus lowering the selling price and increasing consumption. This association, however, accomplished but little or nothing. The Anthracite Coal Operators' Association sent an agent to Europe to endeavor to develop a demand for anthracite there, but his mission did not result in anything of immediate importance. A number of the independent operators of Scranton and Wilkesbarre had surveyors in the field to lay out a line of railway from Scranton to New York, to be called the New York, Wyoming & Western; but the year closed with nothing definite done. The reorganization of the Philadelphia & Reading was completed, the whole property passing under the control of the new Reading Co. The controlling interest in the New York, Susquehanna & Western Co. was acquired by the Erie Railroad Co., largely increasing the latter's interest in the anthracite trade. The Delaware & Hudson Canal Co. finally decided to abandon shipments by its canal from Honesdale to Rondout, and made a closer contract with the Erie for its northern trade. Samuel Sloane retired from the presidency of the Delaware, Lackawanna & Western Railway Co.

THE BITUMINOUS COAL TRADE.

The main lines of railways, *i.e.*, those carrying coal from the mines to the seaboard shipping points, advanced rates 15c. a ton early in the year, but this advance was not maintained, secret rebates being given. The Atlantic seaboard bituminous coal trade was dull during the first quarter of the year. The contracts for the shipping season of 1898 were arranged unusually early, most of them being closed before March. They were made at prices about 5c. under those prevailing in 1897. The scare over the possible depredations of a Spanish fleet completely demoralized the coastwise traffic, and by the end of April freights from Philadelphia to Providence and Long Island Sound ports were $1·10@$1·15, and to Boston, Salem and Portland $1·25@$1·35, these prices being 50c. higher than at the beginning of the month. It was not long, however, before this scare was forgotten, and the demand for coal falling off, freights gradually declined and business became limited to shipments on season contracts. By the middle of June the trade was very dull, and this dullness continued through the summer. In October trade improved. The export business, which received an impetus through the strike of the Welsh coal miners, was good during the autumn, a considerable number of vessels which had been in the coastwise trade being engaged to take coal to South and Central America. The severe storm of November 28 disabled the coal-carrying fleet and caused considerable losses of coal. Freight rates climbed up, rose especially to points east of New York, and by December 10 $1@$1·10 was paid from Philadelphia to Sound ports, and $1·25 to Boston and Portland. The demand for coal was particularly heavy at Sound ports.

Late in November an attempt was made among the coal-carrying railroads to come to an agreement as to freights, but this failed. The price of soft coal declined most of the year. In the early summer quotations were: George's Creek, at Baltimore, $1·75; New River, at Newport News, $1·75; Pocahontas, at Norfolk, $1·75; Clearfield, at Philadelphia, $1·20@$1·75; poor grades, at New York, $1·50, the best grades, $2·25. By November quotations 10c. less than these figures were made. The subsequent interruption of navigation advanced prices for spot coal, and the year closed with prices above where they were at the beginning. Generally speaking the year was a good one. Soft coal continued to supplant anthracite, aided by the great difference in price. The low price helped to increase the consumption, although it did not leave a large margin of profit to the miners.

Chicago.—The market was rather inactive during the first six months of the year, but the remaining six months brought the year's total sales up to a very fair aggregate. The demand for soft coal used for domestic purposes has grown steadily, 1898 having surpassed all other years by far in the sales of the better grades. The immensely increased business among the railroad, iron and other industries made itself manifest to the trade during the closing months, an enormous tonnage having been placed; so much so that for a time there were doubts as to the supply holding out. Scarcity of cars for carrying coal, the decreased hours of the miners for working, and the fact that shippers were wholly unprepared for the business that poured in on them, were the reasons why the soft coal supply of Chicago got so low in November and December. Prices on spot coal for the larger part of the year were low, but later on stiffened, and the months of October, November and December brought with them prices that permitted a fair profit. Freight rates from the bituminous coal fields did not vary much during the year, the railroads having maintained prices extremely well, though they carried soft coal cheaper than ever before. The year 1899 opened with excellent prospects for the trade.

Pittsburg.—The Ohio River coal trade of Pittsburg was not very active during 1898. The disputes over the adoption of the Chicago scale for mining interfered with production, while shipments were restricted by low water, there being none during June, July and September, and November shipments were very small. In all the river districts trade was injuriously affected, not only by these causes but also by the continued active competition of the Kanawha region in West Virginia, for the coal business of the cities on the lower Ohio and Mississippi rivers. This continued exceedingly active, and the competing elements were increased by the shipments of Alabama coal from Greenville, Miss., to New Orleans. In one respect Pittsburg had an advantage in 1898, owing to the purchase by the government of the Monongahela improvements. The slackwater navigation is now free from Pittsburg to the headwaters of the river. The great point is that the mines in the pools along the Monongahela are now placed on equal terms with the West Virginia and Kentucky producers, so far as freights are concerned.

SHIPMENTS OF COAL FROM PITTSBURG BY THE OHIO RIVER. (IN SHORT TONS.)

Year.	Cincinnati.	Louisville.	Total.	Year.	Cincinnati.	Louisville.	Total.
	Tons.	Tons.	Tons.		Tons.	Tons.	Tons.
1883	1,261,320	2,258,480	3,519,800	1891	1,125,000	1,931,600	3,056,600
1884	985,240	1,232,040	2,217,280	1892	973,560	1,519,960	2,493,520
1885	1,303,600	1,693,360	2,996,960	1893	879,950	1,617,840	2,497,790
1886	1,329,160	1,537,406	2,866,566	1894	1,139,920	1,383,280	2,523,200
1887	830,800	1,438,920	2,269,720	1895	984,400	1,384,080	2,368,480
1888	2,053,560	2,340,520	4,394,080	1896	2,029,760	2,578,120	4,607,880
1889	1,214,400	1,515,800	2,730,200	1897	1,144,568	922,800	2,067,368
1890	1,304,640	2,042,160	3,346,800	1898	1,236,800	1,319,120	2,555,920

In the railroad districts a fair business was done through the year, notwithstanding the great competition which had to be met from the West Virginia and Ohio mines. The lake shipments of coal in 1898 were the largest on record, and Pittsburg secured a better share of them than was expected at the beginning of the year.

The accompanying table shows the shipment of coal by river from Pittsburg for 16 years; during that period the largest shipments were in 1896, and the next largest in 1888, while the lightest were in 1897, when there was a low stage of water almost all the year, and coal loaded in Pittsburg waited for months before a coal-boat rise in the river came.

Coal Mining Methods and their Economic Bearings.

By W. S. Gresley.

Survival of Antiquated Methods and Local Customs.—The geological and physical conditions in which coal beds exist and have to be mined largely determine plans or methods devised and developed for working them. Besides these conditions there are those relating to custom, and still others having reference to individuals, and perhaps even to prejudice. It is highly probable that many seams worked only on the room and pillar system in one mining region or country would in some other country be mined on the long wall plan, and *vice versâ*, the preference being due to local custom rather than to the physical conditions imposed, and progress is thus retarded. In the preparation of this article, therefore, the author has endeavored to treat the subject on the basis of custom, or that which is actually being done in various coal regions, and not so much that which in his judgment (and maybe that of others) would suggest itself as being the more rational method were it practicable to make the change; in other words, this paper is intended to be practical, not theoretical.

Evolutionary Process Apparent.—The mining of coal, like other industries or pursuits, is continually passing from one stage to another, undergoing improvement, notwithstanding temporarily imposed retrograde movements sometimes noticeable. One of these movements relates to an aspect pertaining to machine mining. This is naturally and necessarily so, because there is too much coal property developed (in the United States at all events); production becomes less profitable; mining conditions get more complicated; laws and regulations stand for more safety and less risk; new plants or large extensions of old ones represent vast amounts of money invested; a better technically educated class of mine officials is rapidly appearing; slovenly methods involving coal waste are being less tolerated. The combination of circumstances demands a minimum production cost, and that operators shall also insist upon a minimum loss of coal in mining.

In striving to obtain the former, beware lest the latter precaution is overlooked and a false result thus reached. In this connection machine mining now plays an important part. Probably the introduction of coal-cutting machines has given, is still giving, and will in the future give rise to a greater number of modifications and changes of methods of working coal than were ever tried previous to their adoption. Wherever machines can be used to advantage they are going to be used, and their successful use will depend a great deal upon adaptation of the mine workings, or the plan of mining to the machine. This subject will be dealt with further on.

Every Mining Method a Compromise.—However good or apparently satisfactory a method of working coal may be, it has some drawback—something to prevent its being an ideal or perfectly satisfactory method. So that, in seeking or planning a suitable method to follow, the one decided upon always represents a choice of evils—a sort of happy mean between the favorable features and those with disadvantageous aspects. So subtle are some of the conditions involved, and so uncertain are others, that to strike an average, in order to retain the least objectionable points, is no easy matter. There are, however, a few conditions, or broadly governing principles, to guide in making choice of fit methods, and they will be named later; but of course the main object sought is to extract the largest tonnage possible from the seam; to do this so as to ship the product in its best marketable condition, and in the safest and most economical way, all things considered.

Different Methods in General.—The thinner the seam worked, the cleaner it is worked, the less the waste of coal in mining. The reason is obvious: the thinner the coal the less there is of it per acre, and, other things being equal, the higher the mining cost; therefore, to make its working pay, all of the coal if possible must be taken out, and if this can be done by one operation, so much the better. The long wall system is the best and cheapest for thin seams. On the other hand the thicker the seam the more of it is wasted in working, the reasons being that it is cheaply mined, the yield is high, and there are no known methods of extracting the pillars except at great cost and risk to life and property, or else by mining these thick beds on some entirely different plan from that customary in Pennsylvania, a resort to which, owing to lack of practical acquaintance with radically different methods, is very rare. Now, getting coal in one country or another is performed by a multitude of modifications based upon or evolved out of two broad systems, viz.: (1) "Room and pillar," or taking out various proportions of the seam as the work proceeds outward or away from the mine openings or main avenues thereof, and the remainder, or a portion of it, when receding or working backward in the opposite direction; (2) "long wall," that is, taking all of the coal as you advance into the area to be mined, or work outward from the shafts, and carry the haulage roads and air courses between packs or walls formed in the exhausted ground or gob. Obviously these two systems admit of and have received much modification; for instance, room and pillar working lends itself to very variable dimensions for rooms, chambers, breasts or stalls, as the wider openings are variously termed, as well as of the ribs or pillars of solid coal separating them and the narrower excavations leading to and connecting them for one purpose or another. Long wall proper also is very frequently modified to suit peculiar conditions, until it becomes hard to tell whether the method practiced is a pillar or a long wall one. The details of forming pillars, etc., and of their removal or reduction, as also regarding the building of packs, gobbing, filling or stowing of exhausted spaces in order to maintain a proper working face or to aid in pillar drawing; of taking down roof coal, and so forth, are beyond the scope of the present paper; and yet as success so largely depends upon detail, *i.e.*, in doing the seemingly trifling or less conspicuous work underground properly or in accordance with well tested and approved custom, to neglect them may mean disaster or failure. Every colliery seems to possess some peculiar condition not to be ignored in the plan of getting the coal. Hence we find a diversity of details, and every departure from the established district method has its explanation in what may be called the balance struck between the advantages on one side and the disadvantages on the other. Therefore it is deemed advisable to present the information accompanying each illustration of a method of working by a tabularly arranged statement of these advantages and disadvantages and for the sake of easy reference and comparison.

Conditions Favorable to Room and Pillar System.—(1) Thick or high seams; (2) rock roofs; (3) absence of gobbing material; (4) surface to be left intact; (5) shallow mining; (6) seam much cut up by rolls, clay veins, horsebacks, etc.; (7) beds of very irregular thickness and pockety seams; (8) irregular working (suits rooms but not rib and pillar drawing); (9) soft coal.

Conditions Favorable to Long Wall System.—(1) Thin seams; (2) weak roofs; (3) heaving floors; (4) regular thickness; (5) freedom from rolls, horsebacks, etc.; (6) plenty of rubbish to gob with; (7) regular or steady work throughout the year; (8) seams composed of several kinds of coal, the separation and preparation of which for market is necessary; (9) maximum yield of lump or round coal; (10) where the surface may be disturbed or let down without much injury; (11) hard coal; (12) deep mining; (13) seams liable to spontaneous combustion (long wall retreating); (14) for coal in its very best condition.

As it rarely happens that all the conditions favorable to either the one or the other system are present in any one locality, choice of method will be indicated or determined by the majority of favorable conditions of greatest importance from an economic standpoint.

THE COAL-MINING MACHINE.

Conditions Favoring the Use of Machines.—(1) Hard, strong coal; (2) fairly thick seams and also seams that are very thin; (3) strong roof; level and hard floors; (4) seams free from faults, rolls, etc.; (5) for obtaining maximum proportion of large coal; (6) coal (provided it be properly cleaned) in best condition; (7) where no pick mining has to be resorted to (in other words, in situations where the surface must not be broken or let down and consequently a large part of the seam cannot be taken out); (8) when rapid development of the colliery is imperative; (9) in cases where the workings of adjacent mines are to be connected with utmost dispatch, say for ventilation, drainage or some other all-important object; (10) makes a more compact mine; concentrates work; (11) where miners are not plentiful; (12) where compressed air or electric machinery for other purposes has already been installed or has to be put in; (13) where "shooting off the solid" is unsatisfactory and there is no soft band in which the miner can "bear in" or undermine the coal; (14) where it is of greater importance to cheapen "mining" than to save coal; (15) fewer men employed.

Conditions Unfavorable to Undercutting Machines.—In general the opposite to those that favor their adoption, to which may be added: (1) If much pyrites or stones in the coal or in the bearing-in horizon; (2) in seams too hard and too full of slip-joints (*e. g.*, anthracite); (3) in seams too soft, and where lump coal is required (*e. g.*, the Connellsville, Pa., coking bed); (4) in mines so wet and gritty that the repairs and maintenance of machines are too expensive; (5) irregular and uncertain work (too many idle days); (6) where the roof or sides cannot be prevented from falling upon the machine, and the weight on the faces pinches and retards the cutting part of the machine while running.

Objections to Coal-cutting Machines.—Deep undercutting and wide places tend to produce more coal, and more lump coal, because less side-cutting shearing or nicking is needed, and less shooting is required. The larger the lumps loaded at face the less are the impurities in them sought, visible and extracted; therefore the dirtier the final product shipped. Less shooting calls for less vigilance in ventilating the working places. Heavy shot-firing in narrow work after machines racks and cracks the ribs and pillars, which, unless made larger to compensate for this shaking, are thereby seriously weakened; and the weaker the pillars the more coal wasted in working, and the higher the cost for timber. There is a growing tendency among operators to sacrifice good coal (in pillars, stumps and ribs) in order that the proportion of machine-cut or cheapest-mined coal may be maximized. This grievous practice of leaving coal because the machines (as at present in use) cannot cut it carries with it the serious evil or danger to be apprehended from explosive gas accumulating in the old chambers, and which cannot altogether be ventilated and may any day cave and force out the gas into the entries, etc. Caution is needed lest the mining machine become the master rather than the servant.

Comparative Merits of Different Machines.—The Harrison or puncher type of undercutting machine cannot compete with the breast or traveling long wall machine except under peculiar conditions, viz., in places where the roof needs much timbering close up to the face, or where the bearing-in is too stony to be cut by chains, wheels, rotated bars, etc. Under equally favorable conditions to both types, the breast machine performs 50 to 100% more work in a given time than the puncher. The breast machine makes least slack. In favor of the puncher it must be said that it will side-cut entries and rooms, may be used where the face is "on the weight," and is thus to a considerable extent capable of extracting rib coal and pillars. Its use is practically limited to compressed air to drive it, though electric punchers are not unknown. The puncher saves about 20% in mining cost, but cost of operating, repairs, maintenance, interest and other charges leave but a small margin in its favor. For long wall work punchers are too slow, and the self-propelling chain machines or horizontal cutting disk, and in some places the rotary bar machine, are in successful operation. In cutting in fireclay perhaps the chain machine leads; whereas in making the undercut in the coal, the disk or wheel is preferred. The original cost of an electric coal-cutting plant is about six-sevenths that of a compressed air equipment of like capacity with the machines working say within about one mile from the power house. Exceeding that distance the latter method rapidly falls behind. Air machines, if any can be used, are all right where locked safety lamps are used. Just how much cheaper coal can be mined and placed on cars at pit mouth where machines do the undercutting or save the blasting, depends, of course, upon circumstances, but unless every item of cost is brought into the account, the true state of things in favor of or adverse to machines cannot be revealed. Cost accounts, even when accurately and conscientiously kept monthly or yearly, showing wages, materials, etc., are likely to be misleading on this point. To illustrate: Unless the value of the coal as it lies in the ground be taken into the accounts prepared to show the results of operating as between pick and machine, an incomplete and therefore untrustworthy statement will appear. Now, since the use of machines in many places tends to greater loss of coal than obtains with pick mining, every ton lost under or due to machine work should be charged (at its present value) against the machine system. If machines are responsible for losing say 25% of the coal which would have been saved under the old method, say 10c. per ton apparently gained or saved by use of machines, may in reality be no saving at all, because the present value of the 25% of the coal seam being lost may exceed the wages, etc., saved in mining the other 75% of it, assuming that all is taken out; of course some loss is inevitable.

Future of Machine Mining.—As avoidable coal waste will not be tolerated for long, and pick mining will be too expensive to continue, the second working in room and pillar systems (*i.e.*, the extraction of the ribs and pillars) will have to be done by machines. Hence we are approaching the time when two types of machines will be used pretty generally in the same mine—one to do the entry work or in headings and narrow places, the other to take slices off the pillars formed by the first machine. This second machine then will be a long wall one. In this way we shall eventually see plans developed, designed and practiced in which the lesser portion of the seam will be cut with breast machines and the greater (the pillars) with the long wall types. I believe that this condition will ultimately obtain, because it will be the only practicable method of preventing serious loss of coal and yet producing it most economically. It cannot be applied universally, however, but only successfully where conditions are in its favor. There must of necessity always be some territory in a mine where the pick alone can accomplish the work. The long wall type of machine cuts more rapidly than any other type; hence it will be brought into requisition in pillar methods, for the reason that rapid cutting of them is necessary to their proper and economical removal. Moreover, the longwall type of machine being the fastest cutter, favors maximum concentration of workings, and this in turn implies minimum maintenance cost inside. It ought to save posts, as well as yield the very best quality of coal. To sum up: The best results with machines will be attained by a combination of the following advantages: Least narrow work; moderate amount of room cutting; most pillar cutting; least moving of machines from place to place; rapid clearing away of coal after machine; concentration of operations; best attention to posting, and the caving-in over the gob; least shooting; deepest undercutting; shearing where practicable; paying attention to cleavage or joints; best organization and vigilance; steadiest running (fewest idle days).

GENERAL OBSERVATIONS ON SPECIAL SUBJECTS.

Soft seams on heavy pitches should be worked by horizontal levels off inclined planes, not by chutes which smash up the coal.

The gob line in pillar drawing should be as straight as possible and as nearly parallel with the longer dimensions of the pillars as practicable, in order to prevent loss of coal and to minimize timber consumption.

Gob Entries.—Entries or narrow places driven in coals with weak floors and tops should be driven as wide as possible and the sides of roadways well built up with stone, etc., and back-filled with rubbish to prevent creep or to minimize the tendency to close up altogether from pressure.

At great depths, even in strong strata, roadways are with great difficulty and expense kept open unless a considerable width of well-filled gob is allowed on either side.

Bumps.—To minimize the occurrence of and damaging effects of "bumps" or sudden outbursts of coal at the working face (the cause of which is attributed to the pent-up gas pressure in the coal), the wider the face is worked and the slower it advances the better.

Pillars.—Under heavy cover, in room and pillar work, extra heavy pillars should be made between the gob or exhausted areas and the air-courses or traveling ways flanking the main haulage ways; otherwise the air-courses will be expensive to maintain. This precaution also offers improved facilities for extracting the main entry and flanking pillars when the time arrives to work them out.

Extracting Pillars and Ribs.—In order to secure the coal in these in its best condition, both as to physical strength and fuel constituents, and also in the most economical manner as pertaining to consumption of timber, use of track and other material, they cannot be drawn too soon after the chambers or rooms are worked their proper distance, nor (as a rule) too rapidly when once started; but failure or neglect to make a satisfactory job of them may be expected unless their removal be accomplished by working them in blocks or series, a given number at a time; all being started simultaneously, drawn at a uniform speed, and finished practically together.

In pitching seams, where squeeze is very serious, the levels, gangways, tunnels and chutes may with advantage often be made in some convenient stratum away from the coal seam, the latter being tapped at convenient places along these gangways or levels.

Second working, or pillaring, should always follow up the first working or the driving of rooms as closely as possible. This practice saves coal; improves its quality; saves timber and other supplies; saves maintenance; improves the ventilation; reduces pit room, and is safer.

Surface land is usually less damaged in the long run when all of the seam is worked clear out beneath it than by leaving pillars or ribs to protect it. This is because the pillars in course of time give way unevenly and at different times.

Weight.—The effect of "weight," creep, or squeeze in pillar working is more serious in mines located in valleys or adjacent to hills than in even country. Hence pillars should be made larger in those mines than usual or level country practice requires.

The author, representing the coal-mining industry in this connection, is greatly indebted to those mining engineers and managers of collieries who have kindly rendered him assistance by supplying data for the compilation of the several illustrations and accompanying descriptive matter herein.

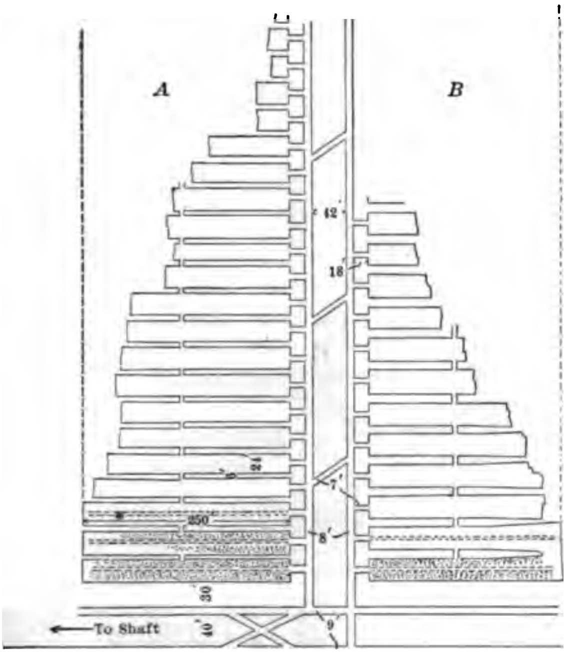

Fig. 1.

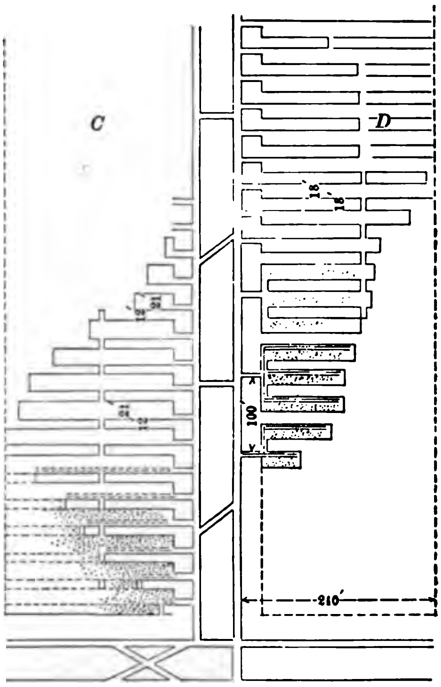

Fig. 2.

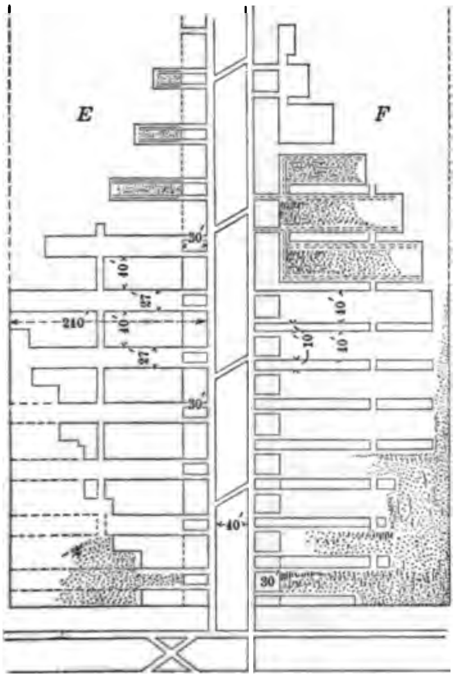

Fig. 3.

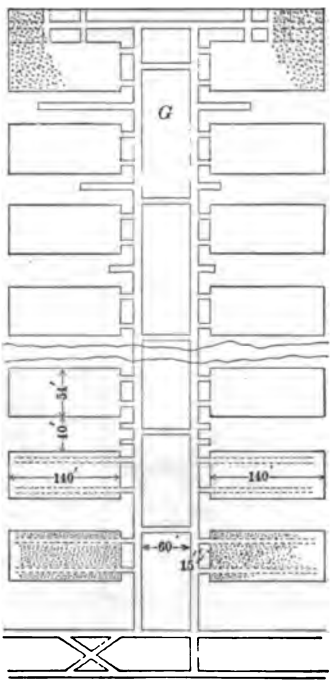

Fig. 4.

Room and Pillar Methods.

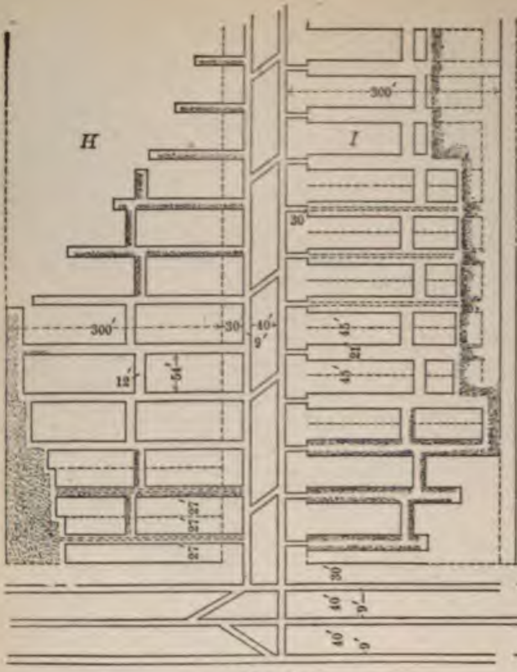

FIG. 5.

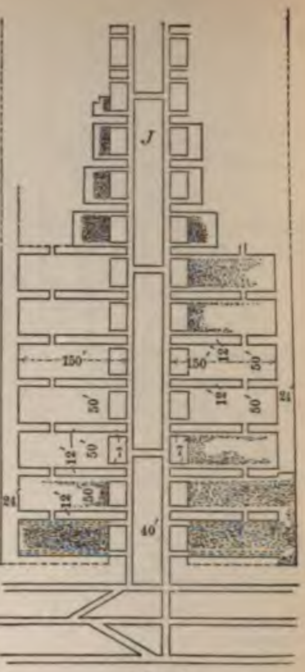

FIG. 6.

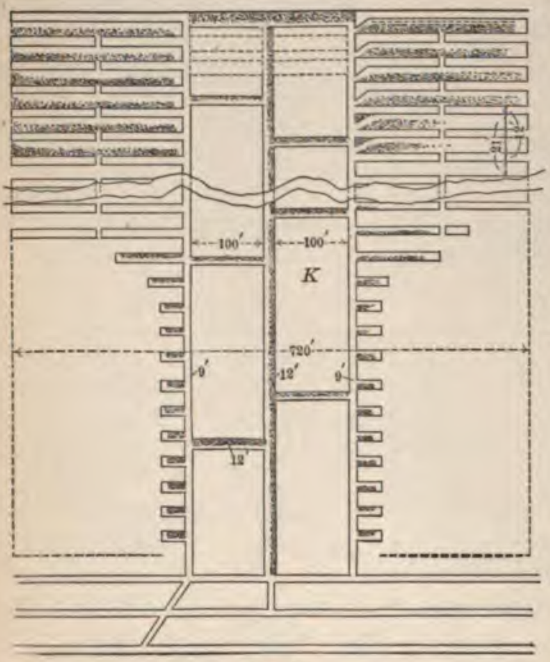

FIG. 7.

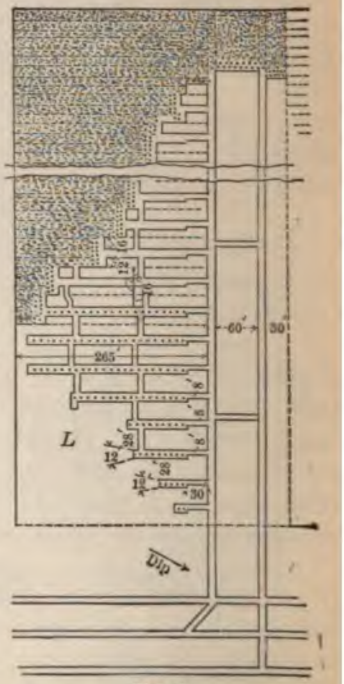

FIG. 8.

ROOM AND PILLAR METHODS.

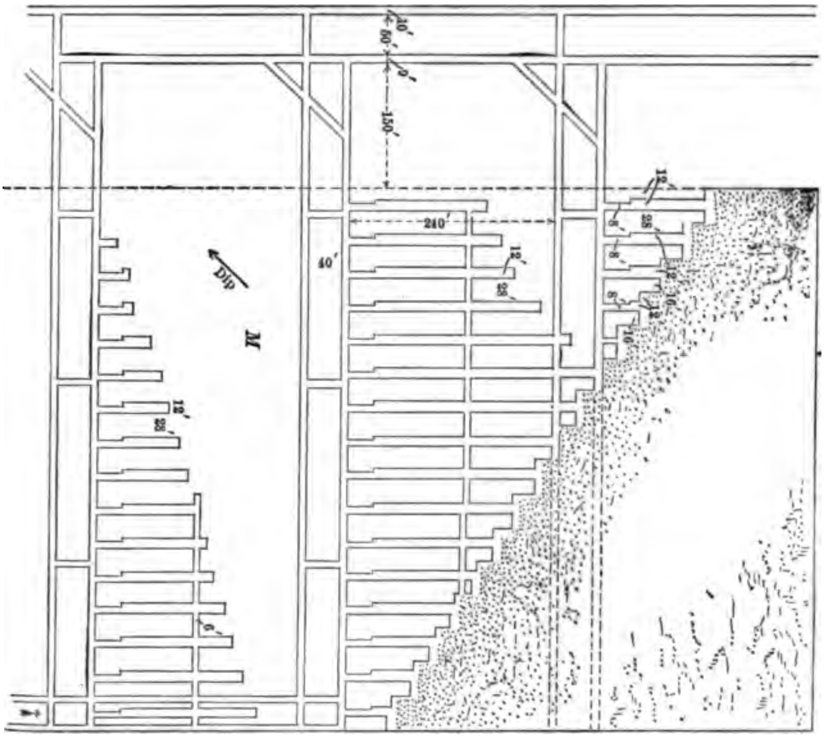

FIG. 9.

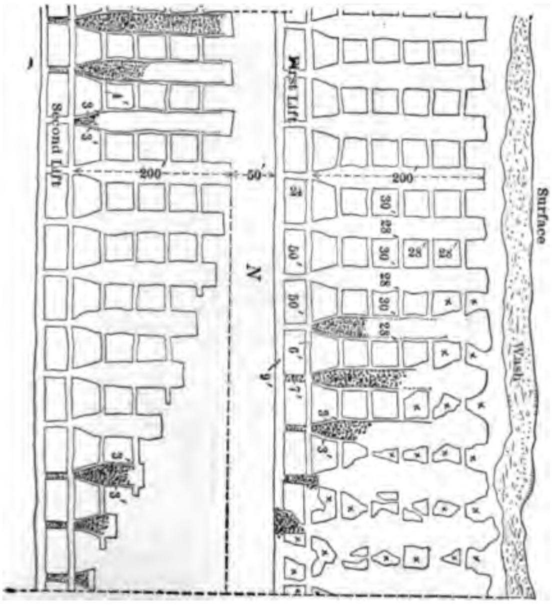

FIG. 10.

ROOM AND PILLAR METHODS.

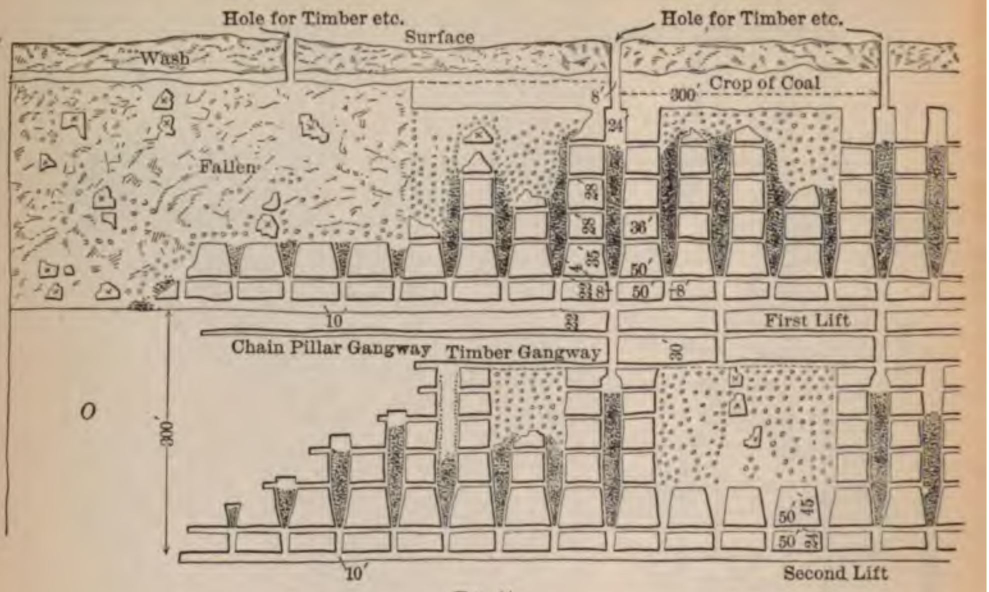

Fig. 11.

Room and Pillar Methods.

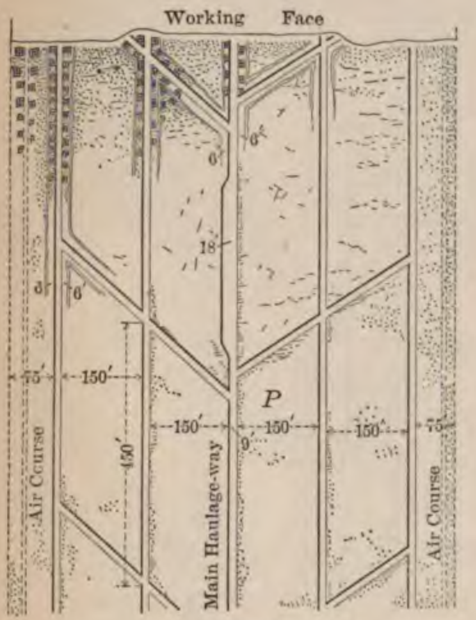

Fig. 12.

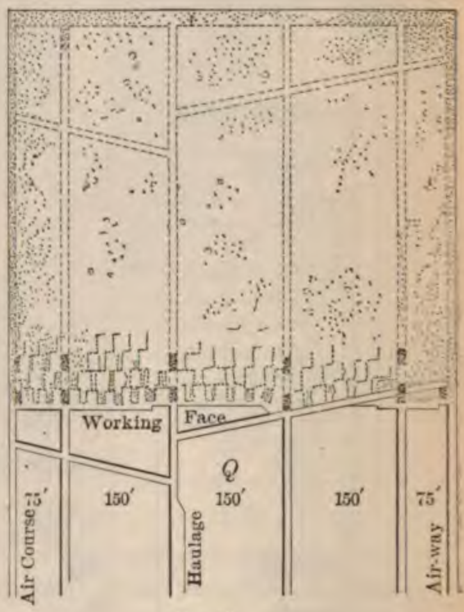

Fig. 13.

Long Wall Methods.

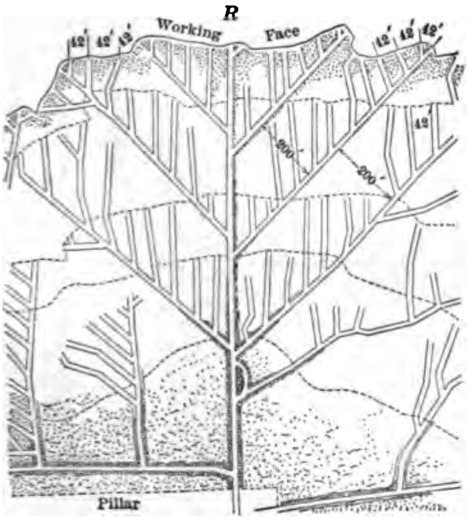

Fig. 14.

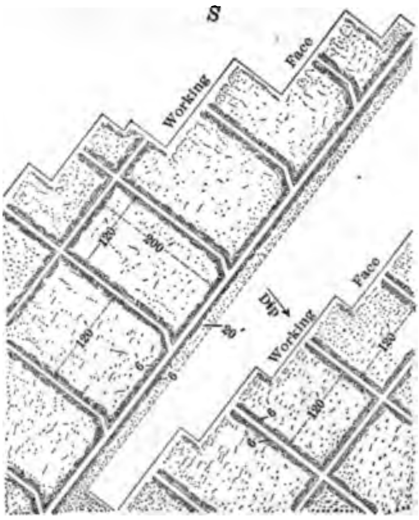

Fig. 15.

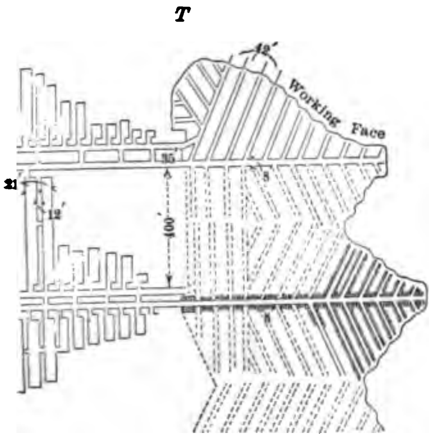

Fig. 16.

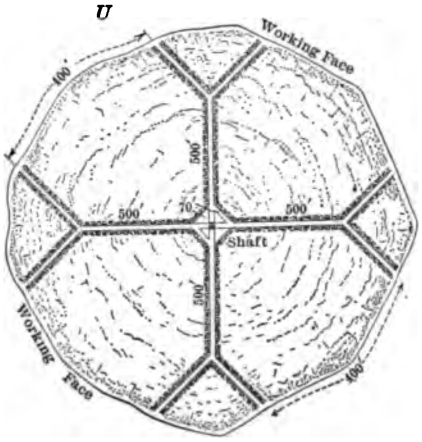

Fig. 17.

Long Wall Methods.

METHODS OF WORKING COAL MINES.

Figures and Cases.	Conditions.	General Description of Methods.	Details of Methods.	Economic Bearings.		Remarks.
				Advantages.	Disadvantages.	
1 A....	Thickness coal, 4¼ to 5¼ ft. capped by 9 to 12 in. fire clay. Height workings 5 to 6 ft. Seam nearly flat. Coal hard or strong bituminous. Cleat vertical, marked and regular. Roof, strong bands of coal and shale 5 ft.; rock above. Cover, 100 to 250 ft. Floor hard. Horsebacks, rolls, and clay veins locally. Refuse, a few thin bands of shale and the top fire clay. Depth, 75 to 500 ft. Surface, mostly farm land. Drainage, natural in drift mines, or pumped and siphoned out of swamps. Mining rate paid on lump coal going over a 1¼-in. screen of 60 sq. ft. area. Unskilled labor plentiful; good miners scarce. To leave no coal in the ground excepting to protect buildings, water-courses, springs, etc.	*Room and Pillar.* Double entry system. Main entries driven on the face or to suit shape of territory or dip of field. Productive entries driven on the butts (end on). Distance 1,000 to 2,000 ft. according to conditions. Rooms worked on the face, or at right angles off the butt entries. In flat areas rooms worked on each side of these entries, dipping on the rise side only. Ventilation by doors, brattices, etc., being passed up one entry, through the break-throughs separating rooms, then down the range of working places on the opposite or counter entry, to be carried to the main return aircourse, or cross over into an adjacent set of rooms, etc., on next pair of butts. Mine cars, from 1½ to 2 tons capacity.	Tunnels and entries 7 to 9 ft. wide, 35 to 60 ft. apart. Flanking pillars 30 to 60 ft. thick. Pairs of parallel productive entries are set away on each side of tunnels, 550 to 600 ft. apart, with 40 to 60 ft. pillars between entries. On these entries rooms turned every 33 ft. Rooms are started 7 ft. wide and driven up 20 to 30 ft. before being widened to 27 ft. Rooms worked up about 240 ft. and widened toward back, sometimes cutting through into adjoining rooms. Car track runs up center of room, two rows posts and gob on either side. Cut-throughs for ventilation, in ribs as required, 6 ft. wide. Usually they are left. On reaching boundary of the section, as much of the pillar coal is robbed as conditions of the place will allow; usually a squeeze overtakes the section before the rooms are well completed. Two men work together in each room. Bearing-in usually done in bottom coal.	Quickly opened mine.	Very wasteful of coal; relatively high deadwork costs; risk of losing much coal, owing to squeezes on entry pillars caused by not taking out ribs; high timber consumption; small yield per acre; small royalty for lessor; much shooting of coal.	Little or nothing to recommend either method A or B, Fig. 1. Entry driving and other narrow work costs about equal for each; increase per ton as acreage yield decreases. Yield, about 5,000 tons per acre. In Figs. 1 to 11 inclusive similar panels or sections of butt entries and rooms are supposed to be on the opposite side of the main entries.
1 B....	Same as A.	Same (see above).	Tunnels, entries and narrow work about same as in method A, Fig. 1. Rooms 27 ft., ribs 12 ft., carried up to within short distance of limit, when the rooms break through into one another. Mining details as above.	Less squeezes, thrusts, or creeps than in A, Fig. 1. Fairly well adapted to breast undercutters.	High deadwork cost for yield per acre or ton.	
2 C....	Same; but thickness of cover or hill over the mine may run up to 450 ft.	Same.	Rooms 21 ft., ribs 12 ft. Track next the rib, which is drawn as soon as room is driven to limit. One miner in each room or its rib. Bearing-in between two	Clean working (minimum coal waste) when ribs and pillars are quickly and skillfully drawn, and stumps (coal between the	Crushed coal and damaged pillars, unless ribs are taken out clean; occupies too much pit room; too slow.	Except for high-priced mining, a fairly economical method. Yield, say 6,500 tons per acre. Cost of production about

			thin slates. When mined by punching machines, about a third ribs and pillars can be so extracted.	room necks) are made large enough at first; 70 to 75% of seam may be mined by machines.		the same, whether pick-mined or puncher-machine mined. Actual mining (cutting, shooting, loading, posting, etc.), costs 20% less, but is offset by cost of plant, maintenance, etc.
2 D....	Same as A and C.	Same.	Rooms 18 ft., ribs 15 ft. One-third number of room necks driven through into entry. Rooms worked on retreating or withdrawing system, *i.e.*, from the inside or back end of the section toward main entry. Pick mining, 1 man in each room, 2 on each rib. Room widened on right or left, according to direction ribs are to be taken out. Compare plan C.	Reduces coal waste to minimum; economical in posts; strong entry pillars and stumps; squeezes impossible.	Requires large pit room; not well adapted to coal cutters, on account of narrowness of rooms; high mining cost.	Yield, 7,500 tons (run of mine) per acre.
3 E....	Same.	Same.	Rooms 25 ft. wide; pillars 44 ft. thick. Each room has two necks; track alongside each pillar; the refuse piled between the tracks and around the posts. Each room works back one-half the pillars on either side; two men work in each room and four on each pillar.	But little coal need be lost; gives fair yield of entry pillar and stump coal.	Not well suited to machine cutters; high cost for cut-throughs; timber cost high, unless gob in rooms built up tightly to roof.	Worked on limited scale; nothing particular to recommend this method.
3 F....	Same.	Same.	Rooms 36 ft. wide; ribs 12 ft. All narrow work and rooms driven by machines. Ribs drawn by pick if at all. Rooms on approaching their limit under light cover, and with solid coal in front of them may be cut over into one another by machines 30 or 40 ft. from limit, and a long face forked thence to limit; 4 loaders in each room.	Cheaply mined coal, because nearly all is machine cut; suits a good roof. Produces high percentage lump coal; 5 to 10c. a ton cheaper than pick mining, including charges for machinery, interest, etc.	Entry pillars and room-neck stumps lost (crushed and unworkable), unless necks are reduced in number to minimum and made of ample length (see upper end of section or panel); ribs crush and cannot be extracted except with great rapidity and skill; not suited to heavy cover.	Decided tendency with cutting machines to leave in the ground all coal that they will not cut. This loss ranges from 5 to 35%. Solid pillars must be left here and there in the mine to stop squeezes.
4 G....	Same.	Same.	Rooms 60 ft. wide; pillars 40 ft. Rooms worked up to limit of the panel or pair of butts; then the pillars attacked by driving a machine up room center and working ribs on each side backwards toward entry.	Claim to get 85% of all the seam by breast machines or "punchers;" well adapted to light cover and sound roof. Large percentage of lump coal.	Necessitates an excellent top or roof; therefore limited in application.	Hardly past experimental stage.

METHOD OF WORKING COAL MINES.—*Continued.*

Figs. and Cases.	Conditions.	General Description of Methods.	Details of Methods.	Economic Bearings.		Remarks.
				Advantages.	Disadvantages.	
5 H....	Same.	Same.	Rooms 12 ft. wide, 300 to 500 ft. long; pillars 14 ft. Entries, crosscuts, necks, rooms and break-throughs driven by machines. Refuse piled on one side of room.	Very little coal lost; entry pillars and stumps worked out clean; 30% less entry work (say $25 per acre cheaper).	Yardage to pay for narrowness of room; room refuse to handle when pillars are worked; does not suit breast machines; not suited to irregular working.	
5 I.....	Same.	Same.	Rooms 21 ft., pillars 40 to 50 ft. An all-machine method, *i.e.*, narrow-work and rooms cut with breast machines and pillars with long wall machines. Say five pillars worked back in line toward entry, being closely followed by another series.	All the coal in the seam mined by machine; concentrates work to maximum; produces maximum of lump coal; saves 30% of the usual narrow work needed with foregoing methods.	Rooms must be comparatively narrow in order to work back very wide pillars; refuse in rooms must be handled to pass long-wall machine from one pillar to next. Considerable skill needed in pillar working.	Not past experimental stage.
6 J.....	Same.	Same.	Rooms about 50×150 ft.; ribs about 12 ft. Ribs extracted by pick. Rooms strongly posted on the gob side of tracks; top, if tender, allowed to fall over the gob.	With rib-drawing properly done, gives satisfaction; makes large percentage of machine-coal with minimum wastage of coal; compact mine workings.	Requires more entry and narrow-work than usual in foregoing plans.	Modified to suit peculiar conditions.
7 K....	Same.	Three-entry system, room and pillar; general description above.	Middle entry and crosscuts 12 ft. wide; gob left in; side entries, 8 to 9 ft., with gob hauled out. Rooms 21 ft., ribs 12 ft. Worked on retreating method, but room necks cut right and left as butts are driven. Entry pillars drawn by removing parallel slices as ribs and rooms are exhausted.	Clean working where no pillars need be left; well adapted to hand mining; costs $30 or so less per acre for narrow work than with most rib and pillar methods in flat seams.	Slow to open up; slate to handle; only fairly well suited to machines.	
8 L....	Thickness of coal, 8 ft. Roof, coal, and shale, 5 ft. Floor, hard shale. Seam, level or slightly dipping. Depth, 400 to	Two pairs of main entries, about 10 ft. wide each, driven on the butts from shaft. These entries have 150-ft. pillar between them; a 60-ft. pillar	100 ft. of coal left between each butt entry. Rooms opened 8 ft. wide and driven 30 ft. that size, then widened to 12 ft. and one row of posts set. To draw the	Yield, 95% of the thickness worked in the productive panels, such as L, Fig. 8; can use 2¼-ton mine cars.	Too much coal standing in pillars, to be worked out (?) at the last; too much pit room—open mine roads to maintain.	Shape of property renders modifications of method necessary. Yield, about 11,000 tons per acre

	700 ft. Coal, tender. No partings or bands of refuse. Coal more valuable than the surface. All taken out except pillars under valuable buildings, rivers, etc. Mining price is based on run of mine. Screens not used. Mines gassy. Coal not hard enough to pay to cut with machines. Cleavage, vertical in two directions.	separates the two entries of each pair; and these are flanked by pillars about 120 ft. wide. On each side of these tunnels three parallel face entries are driven at right angles every 1,300 ft. and go to the limits of mine. These face entries have pillars 120 ft. thick between them. Off these entries, at every 450 ft., are driven pairs of butt, or productive entries, about 1,000 ft. long, off which again the rooms are turned, commencing at inner end and working the coal on retreating plan. Exhaustion of a block of productive entries leaves the gob surrounded by solid pillars 350 to 450 ft. in width. Ultimately these are attacked at inside ends and cleaned out up to main tunnel pillars.	ribs 12 feet of coal are taken off one side the room and 16 ft. off opposite side. As little "stopping" of the faces in rib-drawing should be allowed as possible, in order to save timber and prevent loss of coal. One man in a room, 2 on the two portions of flanking ribs. No shooting required. Roof of working places consists of a few inches coal, left up to keep fireclay over it from falling. In dipping coal, rooms only worked on the rise side of butts, when their length is increased from about 70 yards to 120 or so.			
9 M....	Same as L.	Drive main entries to near extremity of field; then lay off pairs of face entries about 250 ft. apart and drive them to the limits, say not over 2,000 ft. in length. Then commence to work rooms on retreating plan. These, as well as the ribs, are worked on the butts. Fresh air goes in through headings on lower right and circulates round through the workings to left.	Open, start or work only such number of rooms and ribs as requirements of mine call for. Start ribs as soon as rooms are driven allotted distance; keep the gob line of stepped faces running diagonally through the various panels, as straight as possible, to minimize weight on faces.	Makes more convenient haul than method L; practically no coal wasted; improved ventilation over L; economy in posts, etc.; concentration maximized; safer than L.	More narrow-work, more entries to drive and maintain before a large output can be had.	Superiority of plan M has yet to be more fully demonstrated. Yield, about 12,000 tons per acre
10 N....	Anthracite, 4 to 6 ft. thick. Roof, shale. Dip, 50° to 75°. Depth, 50 to 300 ft. Surface less valuable than coal. All coal is blasted, unless loosened by joints and free to "run" by gravity down chutes or breasts. Mining costs paid on yardages; timbering paid for by set; buggying, at per mine car. Much water. Some C_2H_4. Cleavage very irregular.	Lift method, or working to the rise; levels or gangways driven about 300 ft. apart commencing near surface and going deeper into the basin as the coal becomes exhausted above. Both shafts and slopes used for hoisting, pumping, ventilating, manways, timber, etc. Chambers and breasts opened narrow off upper side gangways and widened upwards; worked on full pitch, with strong, square pillars until within about 100 ft. of coal	Gangways 10 to 12 ft. wide in the bottom and 7 ft. at top; 30 ft. from the gangway and up the pitch a heading or monkey gangway (the airway) driven parallel with it. At every 40 to 60 ft. along gangways chutes are opened 10 ft. wide. Above the heading breasts opened out gradually until 24 to 30 ft. wide, as conditions of top, etc., will allow. Two plank manways, one on either side of breast; these serve for traveling ways,	Satisfactory as to cost, yield, etc., in prosperous times, and the thicker the seam.	40 to 50% of seam left on pillars or parts thereof; high royalty to pay, because low yield; high cost for gangways, airways, chutes, and all narrow work, owing to bad yield; shortened life of colliery; very high total cost of production.	

Figs. and Cases.	Conditions.	General Description of Methods.	Details of Methods.	Economic Bearings.		Remarks.
				Advantages.	Disadvantages.	
		crop, or of next gangway above. When the range of rooms has reached limit, pillars are partially removed (see × × ×, etc., fig. 10) and lift abandoned. Fig. 10 shows portions of first and second lift on one side of slope or shaft, which is supposed to be on the left. Pillar-robbing going on in the first lift, chutes and breasts being opened and work on the second lift.	ventilation and conveyance of surplus coal mined in breast. Breast kept full of loose coal until finished, coal being drawn off as needed at the gangway below. Timber stoppings, called "batteries," regulate flow of coal out of breast. N indicates "chain pillar" separating lifts, and cut through occasionally for manways, air, timber, refuse, etc., but seldom robbed or wholly removed. Nearly all gangways and headings timbered with sets. Two miners work in each breast.			
11 O...	Same as N.	Similar to N, so far as opening and developing on gangway or lift method go, but about every fifth or sixth breast on first lift is driven through narrow to surface, for timber, etc., to save labor of carrying same up the chutes and manways. Lower lifts similarly connected with upper, and surface, for similar purpose. Drawing of pillars from top downwards; protection for men by ample posting. Rock debris from upper lifts or surface is dumped into areas cleared of coal; which with rock roof falls, caused by caving, practically fills spaces, so that removal of adjoining pillars may be safe. Fragments of pillars lost (shown by × × ×, etc., fig. 11), main slope or shaft, supposed to be on right side of plan.	Very similar to N; every fifth or sixth breast is extended to the surface. If top or floor of breasts is rotten, posts are set as breasts go up. Pillar-drawing begins at top of lift, next to chain pillar, or a small gangway about 30 ft. below main gangway. As removal of pillars proceeds in descending order, posts are set in gob. When area of gob or post-supported roof is too large to stand open, top caves, and with rock and debris dumped from upper gangway or through chutes from surface worked-out area settles, and adjacent pillars saved from serious squeeze. To work away the chain pillars, coal has to be removed by "buggying" it to tops of breasts prepared to carry it to lower gangways. To supply timber to gangway and pillaring a small gangway is run along upper edge of gob.	Yields 90 to 95% of the seam; reduces royalty per ton; reduces labor in timbering, etc.; improves product; helps ventilation; provides safe retreat should anything happen in lower workings. Pillar coal safely and very cheaply taken, because done in panels or separate areas of governable size; comparatively low or inside or production cost, because of splendid yield per acre or unit area worked.	Costs more than N for labor and timber on pillar-drawing by about 3c. a ton prepared coal.	Much superior to plan N, Fig. 10, as demonstrated by over 3 years' practice.

12 P....	Bituminous coal, 12 ft. Upper 7 ft. worked; lower portion inferior. Seam consists of several kinds coal, hard, soft, hard and soft mixed, bony layer, etc. Depth, 800 to 1,050 ft. Roof, shale and coal, then strong shale, etc. Floor, fire clay. Cleavage, vertical. Liable to spontaneous combustion, but not much pyrites. Water in coal quite salt. Dip, 5%; cars travel working faces. All coal taken out except beneath important buildings.	*Long Wall.* On starting to open up workings at the shaft pillar, all the coal (*i.e.* 7 ft. of it) is taken out on one or more sides and face extended laterally as far as hoisting capacity calls for. Roadways laid off through gob about 150 ft. apart. Roof taken down to make height as top subsides following removal of coal. Sides of entries maintained by wooden chocks or cribs filled with debris. Walls of puddled clay along the gob side of cribs to keep the air from gob or prevent fires. Coal face undercut by pick, wedged or shot down and loaded out to nearest roadhead. Return air leaves working faces through holes in the bottom coal, and passes along small headings in that part of coal and beneath gob.	For carrying on working face reached by each entry, 7 to 10 men employed; of these some work at night undercutting, taking down roof in roadways, cleaning up, posting, cribbing, etc.; followed by breakers-out and loaders. Undercutting 6 ft. and face held up by stout sprags set as braces against coal to roof and floor. Slack and debris thrown back in heaps or ridges in gob. With or following each slice coal removed from advancing face, back row of posts drawn and the top allowed to cave. Car track moved up every time face advanced. To avoid breakage and loss of timber and to help roof settle evenly and gradually upon gob, all posts are set upon a few inches of dirt or slack. Clay walls are built about 10 in. thick, but squeezed to 20 when roof has fully settled.	Coal is quickly opened up; no entries to drive outside shaft or other pillars, but this is offset by the cost of maintaining the roads, gobs, fighting gob fires, etc.	Expenses, danger, and time lost attending gob-fires; coal lost owing to fires; high cost timber and supplies; high temperature; difficult ventilating faces, especially as distance increases; expense driving air courses in bottom coal; very expensive in partings or turnouts, making branches and maintaining these wide places; does not prove the coal field in advance of the workings; too much slack and nut gobbed to help fill up behind.	Gob fires usually start on one side of a roadway or at edge or side of pillar, though sometimes in the bottom coal near an air course. Upper left of Fig. 12 shows cogging (chocks, etc.) and puddle walls as commonly used in this mode of working.
13 Q....	Same as P.	Mine blocked off, as circumstances and conditions require, into large panels or districts of 50 to 75 acres, which are developed and mined out in more or less regular order. Entries (headings and air-courses) driven to back end of each panel and coal worked back toward and right up to shafts or main haulage-way pillar in one or more nearby straight longwall faces. Refuse cast back into gob and helps carry roof, otherwise temporarily steadied by rude piles inferior lump coal (Fig. 13).	No airways or other headings driven in bottom of seam as in P. All gob left behind and cut off from air current; hence no spontaneous combustion so long as face does not stand idle too long. Entries 7 ft. wide, 6 ft. high. When more than say 7 ft. of seam taken out, seam is treated in two benches, lower one being worked about 10 ft. in advance of upper, which is dropped upon debris and slack made by the first.	Far better results as to yield and production cost than with P; spontaneous combustion almost unknown; improved ventilation; better haulage; safer; faults and bad coal proved ahead of working, therefore most economically dealt with; ability to work full thickness of seam where quality allows.	Greater the thickness of seam marketable the more of the coal wasted; the less coal left for protection of the miners, greater the danger and higher the cost for timbering.	
14 R...	Coal, 3½ ft. Nearly flat. Floor, fire clay. Top, soft shale, thick. Depth, 100 to 480 ft. All coal	To leave coal around the shaft for protection of same and of buildings at surface. Seam then opened out all round the	Entries in shaft pillars wide enough for two tracks and passing room, and heavily timbered, or better still main-	Working-face, being reached by a road every 42 ft., seldom caves; gives every 2 to 3 miners a separate	Multiplicity and length of roadways and track and expensive to maintain. Prevents tracks along	Where coal is not over 150 ft. deep, shafts about half-mile apart. Where 300 to 500 ft. deep, about

Figs. and Cases.	Conditions.	General Description of Methods.	Details of Methods.	Economic Bearings.		Remarks.
				Advantages.	Disadvantages.	
	taken out. Workings wet where shallow, dry below 300 ft. Coal strong. No well-defined cleavage. Occasional rolls or horsebacks in top of seam. Lump coal an object. Spontaneous combustion not uncommon. Much "sulphur" in coal, in shape of binders and balls.	pillar and worked outwards in a kind of circle. Roadways or rooms laid off every 42 ft. apart. Main entries for haulage and ventilation are set away N., S., E., and W., with branches right and left off at suitable distances. The rooms or roadways are cut off by new branches about every 300 ft. Air current splits at bottom of shaft, passes in sections around workings, then returns to upcast opening. Where floor not too hard, undercutting machines used with advantage.	tained by masonry sidewalls and steel I-beams placed transversely upon them and intervals filled with brickwork or plated or heavily timbered with oak, etc. Through gob main entries are 8 ft. wide, 5½ ft. high, timbered where necessary. Stout timber cogs or cribs are set beneath the shale roof at all angles or points where branches are put in. To give height on roads 3 to 6 ft. shale are taken down, broken material built up in stout wide walls on either side and close up to coal face. These packs, as they are termed, are 6 ft. thick, 4 ft. high, well backed on gob side with clay, etc., undermined below coal. Two miners (3 in some places) every 42 ft. of face. Track laid to each roadhead.	loading place; provides large daily output from a thin seam; enables 3,200-lb. cars to be used; suits a bad top, which can be posted as conditions demand; makes 82% lump coal over ⅞-in. seam; requires no shooting to break out the coal after face is undercut.	face, though buggying (basketing on greased rails) sometimes resorted to, to save dead work. Much rock, shale, and rubbish to hoist and get rid of, because too costly to shovel into gob when no room for it elsewhere underground. Not easy to ventilate, because so many roads, irregularities, and tight places along face. Not always room for rubbish made in the working places, because so many packs. Takes a roadman to every 10 or 12 roadways.	1 mile apart, or rather more in some cases. Miner's tonnage contract includes undercutting, breaking out, cleaning coal of impurities, loading, posting, spragging coal, brushing 2 ft. of roof on entry, pack-building, and laying of track. Where machines used, cutting is at a per ton price, and loading, etc., at another per ton rate. Machine mining increases proportion of lump coal. Conditions favoring machines largely dependent upon nature and management of roof and road packs; also upon facilities for rapid removal undercut coal, so face can be cut again before top caves, floor heaves, etc.
15 S....	Splint coal, 5 ft. One or two thin bands shale in coal. Top, strong shale. Bottom, strong fire clay. Dip, 5%. Depth, 1,000 ft. All taken except under valuable buidings. Maximum lump needed. Mine dry. Cleavage, regular, vertical.	Main headings and levels driven in solid coal, protected by pillars between and flanking. Off these headings the working levels, districts or "flats" are set away; all the coal is removed step fashion in a level course and to the rise. Few gob roads, because cars travel along working faces. Each stall about 120 ft. long, say 60 ft. each side roadhead. Some managers prefer to work without steps, keeping faces in unbroken line.	Face undercut in the fire clay floor or in some soft band in coal, preferably near floor. On removal of sprags the weight brings coal down or it is barred out or loosened by light shooting. Top shot down in roadways at face for height, and the shale built into packs. About 8 miners work a stall. Coal gets easiest on face, but makes most lump on the "end" or "butts." Regularity of work and skill in management of face and gob required to make system success.	Excellent yield per unit area; produces coal in about best condition; not difficult to ventilate; minimizing cost, and dead-work cost; economical in timber; min. tracks to use and maintain; surface subsides evenly; suited to machines when "noses" or steps removed; plenty of room.	Makes angles or "noses" along face or gob line, which tend to crush coal at those points. Some trouble in keeping air courses open between stalls, due to squeeze on edge of coal, and lack of enough gobbing material there.	Often called "Nottingham," sometimes "Barry" system. In some places timbering in the stalls done by special timbermen, not by miners. All roadways beyond the main pillars should be made in gob, otherwise very costly to maintain. Leading places on each flat being harder to work and having to put in the branches paid extra for.

16 T...	Coal, 4½ ft. Slate top; hard floor; nearly flat. Depth, 150 ft. Much water. Very strong coal. Cleavage, fairly regular and vertical. To take out all the coal. To make most lump possible. Coal hard to free of impurities.	Illustrates case where room and rib work was changed to longwall in same mine; change because former method very wasteful of coal, very expensive in powder, very full of sulphur balls, and pit wet. Room and rib method used very similar to Fig. 2, C; longwall which it gave place to practically that of Fig. 14, R.	Butt entries driven forward; coal opened out on both sides until the faces meet; 2½ to 3 ft. of top blasted down for head room and usual packs built along sides to maintain roadways. Stalls 48 ft. wide. Coal too hard to undercut, so shot off the solid. Packs 9 ft. wide on main entries, 8 ft. wide in branches.	Long wall plan consumed one keg powder per 37 tons produced; less labor by 25% in long work; coal better cleaned; water easily handled; improved yield, 30%; cars better loaded, by 25%; decreased cost haul, by 20%; vastly improved ventilation; reduced tracklaying. All told, saved 4c. a ton.	The old method produced 22 tons per keg. Miners, not being skilled in long wall work, full benefit of the system was not secured.	
17 U...	Coal, 25 in. Floor, fire clay. Top, black shale 12 in over it, 15 ft. limestone. Coal field, nearly level. Nearly dry mine. Depth, 150 to 200 ft. All coal removed.	From bottom of shaft roadways set away in four directions. Shaft pillar formed and working face carried on in rude circle. Top being very strong, very few roads needed, the cars taken along faces on movable tracks. Maximum length face between each roadhead about 500 ft. Entries 8½ ft. wide; track 17½ in., 8 lb. per yard. Cars hold 1,000 to 1,200 lb.	Coal undercut by pick or machine to a depth equal to height (more with machines). The "weight" breaks off coal when sprags or wood blocks are knocked from under. Packs on each side entries built with shot down limestone and blocks of black slate from over coal. Ventilation easily produced; excellent along face. Strong top over black shale makes plenty of area along face possible; very few posts needed. Undermined debris and impurities in seam, with slate taken down everywhere for safety, give plenty gob material.	Secures all the coal in seam; excellent opportunity to use machines to best advantage, because one continuous face to cut along; hardly any nut and slack made, nearly the whole of the seam coming out in large lumps; deadwork very cheap.		Machines average 50 lineal ft. face cut per hour. An average shift's run=450 ft. face cut. A machine's crew consists of 5 men; 3 machines produce 280 tons per 10-hr. shift. Undercutting paid at per hour. It equals about 12½c. per ton. Loading after machines at 3-5 pick mining price =45c. a ton. To each roadway 10 to 12 loaders after machines.

NOTES ON THE MANUFACTURE OF COKE IN THE UNITED STATES.

The total production of coke in the United States in 1898 was 15,897,797 short tons, valued at $30,505,563 at the point of manufacture, against 12,768,891 tons, valued at $23,203,920 in 1897—an increase of 9·7% in tonnage. The production of coke in Pennsylvania, West Virginia and Alabama was large in consequence of the activity of the iron trade in those and other States. In the last-named State the output reached 1,541,250 tons. In Kentucky there was a slight decrease. New plants and additions have been installed in the Connellsville region, Pennsylvania. A development in the coke trade is in the increasing number of by-product ovens, and in the tendency which is beginning to be apparent for iron-making plants to erect ovens of this class and make their own coke. Plants of this description have been erected at Ensley, in Alabama, and Benwood, in West Virginia, while a very large plant of the same kind will be erected at the Lorain works of the Federal Steel Co. to supply fuel for the new blast furnaces there. An important movement of this description is also found in the very extensive works which the New England Gas and Coke Co. has erected at Revere, near Boston, which are intended to supply gas for heating and power purposes, as well as for illuminating in Boston and other New England cities. These works use coal from the mines of the Dominion Coal Co. in Cape Breton, Nova Scotia.

The beehive oven coke of the New River district, W. Va., is usually made from slack, which naturally carries an undue proportion of impurities, but in spite of this runs only about 6% ash. A shipment made in 1898 for special purposes, coked from slack which had been screened to separate slate, etc., showed only 4·23% ash. This result is suggestive. The sulphur, while variable, is usually quite low, running from 0·5 to 0·6%, and often less. As this coke has been used only to a limited extent for the manufacture of Bessemer steel, little attention has been paid to the phosphorus content, but recent analyses show exceptionally small amounts. In another direction the operating results are less favorable; a coal of average composition from this district should yield about 75% coke, but in practice the beehive ovens seldom give 60%, and some run below 55%.

R. M. Atwater, discussing the question of making coke from Ohio coals, explains the difficulties to be overcome, as (1) the high content in sulphur, which is prejudicial in iron smelting, and (2) the high percentage of volatile matter in the coal, which results in a soft-bodied coke; and suggests the possibility of utilizing these coals for making domestic coke and briquettes in the retort oven. The proposition is to utilize all by-products (gas, ammonia, tar, etc.), and also the waste heat for steam raising, and to make briquettes from coke breeze, as well as from coal slack, using the coal tar as a binder.

The value of coke in the far West is exemplified by quotations of locally-made coke in Montana—$7·50 to $8 f. o. b. cars at ovens. In this State three companies produced coke in 1898, all having beehive ovens, the coke being largely used in copper smelting. On an average 2·04 tons of Montana coal yield one ton of coke in the beehive.

THE OCCURRENCE AND UTILIZATION OF PEAT.*

BY ERNST A. SJÖSTEDT.

IN the process by which peat is formed the hydrocarbons of the vegetable matter are eliminated principally by fermentation, and the substance therefore consists mainly of the ligneous parts of the plants, in a more or less altered form. And since these, on an average, amount to about 25%, it may be assumed that ripe peat represents the remains of four times the quantity of plant substance. The different grades of peat are chiefly due to the more or less complete decomposition the plants have undergone, the mineral substances mixed with them, and the degree of compression to which they have been subjected. The least decayed kind is called "moor-peat," and from it "moss litter" is manufactured; whereas peat of a more advanced state of decomposition is known as "fuel peat," and is of a dark, compact character.

Distribution of Peat in America.—Within the limits of North America peat deposits of considerable depth exist nearly as far north as the Arctic circle. A line drawn from the Atlantic coast in southern New Jersey westward through northern Pennsylvania, Ohio, north Indiana, north Illinois, central Wisconsin, and eastern Minnesota will roughly include the southern and western limit of the United States in which peat deposits are common and of sufficient importance to deserve mention in the resources of the country, although many local deposits exist in other parts. In the peat districts of the northeastern United States and the adjacent portion of Canada the accumulations are generally of the lake-bog type; but in eastern Canada, Newfoundland and the Maritime Provinces climbing bogs of the type so common in Ireland and other portions of northwestern Europe occur, though they lack the extent and depth of those in the Old World. Deposits of peat are also found at many places in the Province of Quebec, but more particularly in the great stretch of comparatively level land lying to the east of the St. Lawrence. The deposits of Ontario are of great extent; but although earnest and successful attempts have been made to utilize them, the economical results have been discouraging, and so far as known to me no peat industry of any importance is at present carried on in Canada or in the United States.

The Peat Industry in Europe.—After much careful investigation and persistent experimenting, however, the peat industry has now been brought to a paying basis and has become of national importance in many countries of Europe where mineral fuel has not been found. Possibly it is in Holland that the peat deposits are most extensively worked. As to the results obtained there we may get an idea from a late number of a Danish agricultural report, wherein it is mentioned that the Rahder Co., of Nineweroord, has eight steam machines in operation which daily produce 200,000 peat bricks, selling at the bog at 3·70 gulden† per thousand. At a minimum of 100 working days, this means an annual production of 20,000,000 bricks, which are obtained from about seven acres of peat bog, and represent a value of 74,000 gulden (or nearly

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* Reference should also be made to the paper on "Peat," in THE MINERAL INDUSTRY, Vol. II., 1893, to which the present paper is supplementary.

† The Dutch gulden is equivalent to 40·2 cents, United States currency.

$30,000) for this plant alone, and it is estimated that Holland has recovered the peat from 300,000 to 400,000 acres of bog, or to a value of about $1,400,000,000. Denmark also has a great peat industry, and the model works at Sparkjaer produce annually 15,000 tons of peat for local consumption alone, making a profit of about 90c. per ton. In Germany peat, or turf, is used whenever its cost is not materially increased by transportation, being in many localities the only substance employed for heating purposes. It is besides turned into account as a fertilizer and as building material, it being employed successfully as a filler between walls, in reservoirs, ice houses, etc. However, the principal use of peat everywhere in Europe is as fuel. Ireland possesses peat bogs to the extent of one-seventh part of the entire area of the island, and extensive tracts of peat land are found in Scotland, England, Sweden, Russia and France.

Peat Cutting.—The operation of peat cutting is performed in spring or summer, and is preceded by a partial draining of the bog, and the removal of the coarse undecomposed vegetable surface matter. The peat is afterward cut out in brick-shape pieces (peats) by means chiefly of a peculiar implement called in Ireland a "slane," and in Scotland a "peat spade," a long, narrow spade, the blade of which is furnished on one side with a tongue set at a right angle to it. This implement is used by hand alone, without pressure of the foot. The soft peats thus cut (about 12×8×4 in. in size) are conveyed to some nearby place where they are set up on end in little stacks ("stick torf") to dry. When sufficiently dry they may be piled in sheds, or stacked in the open air.

Another method of preparing peat for fuel is to work the black mud of a semi-fluid bog by the feet of a party of men, women or children, until it acquires such a consistency that it can be molded by the hand. This process is laborious, but the fuel obtained is of a superior quality.

Attempts were made early to concentrate peat by compression, making it into briquettes by machine. In Sweden, where, in the seventies, the machine-made "cylinder peat" was so common, this form has been lately abandoned, and now large horizontal pug mills are used, in which the peat is worked into a thick pulp, which is either molded or spread on the ground to a depth of 8 to 10 in. and then cut into rectangular bricks. Such a pug mill of about 5 h. p. costs about $125, and will produce 1,200 cu. ft. of dry peat per day, but requires much drying space.

Preparation of Peat Fuel by Hodges' Process.—The prime obstacle in the utilization of peat as fuel is the large quantity of contained moisture, and the great difficulty experienced in its removal, this difficulty arising in large part from the porosity of the material, and its tendency to absorb moisture; the plan of pulverizing the peat as it comes from the bog, drying it as rapidly as possible, and then solidifying it under great pressure, has so far been found to give the most satisfactory results.

Applying this principle, an Englishman, named Hodges, invented in the sixties a machine which, by means of huge revolving disks, attached to the front of a large scow, cut the material of the peat bog, the surface of which had been previously prepared by clearing away the roots, and at the same time reduced the peat to a very fine and uniform pulp. The pulp was then distributed by means of a long spout, over the side of the scow, upon the prepared surface of the adjacent bog, and left to dry in the sun for some days. As soon as a skin formed on the surface of the drying peat, the sheet was divided lengthwise by cuts about 6 in. apart, and after further drying by a series of cuts at right angles to the first at intervals of 18 in. When sufficiently hard for handling, these blocks were carefully taken up and stacked for further drying, the final result being an excellent quality of fuel at a low cost. The machine was capable of digging, pulping and spreading over 14,000 cu. ft. per 10 hours, yielding 50 tons of air-dried peat fuel, which cost in the barges on the canal 92c. per ton. It contained about 45% water, but was too fragile and absorbed too much water to be useful for purposes to which it had to be transported far.

The above experiments were carried on by the Canadian Peat Fuel Co. at St. Hubert, 10 miles from Montreal, and at Ste. Brigide, 10 miles from St. Johns, on the Richelieu, prior to 1877, 20,000 tons of prepared peat having been produced.

Aikman's Process.—Upon the suspension of operations by the Canadian Peat Fuel Co. Mr. Aikman, who for nine years had been its manager, made some improvements in the process. The peat, after being pulped and freed from roots and other impurities, was artificially compressed, and then dried in the air, the drying requiring about six days. This machine, which had a capacity of 12 tons of compressed peat per day, is thus described by Mr. Aikman:

"The machinery is carried on a scow which is made to float in the bog, the peat being extracted by a pair of large screw augers, working in front of the bow, which deposit it in a well where it is submitted to the action of steam taken from the exhaust of the engine. It is then carried by an elevator to the squeezer, where it is freed from about 33% of water, after which it is broken up by revolving teeth, passed through a centrifugal dryer of a special construction and delivered into the press in the form of a dry dust, which by the action of heat and the screw is brought into a pasty condition, the tar or resin naturally contained in the peat being softened. Under the pressure it becomes partially carbonized, and issues from the press in a continuous stick of peat fuel, ready for use. The whole process is entirely automatic, and only takes about 20 minutes from the time of excavation from the bog to the time it is delivered from the press as a marketable fuel.

"The approximate cost of erecting a plant capable of producing 10 tons per day is $4,690, and the cost of operation is $16 per day."

Dickson's Process.—Another Canadian peat process of considerable promise is that of A. A. Dickson, of Toronto, whose experiments have been directed toward means of compressing the peat after sun-drying, without application of artificial heat, which he thinks has an injurious effect upon the peat as fuel by dissipating much of the valuable volatile matter contained in it. His process consists in spreading the peat (excavated from the bog by hand, or by a dredge specially constructed for the purpose) upon the surface of the ground or in open sheds to dry naturally. The dry peat is then pulverized, by means of a breaker which revolves at a high rate of speed; in this operation, however, the fiber is preserved as much as possible. From the breaker the powder is conveyed to a large hopper, whence it descends to a vertical press, where it is stamped into cylinders 2 in. in diameter, and of the same length. The compressing is done in a tube without a bottom, the resistance to the pressure of about 30 tons being obtained entirely by the friction of the material against the side of the tube. The reduction in bulk from the raw material to the finished block is in the proportion of 6 to 1. The product is hard and dense, weighing 83 lb. per cu. ft., or somewhat less than anthracite, but more than bituminous coal, and so solid that it will stand being dashed with violence against a brick wall without going to pieces. Although the Dickson fuel has not as yet been put upon the market, hundreds of tons have been made and used for experimental purposes, and it has been subjected to numerous thorough tests which have demonstrated its adaptability for both steam raising and domestic purposes.

Danish Practice.—At the Danish works previously referred to (Sparkjaer), the peat machine consists of a scow, floating in a canal dug in the peat bog, on which the pulper (a pug mill) is placed, the plant being operated by a naphtha motor on the scow. The peat is dug by hand, the digger standing higher than the deck of the scow, and throwing the peat into the pug mill, into which water is constantly pumped. The pulp is raised automatically by a conveyor and discharged into dump cars on the tramway parallel to the canal, by which it is taken to the drying place, where the molds are placed ready to be filled by dumping the contents of the car over them, and leveling off the peat pulp with a rake. In this simple manner one man can here dig a ton of peat (calculated as dry) in one hour, or 10 to 12 tons per day, while at many other places 2 tons per man only are recovered.

Manufacture of Peat Briquettes in Pomerania.—A peat briquette factory at Langenberg, Pomerania, Prussia, which has been in existence only two years, and proved to be a complete success, is thus described: The damp peat is thrown into a machine resembling a large turnip-cutter, in which it is broken into small lumps, which pass to a second machine, where they are ground into a fine powder. Thence it goes to a dryer, which is an inclined cylinder with tubes like those in a boiler, but larger, the powder passing through the tubes and exhaust steam from the engine around them. From the dryer the powder passes to a hopper which feeds the plunger. The latter is operated by a 75 h. p. engine, and presses the powder in a mold against the preceding briquette, pushing them forward, each stroke the width of one briquette. The factory turns out 80 briquettes per minute, or 35 tons per day, with an average output of about 12,775 tons per annum, and the demand is far greater than the supply. These briquettes sell at 10c. per 100, which is very cheap fuel. In a closed oven one briquette will remain aglow for 24 hours; in an open grate it burns quicker, but remains aglow a longer time than any coal, giving a good red heat. The cost of production at Langenberg is about $1·70 a ton, and with a larger plant it is said it can be reduced to about $1·20 a ton.

Peat Coal.—By heating peat to about 200° C., or until combustible gases begin to come off, peat coal is formed. During the last decade much has been said and written in regard to its relative value as a fuel, especially in connection with the patents taken out for its manufacture by Mrs. Angel and Lieutenant Ekelund, both of Sweden. In Mrs. Angel's process the peat is placed in closed vessels without gas exit, and charred at a comparatively low temperature, whereby the products of distillation are retained in the coal. The peat used should previously be freed from the larger part of its moisture, and the apparatus heated to a temperature of 50 to 400° C. during one to six hours. The coal thus produced is said to have a metallic luster, and to be hard and free from soot, and can be favorably compared with bituminous coal.

Ekelund's method is based on the burning of the non-condensible gases given off in heating peat to convert peat into coal, and also for drying raw peat. The peat coal is afterward pressed into briquettes by a special process.

The Value of Peat as Fuel.—The value of peat as fuel naturally depends on its calorific effect and its cost of production. To ascertain its heat value thorough tests were made a few years ago by Prof. Klasson, one of a committee appointed by the Swedish government to investigate the merits of peat coal made by the Angel and Ekelund methods. The conclusions reached by him are embodied in an exhaustive report* from which the following table has been computed, showing the average composition of the organic substance in different kinds of fuels, together with the mean calorific value of the absolute dry and ash-free fuel, and the average percentage of moisture in its dried state.

Composition.	Wood.	Peat.	Brown Coal.	Swedish Coal.	English Steam Coal.	English Gas Coal.	English Coking Coal.	Welsh Anthracite.
	%	%	%	%	%	%	%	%
Carbon	52·0	58·0	66·0	78·0	81·0	87·0	87·0	91·0
Hydrogen........	6·2	5·7	4·6	5·1	5·2	5·2	4·9	3·5
Oxygen..........	41·7	35·0	28·0	14·8	11·5	5·5	4·1	3·5
Sulphur..........	....	....	....	0·8	1·0	1·0	1·0	1·0
Nitrogen.........	0·1	1·2	1·0	1·3	1·3	1·3	1·0	1·0
Calories (*a*).......	49	57	60	75	80	87	86	86
Moisture.........	20	22	25	13·5	7·6	1·7	1·4	2·0

(*a*) By calories is here understood the amount of heat required to raise 1 kg. water from 0° C. to 100° C.

With respect to the relative value of peat briquettes, peat coal and peat dust, the Swedish committee expressed a favorable opinion as to each as a fuel, but the peat coal was pronounced to be economically short of what it was claimed to be, 16% of the heat value of the peat being lost in the process of carbonization, and the opinion was expressed that with coal at $4 per ton it would be too expensive to carbonize peat, since peat coal, whether produced in kiln, retort or meiler, would cost about $7 per ton. On the other hand it was considered that, for consumers near the bogs, uncarbonized peat is the cheapest fuel that can be had in Sweden.

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* Jubilee number for 1896 of *Teknisk Tidskrift*.

Peat in form of a highly dried powder has also lately been recommended as an easily transportable, cheap and effective fuel, especially when used in connection with some of the late devices for continuous firing, with or without blast.

Cost of Peat.—As to the cost of producing peat, the committee gave the following average prices for Sweden: Cut peat, $1·05 per short ton to $1·70; molded peat, $1·16 to $1·82; tube peat, $1·60 to $2·19; machine peat, $1·86 to $2·45, the latter figures in each case including interest and depreciation of plant and property. These prices are for peat recovered principally by manual labor; but recently machines have been invented for cutting and raising the peat from the bog which have materially reduced the cost of manufacture, and in *Teknisk Tidskrift*, for July, 1896, the average cost for Sweden is stated to be 50c. and 60c. respectively for cut peat, and $1·01 and $1·14 respectively for machine peat, with an average of 20% water. At the Sparkjaer works in Denmark the best machine peat, weighing 28 lb. per cu. ft., is manufactured for 36c. per ton, or 71c. per ton when interest and depreciation of plant are added, the cost of the bog being estimated at $155 per acre. The wages there range between 82c. and $1·10 per working day. In Würtemberg the cost of cutting, drying and storing, by combined manual and machine work, is 94c. per ton.

Use of Peat in Iron Smelting.—Besides for domestic purposes peat is used as fuel in certain important industries, as for instance, at several of the large iron works in Sweden, in heating furnaces and in the gas producers. Good peat is there considered to be 50% better than an equal volume of soft wood; and when used as peat coal in the blast furnace it has been reported to be 50% better than charcoal. At a recent meeting of mining engineers at St. Petersburg, N. M. Dmitrijewsky* gave the results of experiments with the use of peat in blast furnaces. These were made at the Kulibak works. With cold blast 1 pood (40 lb. adv.) of pig iron required 41·48 lb. of peat coal, and with hot blast only 35·45 lb. were required. And in the neighborhood of Drumkeerin, county Leitrim, Ireland; bordering the shores of Lough Allen, a furnace is now in course of erection for smelting the native ore with peat, and so far as the preliminary trials have gone, the results have been reasonably satisfactory.

Value of Peat as a Fertilizer.—In regard to the use of peat for manurial purposes, Prof. N. S. Shaler says that "Peat may be of service to the agriculturist in several ways. Where the mass has been made up by the decay of leaves and twigs, such as are often borne in large quantities into a lake basin by a swiftly flowing stream, the substance has a distinct value on account of the fertilizing materials which it contains. In general, however, peat, especially that formed by the water-loving mosses, will not, from the chemical nature of its components, repay even a short transportation. On dry soils, however, this half-decayed vegetable matter may be found of distinct value, for the reason that the sands of which they are composed are unable to retain moisture in sufficient quantity for the effective growing of crops, and the water-retaining capacity of the soil is very much increased by the admixture of peaty matter. For such use the varieties of peat formed in bogs fed by streams

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* *Journal of the Iron and Steel Institute*, No. I, 1898.

which are muddy in times of rain are the most valuable. The fine sediment serves to fill in the interstices between the grains of sand and the soil. A third use of peat in agriculture is as an absorbent material, when commingled with other substances; thus in utilizing waste fish, or the offal from slaughter-houses, or manure from barns, it may be made to serve a valuable purpose. It is, indeed, extensively used in this way in many parts of the United States; and as our agriculture becomes more intensive, the service which it can render will be better appreciated. It seems very likely that in the development of our tillage arts it will be more and more the custom for farmers to procure the raw material, and to compound fertilizers according to their needs. When this is done peat will be found a valuable substance in forming the compost beds, its great absorbent power giving it a peculiar value in such work."

Moss Litter.—The material between the upper and the deep-lying peat layers, in which the mosses and the rootlets are only partially decomposed, is used for the manufacture of moss litter. This was first made on a large scale in Hanover in 1879, and since then it has gained steadily in demand, owing to its many useful qualities. As a bedding for domestic animals it is soft, elastic, dry, and absorbent. By reason of the last characteristic it thus becomes a valuable fertilizer as well as a disinfectant (absorbing the malodorous gases, especially ammonia, formed by the decomposition of the manure—a quality not possessed by straw litter). Saturated with the waste liquids from the sugar refineries, it leaves a highly-prized fertilizer for grain and sugar-beets, and as a packing for fruit and fish it has proved to keep these perishable goods in good condition during a long time and under transit. It has also been made to do duty in fish culture, it being pronounced a highly suitable substance in which to pack and ship spawn. Owing to its hygroscopic qualities (absorbing six or eight times its weight of water) it can also be used for drying out damp rooms, while it is also used in the manufacture of porous brick, as filling material of mattresses in hospitals, for storing ice (being a poor conductor of heat, owing to its porosity), for filtering oils, etc.

Peat Fiber.—Peat fibers have long been used in textile manufacture. Thus in Holland, England, and Ireland it is made into mats, blankets, and even finer goods for wearing apparel; and in France and Russia it is made into cloth for bandages and other surgical purposes, owing to its antiseptic properties (peat retarding the development and increase of microbes). During the last few years peat fiber has been recovered in connection with Ekelund's method of manufacturing peat dust, from which operation it is obtained as a by-product. After drying and crushing the peat the fine particles are separated from the coarser fibers, which latter are subjected to a washing process and freed from adhering peat dust. In this state they are ready for use as a yarn or are compressed into bales for export. The peat fibers can be bleached quite white, but are also used in the unbleached state. Peat as a fiber for paper manufacture is also receiving attention in England.

Other Uses of Peat.—Some years ago a method was developed in New York City for making kindling by immersing dried peat blocks (which first had been sawed into small bricks) in raw petroleum, whereby they absorb from 15 to 20% of their weight, after which they are coated with resin to prevent evaporation.

The manufacture of artificial wood from peat is the subject of a German patent. After the peat has been thoroughly mashed and broken up into fibers and dust it is agitated into a pulp, which is dried and then mixed with plaster-of-paris water, to which is added a little lime. This mixture is then put into molds and subjected to strong hydraulic pressure until most of the water is eliminated, after which it is kiln-dried and coated with oil or a solution of resin and alcohol. This artificial wood is said to be very durable and to resist well heat, cold, and pressure, while it can be worked in the same manner as common wood.

In the ordinary process of carbonizing peat its volatile products are lost, but when distilled in closed vessels it yields, besides water, ammonia and inflammable gases, a large amount of oily matter. M. Miron gives* some results from such a distillation at the Montaugier peat works in France. From 2,250 kg. peat the following products were obtained: Peat coal, 1,000 kg.; ammonia, 15 liters; methyl alcohol, 25 liters; acetic acid, 30 liters; benzols, 15·5 liters; illuminating oil, 120 liters; paraffine, 6 kg.; tar, 65 kg.; heavy oil for lubrication, 18 liters. Confining themselves to the recovery of the three principal products, they obtained, per ton of peat, 9 liters of illuminating oil, 4·5 liters of heavy oil, and 1·3 kg. of paraffine. The cost of the distillation of 2,250 kg. of peat was as follows: Drying to 100° C., 47·25 fr.; distillation, 9·77; sundries, 2·75—total, 59·77 fr.

Subjected to a different treatment, peat can also produce ethyl alcohol. This is accomplished in the following manner: Add to the peat sulphuric acid of 30° to 35° B. in sufficient quantity to obtain, with the water in the peat, a 2·5% solution of the acid. Boil for five hours under pressure at 115° to 120° C., filter off (by means of a filter press) the insoluble residue, concentrate the solution, and neutralize the acid with milk of lime and calcium carbonate. Cool the solution to 25° C., allow it to ferment, and distill in the ordinary way the alcohol produced. In this manner 1 gal. of absolute alcohol may be produced from 215 lb. of peat, containing 14% water. Compared with the best yield from good potatoes (with 20% starch), which is 1 gal. of alcohol from 111 lb. of potatoes, this result is very encouraging.

THE COALS OF THE CANADIAN NORTHWEST AND ROCKY MOUNTAIN REGION.

BY GEORGE M. DAWSON.

THE mineral fuels of the Northwest Territories of Canada and those of the Province of British Columbia are referable to the Cretaceous and Tertiary periods. The Carboniferous formations, in which the coals of the eastern part of the continent are contained, have not yet been found to yield useful fuels in the western part of the Dominion, the conditions, in this respect, repeating those met with generally in the West of the United States. The development of such fuels in the western part of Canada is, however, very important, and

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* *La Revue Technique*, April 18, 1898.

it appears to surpass, both in regard to area, thickness and quality of seams, that found and already to a great extent utilized in the Western States.

To the east of the Rocky Mountains proper (the eastern range of the Cordilleran system), and within that range, these mineral fuels are wholly referable to the Cretaceous, with which, for the present purpose, we may include the Laramie. In British Columbia, however, such fuels occur also in beds of Tertiary age and probably referable both to the Oligocene and to the Miocene, or Middle Tertiary.

The mineral fuels of the Great Plains and of the foothills to the east of the Rocky Mountains are found at several distinct horizons. Where the beds lie nearly horizontal and unaltered, they are lignites or lignite coals, containing a large percentage of hygroscopic or partially combined water. They are non-coking and generally crumble on drying, and their utility is thus seriously reduced for general purposes, although they may be employed with advantage locally. The fuels of the Souris region may be taken as typical of this class, the following being a proximate analysis of a characteristic example: Hygroscopic water, 15·11%; fixed carbon, 47·57%; volatile combustible matter, 32·76%; ash, 4·56%—total, 100%.

Further west, better fuels are met with underlying the plains. Though still non-coking, these do not crumble on exposure, and for most purposes take the place of true bituminous coals. They may be designated as lignite coals, in contrast with the lignites proper, above referred to. Of this class is the coal somewhat extensively mined at Lethbridge, Alberta, shipped to Great Falls, Mont., and used largely on the railways. It contains—hygroscopic water, 6·50%; volatile combustible matter, 31·59%; fixed carbon, 54·36%; ash, 7·55%—total, 100%. This particular seam somewhat exceeds 5 ft. in thickness, while some of those of the Souris region attain a thickness of 7 ft. or more. The seams are regular and great areas are underlain by them at easily workable depths, so that the quantity of fuel of this character is in the aggregate enormous. The workable fuel underlying a square mile (without allowing for waste) often runs from 5,000,000 to 9,000,000 tons.

On approaching nearer the mountains, a line is encountered at which the nearly horizontal beds of the plains suddenly become affected by the folding and disturbance characteristic of the foothills. This, in Alberta and for 250, or more, miles north of the International boundary, may be said to lie at an average distance of 15 miles from the base of the mountains proper. Roughly speaking, and as the general result of numerous analyses, it may be said to correspond with a water-content in the fuels of about 5%. An investigation of the facts shows that the diminished percentage of water is probably directly due to the increased depth to which the coals have been buried beneath overlying strata, in the vicinity of the mountains.*

Between the eastern edge of the foothills and the Palæozoic rocks of the mountains, fuels of varying composition occur, in beds which are nearly always much folded and at times contorted and even overturned. These include coking bituminous coals and nearly dry steam-coals, closely resembling

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* *Report of Progress*, Geological Survey of Canada, 1882-84, p. 132 C.

in composition those in some parts of the Carboniferous period in the East. Some of these seams have been opened and are worked intermittently on a small scale where local requirements warrant. Their greatly disturbed condition is often against their profitable working, but there are without doubt many places in the foothill belt where extensive mines will be operated when means of transportation are provided and markets grow up for their product. This is especially the case near the actual base of the mountains, where the coal-bearing strata are often nearly flat, the Palæozoic rocks having been thrust over them by great earth movements acting from the westward.

Within the general eastern line of the Rocky Mountains, and between this line and the great western bounding valley occupied by the upper parts of the Kootenay and Columbia rivers, are several important infolds of coal-bearing rocks, belonging chiefly to the lowest, or Kootanie, group of that formation. These constitute basins or troughs, sometimes entirely cut off by the older rocks from the area of the Great Plains.

The most important of such infolds so far determined are those known as the Cascade and Crow's Nest basins. Mining has been in progress for some years in the first-named basin, where crossed by the Canadian Pacific Railway, at Anthracite and Canmore. The coal-bearing beds here occupy a synclinal fold, overturned to the eastward. The fuel mined at the former place is to all intents an anthracite, while that at the latter is a steam coal. Their composition is shown in the following analyses: (1) Hygroscopic water, 1·04%; volatile combustible matter, 9·15%; fixed carbon, 87·18%; ash, 2·63%. (2) Hygroscopic water, 0·70%; volatile combustible matter, 11·08%; fixed carbon, 79·78%; ash, 8·49%. These coals, like those mined at Lethbridge and previously referred to, are extensively on the railway and are distributed to many parts accessible by rail.

The recent construction of a line of railway through the Crow's Nest Pass, to the south of the main line of the Canadian Pacific, has rendered the exploitation of the Crow's Neck Pass coal-basin possible. The *raison d'étre* of this railway has, in fact, been found largely in the purpose of bringing the Crow's Nest coals cheaply to the metalliferous mines of West Kootenay. The new railway strikes nearly through the center of the coal field, which, although it has not yet been fully defined, must have an area of at least a couple of hundred square miles. There are numerous superposed coal seams, ranging in thickness from 2 to 30 ft., and although the whole series, supposed to comprise about 20 seams, covers only the central part of the field, it is already manifest that we have here one of the most remarkable coal basins known. Dr. Selwyn roughly estimates the total coal underlying each square mile, in one part of the field, at 49,952,000 tons.*

The following analyses may be quoted as illustrating different qualities of the coal from the Crow's Nest field: (1) Hygroscopic water, 1·79%; volatile combustible matter, 25·45%; fixed carbon, 69·14%; ash, 3·62%. (2) Hygroscopic water, 2·10%; volatile combustible matter, 44·41%; fixed carbon, 43·63%; ash, 9·86%. The coals often afford a good hard coke, well suited for

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*Annual Report, Geological Survey of Canada, Vol. V. (N. S.), p. 14 A.

metallurgical use, while other seams partake of the character of cannel coal and yield large quantities of highly illuminating gases.

The new town of Fernie, in the Elk River Valley, is being made the center of the coal industry, and 100 modern coke-ovens are in process of construction there, with provisions for doubling this number at an early date. A spur line about 4 miles in length has been built up the valley of Coal Creek to the point at which the seams are being opened by drifts run into the mountain sides, and it is now probable that every month will show an increasing output of oven-coke from this vicinity. So far, the supply of coke for smelters in West Kootenay has been largely derived from Puget Sound, Wash., Comox, B. C., and from Wales, but the lower figures at which the Crow's Nest cokes can be laid down will materially reduce smelting charges and largely displace the more expensive fuels alluded to.

It is not intended to include here any review of the mineral fuels of the inland regions of British Columbia, to the west of the Rocky Mountains proper, or of those of the coast of the same province. It may be stated, however, that both Cretaceous and Tertiary coals and lignites are abundantly developed in these parts of that province. In the inland region, the utilization of these fuels awaits, in most cases, the provision of local railways, but it may be noted that true bituminous coking coals are known to occur there in strata of distinctly Tertiary age, in the Nicola Valley and on the North Thompson.

On the Pacific coast of British Columbia, the most important fuels are those of the Cretaceous period, extensively developed in Vancouver Island and on the Queen Charlotte Islands. These coals have long been worked on Vancouver Island at Nanaimo and more recently at Comox, the output in 1898 amounting to about 1,253,200 short tons. A large part of this output is shipped to California and shipments are also made to Alaska and many ports on the Pacific. The Queen Charlotte Islands contain beds of both bituminous and anthracite coal, and although the workable character of the latter can scarcely yet be considered to have been demonstrated, the bituminous seams appear to be well suited for profitable working whenever the requisite demand for new sources of supply may arise.

COPPER.

The production of copper in the United States in 1898 reached the enormous total of 535,900,232 lb. (239,241 long tons) exceeding the output in 1897 by 34,529,937 lb., notwithstanding the decrease in the output of Montana, which was due chiefly to the falling off in the yield of the Anaconda mines. The Lake Superior mines, however, made a considerable increase while the Arizona mines turned out a very much larger amount of copper in 1898 than in 1897, and California and Colorado both made important increases. There were numerous new discoveries of copper ore in 1898 and the prospect is for a still larger increase in the production in 1899 especially in view of the high price for the metal which has been maintained throughout the first half of 1899.

The total production of copper of domestic origin in 1898 was 535,900,232 lb. (239,241 long tons), against 501,370,295 lb. (223,825 long tons) in 1897. These figures include the copper contained in copper sulphate recovered as a by-product, chiefly by gold and silver refiners, the tenor of the sulphate in copper being estimated at 25%. The total production of copper sulphate in the United States in 1898 was 55,119,361 lb., against 46,128,822 lb. in 1897. Of the production in 1898, 27,057,860 lb., and of that in 1897, 29,397,575 lb., were produced from pig copper or copper products otherwise included in our statistical reports, while the remainder of the output each year was made as a by-product. The average price of Lake copper in 1898 at New York was 12·03c. per lb. against 11·29c. in 1897; copper sulphate averaged $3·41 per 100 lb. against $3·86 in 1897. A noteworthy feature of the domestic copper industry in 1898 was the further increase in the percentage of the production refined electrolytically. Nearly the whole of the Montana product is now refined in this manner. The total production of electrolytic copper by American refiners in 1898 was 314,107,776 lb. against an estimated production of 250,000,000 lb. in 1897. The stock of copper on hand at the end of 1898 was 54,361,470 lb., against 48,882,143 at the beginning. These figures include copper in course of treatment at electrolytic works, but do not include the large stock in the form of "mineral" carried at Buffalo by the Calumet & Hecla Co. nor the stock at railway depots and in second hands. The statistics of imports reported in the subjoined table are compiled from direct returns from the importers, the statistics of the Treasury Department not including all the copper brought into the United States.

COPPER PRODUCTION IN THE UNITED STATES.

(Pounds of fine copper.)

States.	1895.		1896.		1897.		1898.	
	Pounds.	Long Tons.	Pounds.	Long Tons.	Pounds.	Long Tons.	Pounds.	Long Tons.
Arizona	48,329,403	21,575	73,745,321	32,922	81,019,922	36,170	110,823,864	49,475
California	225,650	101	1,971,545	880	14,129,920	6,308	21,543,229	9,618
Colorado	6,125,000	2,734	9,539,245	4,259	9,437,663	4,213	10,870,869	4,853
Michigan	129,740,765	57,920	144,058,524	64,312	145,839,749	65,107	156,669,098	69,951
Montana	194,768,925	86,950	228,958,164	102,213	237,158,540	105,874	216,979,334	96,866
Utah	2,664,757	1,190	3,550,050	1,585	3,854,821	1,721	5,385,246	2,404
Eastern and Southern States	3,255,000	1,453	3,750,124	1,674	3,727,939	1,664	4,478,218	1,999
All others	1,344,350	601	2,050,000	915	2,018,929	901	2,134,999	953
Copper in sulphate (*b*)	12,000,000	5,357	12,183,210	5,439	4,182,812	1,867	7,015,375	3,132
Total domestic production	398,453,850	177,881	479,806,183	214,149	501,370,295	223,825	535,900,232	239,241
Stock January 1	78,738,689	35,151	86,961,280	38,822	83,600,000	37,321	48,882,143	21,821
Imports bars, ingots, old, and ores	13,875,560	6,194	*a*27,404,087	12,234	*a*26,938,254	12,026	38,922,552	17,376
Total supply	491,068,099	219,226	594,171,550	265,255	611,908,549	273,172	623,704,927	278,440
Deduct exports	138,358,080	61,767	281,905,217	125,851	288,626,240	128,851	299,765,054	*c*133,824
Deduct consumption	265,738,739	118,633	228,666,333	102,083	274,400,166	122,500	269,578,403	120,348
Stock December 31	86,971,280	38,826	83,600,000	37,321	48,882,143	21,821	54,361,470	24,269

(*a*) This includes copper imported in low-grade Spanish and other pyrites chiefly for sulphur, and the copper imported from Canada in copper-nickel matte, in which the nickel is the metal of chief value; also the copper in certain gold and silver ores. These items do not appear in the United States statistics of imports. (*b*) Including only the copper in sulphate obtained as a by-product in 1897 and 1898; in 1895 and 1896 all the sulphate produced was reckoned, which makes the total production of copper somewhat larger than it ought to be on account of certain duplications. (*c*) Not including Mexican copper en route for Europe.

Alabama.—A promising deposit of copper ore was discovered 15 miles S. E. of Piedmont, and within 6 miles of the Southern Railway, where several veins were exposed, one of which was 14 in. wide, with ore assaying 5·24% Cu.

Alaska.—Discoveries of copper ore were reported in the country between the White and Copper Rivers and along the Tannana River. The latter region was named Healyville, and it was said that its exploitation would be undertaken by the North American Transportation and Trading Co. The existence of copper ore in Alaska and the Northwest Territories is known, but it will probably be long before any important supply will come thence.

Arizona.—There was a large increase in the production of this territory in 1899, as shown in the following table:

COPPER PRODUCTION IN ARIZONA.

(Pounds of fine copper.)

Mines.	1893.	1894.	1895.	1896.	1897.	1898.
Arizona Cop. Co.	7,871,819	9,935,812	11,308,910	13,042,000	13,727,911	18,169,096
Copper Queen	13,795,618	12,688,372	15,741,731	23,298,150	23,999,873	33,749,390
Commercial	273,330	90,805	*Nil.*	46,040	*Nil.*	*Nil.*
Detroit	4,942,728	5,777,744	3,790,128	7,016,348	8,405,138	11,428,992
Old Dominion	7,665,293	4,839,386	*Nil.*	5,650,000	2,000,000	1,800,000
United Verde	9,121,146	10,904,453	16,491,402	22,327,950	31,355,025	42,328,926
United Globe	*Nil.*	*Nil.*	*Nil.*	2,020,565	1,241,975	2,847,460
Other mines	103,741	214,536	997,232	344,268	290,000	*e*500,000
Totals	43,773,675	44,531,108	48,329,403	73,745,321	81,019,922	110,823,864

(By James Douglas): "Arizona's greater prominence among the three great copper-producing districts of this country is due to the utilization in her principal mines during late years of sulphide ores. As long as operations were confined to the treatment of oxidized ores they were limited, not only by the comparatively scanty supply of that class of ores, and their inevitable exhaustion at no very distant date, but by reluctance to incur the waste of 1·5 to 2·5% Cu in the slags, which loss seemed to be unavoidable under the most expert furnace management when making copper direct by one fusion in the cupola. The oxidized ores were most plentiful in the southern groups of mines, where the ore occurs in limestone or associated with limestone. The United Verde deposits are lenticular masses in slate, and, therefore, atmospheric agencies not having penetrated to great depth, the copper carbonates, though rich, were shallow, and were exhausted early in the mine's history. This company, therefore, has for many years relied exclusively on altered or unaltered heavy sulphurets, which have been heap-roasted preparatory to matte smelting. But in the Bisbee and Clifton groups there remain resources of oxidized ores sufficient, by mixture with the sulphurets, to make a matte of bessemerizing grade for years to come. The carbonates, however, even in these mines, exist in quantities so much less than the sulphurets that the managements are impelled by caution to restrain their operations within moderate bounds if they would postpone as long as possible the evil day when roasting must be resorted to. Nevertheless, greater freedom of action was obtained as soon as the companies which control these groups of mines decided to handle the sulphurets that had been developed during the exploitation of the oxidized ores, and which have been since exposed.

"The resources in sulphurets are, of course, far greater than the visible supply of carbonates ever was, and, therefore, the Copper Queen, the Arizona Copper Co., and the Detroit Copper Co. have all felt emboldened to extend the scope of their mining and smelting. All three have adopted the pneumatic method of concentrating mattes. It is one of the advantages or disadvantages of this process that a converter to be run economically must be fed up to its capacity, and as this amounts to from 1,000,000 to 1,500,000 lb. of copper per month, any decided increase of plant of necessity involves a considerable leap in production. Another source of increase from the southern mines during 1899 will be the copper bullion from the two Globe companies. Both have been inactive of late pending the arrival of the Globe, Gila Valley & Northern Railroad, which has now reached its terminus in the town of Globe, and both companies will, therefore, during 1899 resume operations. The United Verde ran its large plant more nearly up to its capacity than formerly, but only during the closing months of 1898, and therefore the contribution of this, the largest mine in Arizona, for 1899 will be considerably in excess of what it has been in previous years. Many other properties are being explored, but none has as yet developed into an active producer, nor is any of these new competitors likely to swell the aggregate to any appreciable extent during 1899. The larger product, however, of the United Verde, the Globe mines, and the two Clifton companies will probably represent an increase in Arizona's production to the world's supply of over 30,000,000 lb. in 1889. But there is no reason to apprehend that this increase will be repeated in successive years."

(By John F. Blandy): "There was in 1898 in Arizona such an inquiry for copper properties that it caused a very general prospecting in all parts of the territory, and many new attacks on deposits that had long been known. The most successful efforts were in the Dragoon Mountains in Cochise County, where two furnaces were erected and at the end of the year were getting fairly under way. Promising developments were also being made in the Whetstone Mountains. Work was also done in Pima County, near Tucson. A furnace was built at Williams, in Coconino County, for smelting ores from the Grand Cañon section, but was not put in blast in 1898. Much work was done in proving up deposits in Yavapai County, especially the Buster in the Peck district, and the Copper Mountain and the Swindler in the Big Bug district. These three will be greatly benefited by the Prescott & Southwestern Railroad, recently finished. There is no more promising section than that south of the United Verde, extending for a distance of 4 miles, where much work has been done in development. Developments were made also in the Black Rock district, near Wickenburg, near Mountain Springs, and in the northeast corner of Yuma County, in Striped Cañon, 50 to 60 miles from the railroad at Congress Junction."

California.—The Mountain Copper Co. of Keswick continued to be the only important producer of copper in this State, but the high price for the metal stimulated prospecting in some of the older districts as well as in the vicinity of the Mountain mines, and there are likely to be several new producers in 1899. Diamond drill developments in the Mountain mines are said to have showed large reserves of ore. Copper ores were also developed in Madera, Plumas and Calaveras counties, the old Lancha Plana and Satellite mines at Campo Seco, in Calaveras County, being reopened by Eastern capitalists. Further developments were made in the Morrow mining district, 25 miles east of Randsburg in the Mojave district, where the ores are to be treated by a wet process, the copper being recovered as cement. The copper belt commences at Granite Wells on the Panamint-Death Valley wagon road and extends in a southeasterly direction nearly to the Calico district, or about 12 miles from Daggett. The principal mines are the Feejie, Manzanita, United Yucca and Vera, in which Boston and New York parties are interested.

The Mountain Copper Co., Ltd., produced 221,895 short tons of ore in 1898 said to average 8·42% copper against 165,060 tons of 8·56% copper in 1897. The amount of ore smelted in 1898 was 168,541 tons, producing 10,721 tons of copper in the form of matte against 97,185 tons containing 7,238 tons of copper in 1897. The amount of fine copper marketed in 1898 was 8,273 tons against 6,025 in 1897. The net profits in 1898 amounted to £163,235.

Colorado.—The production of copper in Colorado in 1898, as reported by the Hon. H. A. Lee, Commissioner of Mines, was 10,870,869 lb. against 9,151,592 in 1897. There was only one mine (the Calumet) near Salida, which was worked for copper alone, the remainder of the output of the State being recovered as a by-product in the treatment of silver-lead, and silver-copper ores. However, copper mines attracted a good deal of attention in Colorado in 1898, and there is reason to believe that the State has large resources in this metal. Larimer County, from the vicinity of Manhattan northwesterly toward the Grand Encampment and Battle Lake districts of Wyoming, is considered to be a promising field. A good copper mine is said to have been opened at Red Gorge Cañon, where Grand, Eagle and Routt counties come together. Another promising section of the State is thought to be the Sangre de Cristo range from the vicinity of Poncha Pass down to Baldy. The La Sal Mountains, in the western part of the State, were the scene of the most active developments in 1898. The La Sal Mountains lie along the Colorado-Utah line, the main range being in Utah with its eastern slope in Mesa and Montrose counties, Colo. The Big Injun group of mines which produced some ore in 1898, said to average 15% copper, are situated on the south slope of the mountains. About 15 miles east on La Sal Creek, Montrose County, are situated the mines of the Michigan Mining & Smelting Co. About 12 miles north in Sinbad Valley, in the S.W. part of Mesa County, are mines formerly owned by Mehane & Co., which will be worked in 1899 by a Chicago syndicate. Shipments will be made from Whitewater station on the Denver & Rio Grande Railway. The Bromide Mining Co., of Routt County, Colo., operating on Douglass Mountain, near the Utah boundary, installed a 36-in. water jacket copper furnace to smelt an ore assaying 10% copper, 12 oz. silver, and $2·50 gold. The mine is situated 95 miles from Rock Springs, Wyo., the nearest railroad point. Charcoal is used as fuel.

Idaho.—Developments in the Seven Devils district have not progressed so rapidly as might be expected in view of the recent conditions of the copper market, and the industry is still in the dark as to the supplies of this district. The first shipment was made in July by the Cuprum Smelting Co. to the Nichols Chemical Co.; it consisted, of 13·308 tons of ingot assaying 98·5% Cu, and 2·191 tons of matte assaying 78%; the silver contents of the lot were 602·72 oz. and the gold 6·65. Some difficulties were encountered in smelting at Cuprum, and in October arrangements were made to move the works to Ox Bow on the Snake River, the ore being transported to the river by an inclined tramway 3 miles long, with a fall of 3,000 ft. Developments are to be carried on in the mines in the meanwhile. A recent report from the Peacock mine stated that all of the surface ore body had been removed, and although the lode presented a good showing no sinking had been done and consequently there was no data to indicate the downward extent of the vein or the amount of ore it would yield.

Maryland.—The Maryland Copper Co., a new concern, purchased the old mine near Liberty, Frederick County, and took steps to reopen it.

Michigan.—The high price of copper caused such a development in mining in the Lake Superior region as had not been witnessed in many years. The Baltic was the only new producer, but the names of the new companies are legion. The Arcadian Copper Co. was organized to operate the old Arcadian mine, the Edwards, the Concord, and the Douglas, the entire property comprising 1,920 acres, with a length of 12,750 ft. on the Arcadian lode; work was begun in June in the Arcadian and adjoining mines; a large stamp mill is being erected on Portage Lake, and a railway to connect mine and mill is being laid, with the expectation that the plant will be in operation about July 1, 1899.

COPPER PRODUCTION IN MICHIGAN.

(Pounds of fine copper.)

Mines.	1893.	1894.	1895.	1896.	1897.	1898.
Atlantic	4,221,933	4,437,609	4,832,497	4,894,228	5,109,663	4,377,399
Baltic	*Nil.*	*Nil.*	*Nil.*	*Nil.*	*Nil.*	42,706
Calumet & Hecla	62,825,674	61,842,722	77,439,907	90,356,733	88,378,986	94,103,000
Central	1,177,500	584,950	370,381	469,243	611,172	291,339
Copper Falls	1,000,000	*Nil.*	*Nil.*	*Nil.*	*Nil.*	*Nil.*
Franklin	3,504,244	3,602,608	3,086,933	2,746,076	2,908,384	3,568,978
Huron	562,776	*Nil.*	*Nil.*	*Nil.*	*Nil.*	*Nil.*
Kearsarge	1,546,318	1,998,710	1,946,163	1,337,226	(*a*)	(*a*)
Mass	22,737	41,805	18,372	38,684	24,081	*Nil.*
National	63,433	36,390	50,128	53,762	28,959	18,441
Osceola Consol'd	6,216,975	6,879,000	6,270,373	6,251,304	9,500,000	11,800,000
Quincy	14,398,477	15,484,014	16,304,721	16,863,477	16,924,618	16,354,061
Ridge	25,988	52,481	64,363	*Nil.*	12,590	....
Tamarack	15,085,113	15,375,000	14,840,000	16,495,000	20,000,000	22,500,000
Tamarack Junior	1,610,259	2,350,000	2,605,000	2,330,000	(*a*)	(*a*)
Wolverine	1,025,062	1,665,255	1,817,806	2,195,791	2,316,296	4,588,114
All other mines	251,304	176,011	94,121	27,000	25,000	*f* 25,000
Totals	113,537,793	114,526,555	129,740,765	144,058,524	145,839,749	156,669,098

(*a*) Consolidated with the Osceola. (*f*) Estimated.

The Onondaga Copper Co., was formed to reopen the old Sheldon and Columbia mines at Houghton. The Pawnee Copper Co. took over the large tract of land, 5 miles east of the Central, in Keweenaw County, which formerly belonged to the Conglomerate Mining Co. The Victoria Copper Mining Co. was organized to reopen two long-abandoned mines in Ontonagon County, namely the Victoria and the Forest, and to develop lands belonging to the St. Mary's Canal Mineral Land Co. The Kaukauna Copper Co. aims to exploit 1,280 acres of land in Sections 9 and 10, T. 52, R. 36, including the old Shawmut and Stonington mines. The Trimountain Mining Co. bought the property S. E. of the Baltic which was formerly known as the Baltic Jr. The Michigan Copper Co. took over the old Minnesota mine and began sinking two shafts. The Old Colony Copper Co. was organized under the auspices of the Union Copper Land & Mining Co. and the St. Mary's Canal Mineral Land Co. to explore land east of the Calumet & Hecla property. The Rhode Island Mining Co. was formed by persons largely interested in the Quincy to develop lands, intersected by the Pewabic lode, north of the Franklin Jr., at Hancock; a shaft has been commenced already. Other new companies were the Mohawk, Winona, Massachusetts, Meadow, Tecumseh, Mass Consolidated (a combination of the old Mass, Ridge, Hazard, and Merrimac mines), Washington and Wyandotte.

A number of the old companies were resuscitated. The Union began a shaft on what is believed to be the Arcadian lode. The Centennial commenced sinking on the Kearsarge lode. The Adventure resumed work in its old openings and started a shaft on the Knowlton vein. The Mass Consolidated Copper Co. ordered machinery and began unwatering its property.

It requires considerable time to develop a productive mine in Michigan and none of the new concerns added anything to the output in 1898, the list of producers remaining the same as in 1897 with the exception of the Baltic, which made a small outturn.

The Atlantic Mining Co. in 1898 produced 5,926,450 lb. of mineral, yielding 73·86% Cu, or 4,377,399 lb., which realized 11·83c. per lb. The ore yielded only 11·8 lb. copper per ton, or 0·59%, its gross value being $1·40 per 2,000 lb. The cost of mining was 89·11c. per ton; transportation to mill, 5·55c.; milling 24·11c., making total working expenses at the mine $1·1877 per ton. The total running expenses, including freight, smelting and marketing the product, were $1·3481. The amount of rock stamped was 370,767 tons.

The Baltic Mining Co. produced about 27 tons of "barrel work" which yielded an average of 80·16% copper. The development work in this mine had not progressed far enough to make it a regular producer in 1898, but it is expected that it will be in 1899. The developments in the lode are considered favorable.

The Franklin stamp mill was destroyed by fire in November. A new mill will be completed about July 1, 1899. The Franklin Jr. is said to be developing into a good property which will replace the old Franklin, now worked out nearly up to the Quincy line.

The Osceola Consolidated Mining Co. (Osceola, Kearsarge and Tamarack Jr. mines) mined 637,603 tons of ore; and milled 505,008 at a cost of 28·94c. per ton, yielding 15,848,928 lb. of mineral. The total receipts from copper, etc., were $1,549,820·12, and the total expenses $1,178,028·22.

The Quincy Mining Co. milled 543,592 tons of ore, which yielded 14,712,685 lb. of mineral, while 5,344,257 lb. of mineral was obtained from the rock houses. The total production of refined copper was 16,354,061 lb. The sales of copper realized $1,986,116·81, while the running expenses were $1,336,538·23. The cost of milling was 22·28c. per ton. The Quincy commenced the erection of smelting works at Hancock in May and put them in operation in December.

The Tamarack Mining Co. milled 670,832 tons of ore at a cost of 22·402c. per ton, the total cost of mining and milling being $2·01 per ton. The yield of mineral was 31,127,623 lb. The total receipts were $2,381,388·95 and expenses $1,862,507·02. The Tamarack added a new stamp mill to its plant in 1898.

The Wolverine Copper Mining Co. in the fiscal year ended June 30, 1898, milled 130,089 tons of ore, which yielded 3,949,045 lb. of mineral, containing, 3,470,927 lb. of refined copper, or 1·334% Cu per ton of ore. The cost of production per ton of ore milled was $1·79. The sales of copper and interest account brought in $399,368·77, while the expenses of mining, milling, smelting, and marketing the product came to $276,186·32.

Montana.—The boom in copper led to fewer important developments in Montana than in either Arizona or Michigan, the copper mining continuing to be limited to Butte, with insignificant production outside of that district. The most important feature of the year was the inauguration of more active work by the Washoe company, which prepared plans for a 1,500-ton smeltery at Anaconda. The Washoe company owns the Washoe, Poulin, Pacific, Oden, Limitation, Gold Hill No. 1, Moonlight, and Gladstone mines, and the eastern half of the Blue Jay. The capital stock of the company is $5,000,000, practically the whole of which is owned by Marcus Daly and J. B. Haggin. Work was suspended on the new smeltery of the Parrot company, near Whitehall, and it is problematical when it will be resumed. The Boston & Montana Co. installed four 20-ton converters in the Butte & Boston smeltery, which it operates under lease. The Butte Reduction Works were closed in December on account of the nuisance of their smoke to the city of Butte.

COPPER PRODUCTION IN MONTANA.

(Pounds of fine copper.)

Mines.	1892.	1893.	1894.	1895.	1896.	1897.	(*d*) 1898.
Anaconda	100,000,000	75,256,657	95,578,000	99,775,294	125,350,693	131,471,127	107,214,059
Boston & Montana	30,386,595	31,800,000	57,937,633	60,746,000	60,250,000	60,000,000	62,000,000
Butte & Boston	10,641,269	20,457,928			4,500,000	....	7,000,000
Butte Reduction Works	2,864,000	2,985,485	2,282,000	3,390,000	4,225,647	7,883,795	9,685,088
Colorado Sm. & Mg. Co.	4,560,972	6,703,488	5,158,730	7,750,000	9,090,680	8,911,578	7,657,938
Hecla Cons. Mining Co.	159,859	77,565	362,897	239,631	312,445	215,431	121,039
Parrot	12,438,782	7,791,167	7,469,908	7,257,000	8,045,648	14,824,487	12,444,828
Montana Ore Purch'g Co.	....	8,400,000	12,293,686	14,861,000	15,049,066	13,047,648	13,084,000
Others	....	827,810	2,011,901	750,000	4,242,583	804,474	(*b*)
Totals	161,051,477	154,300,100	183,094,755	194,768,925	*a*228,958,164	*c*237,158,540	*c*216,979,334

(*a*) In addition to 818,531 lb. of Canadian copper smelted by the Montana Ore Purchasing Co. and deducted in the above table, there was also deducted 1,000,000 lb. more of foreign copper estimated as having been included in the aggregate of the above returns, leaving the net amount 228,958,164 lb. (*b*) Included in reports of smelters itemized above. (*c*) Totals reported by E. B. Braden. (*d*) The individual reports include some copper derived outside of Montana, wherefore their sum exceeds the total as given.

The Anaconda Copper Mining Co., during the fiscal year ended June 30, 1898, received at its reduction works at Anaconda 1,459,249 short tons of ore, of which 17,711 was purchased, the remainder coming from the mines of the company. There was shipped from Anaconda 124,417,471 lb. of copper, of which 64,109,486 was in the form of converter bars and the remainder was the product of the electrolytic refinery. There was also shipped 5,074,036 oz. of silver and 16,610 oz. of gold contained in either converter bars or bullion. The output was consequently equivalent to 4·26% copper, 3·48 oz. silver, and 0·01 oz. gold per 2,000 lb. of ore. The sales of the company during the three years of its existence have been as follows:

	1895-96.		1896-97.		1897-98.	
	Quantity.	Av. Price.	Quantity.	Av. Price.	Quantity.	Av. Price.
Copper, fine, lb.	85,476,795	10·03c.	120,864,097	10·76c.	135,002,147	10·82c.
Silver, fine, oz.	4,498,560	67·91c.	7,057,067	64·08c.	5,706,377	56·73c.
Gold, fine, oz.	14,384	$20·62	18,511	$20·65	19,930	$20·67

Up to June 30, 1898, the total production of the Anaconda mine had been 9,614,203 tons of ore, yielding 550,962 tons of copper, 40,658,103 oz. silver and 135,244 oz. gold. The net cost of mining in 1897-98 was $3·77 per ton. The total cost of mining, smelting and refining was $7·89 per ton of ore and the average yield $11·22. The total costs at the mines and reduction works after deducting profits from subsidiary departments were $11,511·999, including $802,956·69 for new construction and new machinery.

With respect to the falling off in production the manager of the company explained this as due to a large portion of the mine having been closed for several months on account of the fire. "This prevented the extraction of ore, and reduced the grade as well as the quantity of what was shipped, as the closed portion of the mine contains a higher grade of ore than the open portion. The fire was not of great extent, and the greatest difficulty was in the gases thrown off. This fire had been smouldering for years, and we supposed it was extinguished, took out the bulkheads and opened up that portion of the mine with the view of taking ore from the stopes. When the air penetrated the fire district, little patches that were still smoldering broke out afresh and threw off so much gas that the men were driven from the workings and we were obliged to replace the bulkheads. I believe the fire has now been entirely extinguished."

The Boston & Montana Consolidated Copper and Silver Mining Co. reported for the year ended December 31, 1898, sales of copper, silver, gold and copper sulphate amounting to $7,448,600 and expenses of $4,053,785, including operating expenses at the mines and works and freight commissions, etc., on the product. The company also realized a profit of $113,743 in the treatment of ores purchased from other mines. The reserves of ore were increased largely during the year, the amount in sight January 1, 1899, being estimated as 2,161,597 tons. The company continued to lease and operate the Butte & Boston smeltery. During the latter part of the year the dressing works ran largely on B. & B. ores, the B. & M. company buying the concentrates and smelting them with its own ores. Two new blast furnaces were completed at Great Falls and the difficulties in running the new No. 7 reverberatory were overcome. The capacity of this huge furnace is about 140 tons per day. The electrolytic refinery was brought up to a capacity of 4,000,000 lb. per month. The dressing works have a capacity of 1,000 to 1,200 tons per day. The steam stamp has been removed and crushers and rolls substituted for it in the third section of the dressing works. The cost of producing copper was reduced to a lower figure than ever before.

(By F. D. Smith): "In Butte in 1898 there was upward of 8,500 ft. of sinking done and a total of 6,550 men employed in the mines. With a total of 50,000 ft. of shafts, 8,500 ft. in one year means much development. The Green Mountain reached 2,100 ft. depth, the deepest in the State, and there are now several shafts ranging from 1,500 to 1,800 ft. Several of the more important companies in 1898 made arrangements to hoist from as deep as 5,000 ft. The Anaconda Mining Co. operated 11 mines, with 10 separate shafts and hoisting plants, employing 3,340 men about the mines, and sunk a total of 3,550 ft. The production fell off largely, work in the richest mine of the group having been arrested by fire, which led to the use of lower grade ore from other mines instead. The Washoe Copper Co., has large smelting works in course of construction at Anaconda. Outside of Butte no new district of special promise has appeared, yet considerable copper ore has been produced and shipped for treatment to smelters at Butte or elsewhere. In particular may be mentioned the Garnet mines, in Deer Lodge County, a new region worthy of note, and certain districts in Cascade and Jefferson counties."

Nevada.—This State made considerable advance as a copper producer. The Glasgow & Western Exploration Co. operated regularly two Brückner cylinders and three reverberatory smelting furnaces on copper ore from the Adelaide mine. The smeltery is situated at Golconda, 12 miles from the mine, and during 1898 was connected with the mines by a railway of 3·5 ft. gauge. Ore is also received from the Copper Cañon mine, 40 miles east of Golconda. These mines are situated in Humbolt County. The product is matte which is marketed partly at Salt Lake City and partly in New York. The Lone Mountain mines near Elko were developed to some extent, and a smelting furnace was put up in the Pine Valley district, about 25 miles south of Elko.

New Hampshire.—The dumps of the old mine at Copperfield, Grafton County, were worked over, and it is said that the mine may be reopened.

North Carolina.—The old copper mines of Ashe and Cabarrus counties attracted attention and early in 1899 plans were made to reopen the Ore Knob, the Copper Knob and the Union. The ores are sulphide, carying gold and silver. According to William B. Phillips about 225 carloads of copper ore have been shipped from the district lying in the southern Halifax County, Va., and the N. E. part of Person County, N. C. The Holloway mine, four miles south of Virgilina, on the interstate line, and the Blue Wing mine three miles south of Virgilina, are producing ore which averages 35% copper. The Thomas mine near the Holloway, is also being developed. The ore of this district is chiefly copper glance and bornite. The country rock is slate, quartzite and porphyry. The ore is shipped by wagon to Virgilina, but a branch line of railway to the mines has already been surveyed.

Oregon.—The Vaughn mine, situated near the mouth of Pine Creek, on the Oregon side of the Snake River, is said to be showing promising deposits of copper ore, the occurrence being similar to that in the Seven Devils district nearby.

Tennessee.—The Pittsburg & Tennessee Copper Co. was a regular producer in 1898. Its mines, together with the London and Burra-Burra, which have not been worked since 1872, have now been purchased by Lewissohn Bros., who intend to exploit them on a more extensive scale. The Ducktown Copper, Sulphur and Iron Co. reopened the East Tennessee mine in 1898.

Utah.—Until 1898, when important discoveries were made in the La Sal Mountains, near the Colorado line, this State had no noteworthy copper mines although a good deal of copper was recovered by the silver-lead smelters in connection with other ores. There were three smelting works in operation in Utah in 1898, namely, the Hanauer, Pennsylvania and Germania, each of which produced copper matte. The Hanauer and Pennsylvania sell their products to the Germania, by which it is reduced to blister copper assaying about 90% Cu. A small fraction of the ore tonnage treated by these works comes from other States, chiefly Nevada. The production of Utah smelters in 1898 was 4,544,076 lb. of blister copper and 192,950 lb. of matte against 3,217,489 and 371,070 in the previous year. Reckoning matte at 55% and blister copper at 90% Cu this was equal to a refined copper production of 4,195,791 lb. against 3,099,829 in 1897. The new Highland Boy smelting works, now in course of construction near Salt Lake City, will be a complete copper smelting plant of 250 tons per day estimated capacity. Reverberatory furnaces are to be used.

The La Sal. Copper Co. began the erection of a smelting furnace in Paradox Valley, San Juan County.

Vermont.—The Elizabeth Mining Co., operating at South Strafford, after three years' work in driving an adit level nearly 1,400 ft. long, cut the vein and connected with the upper workings, putting from 300,000 to 350,000 tons of ore in sight above the adit level. The vein was shown to be upward of 35 ft. in width. The ore assays 6·3% copper after a little sorting. It is expected to put the smeltery in operation during the summer of 1899, when the mine will be able to turn out from 200 to 300 tons of ore a day.

Elsewhere in Vermont there was not much copper mining in 1898. Fear of litigation apparently stopped all prospecting at Ely, though it is said a newly patented pyritic smelter is soon to be tested there on low-grade material. Prospecting was started at Corinth, a few miles north of Ely.

Washington.—A mine of copper sulphide ore, assaying 17·8% Cu, 12 oz. Ag, and 0·1 oz. Au per ton was opened in King County, on the west side of the main divide of the Cascade Range, between Skykomish and the middle fork of the Snoqualmie River, and plans for the erection of a smelting furnace were made by the owner, H. P. Fogh, of Roslyn, Wash. The Washington-Anaconda mines on the Beckler River are to be connected by a tramway with the Great Northern Railway terminus at Skykomish. A new copper district is said to have been opened on the northern slope of Mount Rainier, accessible from the line of the Northern Pacific Railway.

Wisconsin.—Developments were made by the North Wisconsin Copper Mining Co. on Middle River, about 15 miles S. E. of Superior, where native copper occurs in the Keweenawan formation.

Wyoming.—A promising copper district was opened at Battle Lake, Carbon County, where auriferous copper sulphide ore rich enough to stand a wagon haul of 50 miles to railway was found. The Rudefeha Mining & Milling Co. (Haggerty mine) began shipments in October. It has a vein 2 to 3 ft. in width, assaying 30% Cu. The district appears to be extensive.

COPPER MINING IN FOREIGN COUNTRIES IN 1898.

Australia.—The copper deposits of the Chillagoe district, Queensland, occur in a limestone country, which is intersected by quartz reefs. There are five lodes of copper ore now being worked, the principal showing an outcrop of 20 ft. in width for 2 miles in length. The district is at present difficult of access, and smelting has to be done with charcoal, while the supplies and products have to be carried by pack animals a distance of 150 miles to and from the coast. It is proposed, however, to build a railway from Bedford to Chillagoe. The ore deposits of the Chillagoe district occur in Devonian or Lower Carboniferous rocks, probably the former. The rocks are chiefly limestone and the occurrences show a good deal of similarity to those of Arizona. The Griffith Magazine and Girofla mines so far show the largest deposits of this character. The Ruddygore mine, however, appears to be opened in a huge stockwork in a porphyry dike inclosed by granite. The district has been examined by several experienced engineers and geologists, who have agreed generally as to the magnitude of the ore deposits. The Chillagoe Proprietary Co. proposes to utilize the water power of the Barron River, at the falls about 16 miles from Mareeba, for the establishment of electrolytic works to treat the Chillagoe ores.

The Wallaroo & Moonta Mining Co., of South Australia, produced 5,073 long tons of fine copper in 1897, against 4,801 in the previous year. The average yield of the ore was 15·9 in 1897 and 16·4 in 1896. The cost of production in 1897 per ton of copper was as follows: Mining, $166·40; smelting, $55·62; shipping, $3·68; interest, $1·16; office and general, $2·12—total, $228·98 per ton of copper, or 1·02c. per lb. The net profit for the year was £22,294. Up to the end of the year the company had paid £1,710,254 in dividends.

THE WORLD'S COPPER PRODUCTION, 1894–98. (*a*)

Countries.	1894.		1895.		1896.		1897.		1898.	
	Tons of 2240 Lb.	Metric Tons.	Tons of 2240 Lb.	Metric Tons.	Tons of 2240 Lb.	Metric Tons.	Tons of 2240 Lb.	Metric Tons	Tons of 2240 Lb.	Metric Tons.
Algeria	*Nil.*	*Nil.*	35	35	*Nil.*	*Nil.*	*Nil.*	*Nil.*	50	51
Argentina	230	234	150	152	100	102	200	206	125	127
Australasia	9,000	9,144	10,000	10,160	11.000	11,176	17,000	17,272	18,000	18,288
Austria-Hungary	2,120	2,154	1,310	1,331	1.285	1,306	1,655	1,681	1,540	1,565
Bolivia	2,300	2,337	2,250	2,296	2,000	2 032	2,200	2,235	2.050	2,083
Canada	3,786	3,847	3,924	3.987	4,190	4,225	5.905	5,999	8,040	8,169
Cape of Good Hope										
Cape Company	5,000	5,080	5,350	5,436	5,470	5.558	5,290	5,375	4,660	4,735
Namaqua	1,500	1,524	1,730	1,758	1.980	2,012	2,150	2,215	2,400	2,438
Chile	21,340	21,681	22,075	22,428	23,500	23,876	21,900	22,250	24,850	25,248
Germany:										
Mansfeld	14,963	15,202	14,836	15,079	18,238	18,536	17.960	18,247	18,045	18,334
Other mines	2,210	2,260	1,695	1,722	1,800	1,829	2,185	2,220	2,040	2,073
Italy	2,600	2,642	2,500	2,540	3,400	3,454	3,480	3,536	3,435	3,490
Japan	20,050	20,371	18,430	18,725	21,000	21,336	23,000	23,368	25,175	25,578
Mexico:										
Boleo	10,371	10,537	10,450	10,617	9,940	10,099	10,170	10,334	9,436	9,587
Other	1,400	1,422	1,170	1.189	1,210	1,239	1,200	1,219	6,232	6,332
Newfoundland	1,900	1,930	1,800	1,829	1,800	1,829	1,800	1,829	2,100	2,134
Norway	1,885	1,915	2,685	2,728	2,500	2,540	3,450	3,505	3,615	3,673
Russia	5.000	5,080	5,280	5,364	5,100	5,181	6,025	6,121	6.000	6,096
Peru	440	447	450	457	740	751	1,000	1,016	3,040	3,089
Spain-Portugal:										
Rio Tinto	32,689	33,212	32,985	33,513	34,501	35,055	33,900	34,442	33,705	34,244
Tharsis	11,000	11,177	12,380	12,638	11,040	11,217	12,000	12,192	12,000	12,192
Mason & Barry	4.200	4,267	4,100	4,166	3,381	3,435	4,300	4,369	3,600	3,658
Sevilla	1,170	1,188	1,050	1,069	1,025	1,041	810	823	800	813
Other mines	4,805	4,882	4,300	4,369	3,400	3,455	3,050	3,099	3,120	3,170
Sweden	350	356	515	523	500	508	545	554	480	488
United Kingdom	445	452	580	589	580	589	555	564	550	559
United States	161,510	164,194	172.524	175,294	208,760	212,112	223,825	227,420	239,941	243,069
Venezuela	2,500	2,540	*Nil.*	*Nil.*	*Nil.*	*Nil.*	*Nil.*	*Nil.*	*Nil.*	*Nil.*
Totals	324,764	330,075	334,554	339,994	378,440	384,493	405,350	412,088	434,329	441,283

(*a*) The figures in this table are taken from the annual metal circular of Henry R. Merton & Co., except where returns have been received by THE MINERAL INDUSTRY direct from producers.

Bulgaria.—The firm of Stojanow, Wrbenow & Co., of Plevna, acquired a concession to explore for copper ores in the neighborhood of Belogratschik.

Canada.—Copper was produced in Canada in 1898 in four districts, namely, Rossland and Nelson, B.C., Sudbury, Ont., and Capelton, Que. An account of the developments at Sudbury will be found under the caption "Nickel and Cobalt," elsewhere in this volume. The chief producers in British Columbia were the Le Roi, War Eagle and Hall Mines, Ltd. The Le Roi shipped 66,000 short tones of ore containing 52,850 oz. gold, 68,000 oz. silver and 825 tons of copper. The War Eagle shipped 42,779 tons, while other mines at Rossland turned out 7,918 tons. The Hall Mines shipped its product chiefly in the form of anodes. Practically all the Canadian copper was imported into the United States. The Le Roi ore was smelted at its new works at Northport, Wash., 16 miles from Rossland. The Trail Creek smeltery, owned by the Canadian Pacific Railway Co., was in operation. The Cape Breton Copper Co., Ltd., has been organized in Boston to take up the old Coxheath mines, which are said to have considerable quantities of 4% copper ore. The erection of a smeltery is contemplated. The total production of copper in Canada in 1898 was 8,975 short tons.

Cape Colony.—The output of the Ookiep mine in 1897-98 was 26,598 long tons, assaying 18·53% Cu, against 26,660 tons assaying 18·93% in 1896-97. At the annual meeting of the Cape Copper Co., December 14, 1898, further attention was called to the gradual diminution of the reserves at the Ookiep mine, owing to the failure to find any considerable quantity of ore in the deeper levels, wherefore a decrease in its production is to be looked for soon. The Nababeep mine at the end of 1898 had reserves of 194,880 tons, and orders have been given to decrease the output of the Ookiep mine until Nababeep and Kjöli (Norway) come into full operation.

Chile and Bolivia.—The following statistics are reported by Jackson Bros. of Valparaiso:

EXPORTS OF COPPER FROM CHILE AND BOLIVIA.

	1889.		1890.		1891.		1892.		1893.	
	Quintals.	Fine.	Quintals.	Fine.	Quintals.	Fine.	Quintals.	Fine.	Quintals.	Fine.
Bar copper.......	436,935	436,935	521,123	521,183	396,453	396,453	432,693	432,693	391,884	391,884
Copper regulus...	143,099	63,462	109,290	48,629	56,217	23,851	134,509	61,694	111,073	49,542
Copper ores......	131,254	34,565	33,181	6,670	94,240	18,380	23,450	3,661	173,903	29,617
Total.	....	534,962	....	576,482	....	438,684	....	498,048	....	471,043

	1894.		1895.		1896.		1897.		1898.	
	Quintals.	Fine.	Quintals.	Fine.	Quintals.	Fine.	Quintals.	Fine.	Quintals.	Fine.
Bar copper......	417,133	417,133	443,532	443,532	459,380	459,380	418,665	418,665	448,585	448,585
Copper regulus..	37,007	16,126	20,547	8,251	67,945	3[illegible],506	86,538	42,661	81,141	41,930
Copper ores.....	212,909	37,524	192,027	35.440	133,712	24,139	94,550	20,786	324,919	64.023
Totals........	....	470,783	....	487.223	....	518,025	....	482,112	....	554,538

NOTE.—The quintal is 101½ lb. avoirdupois.

China.—Copper constitutes one of the most important sources of mineral wealth in the province of Yun-Nan, where it occurs in large quantities, being also found in the Koui-Tcheou and Tien-Tchang, while it appears that certain red sandstones in the central portion of Se-Tchouan are impregnated with this metal. At Yun-Nan and Koui-Tcheou two different cupriferous formations are distinguished: Veins in (1) limestone, and (2) in the red sandstones, in the neighborhood of a green porphyrite or a melaphyre. The ore is chiefly sulphide, but native copper also occurs. As a rule the copper contains gold and silver, and sometimes nickel, as at Kien-Tchang where the metal obtained is called "white copper."[*] According to same authority the Province of Yun-Nan now exports annually 1,100 tons of copper.

Congo.—Copper ores have been discovered by the natives in four regions in the Congo basin, two of which are already productive. (1) In the lower Congo, north of Manyanga; at Mboko-Songo deposits intercalated in limestones are now being worked; they occur in the French territory and seem to be extensive, probably running into the Congo State. (2) Copper ores exist in the neighborhood of the junction of the Uelle and Mbomu Rivers, also north of Yambinga; these deposits have not yet been examined. (3) Important deposits of copper ore are said to exist in the little known region west of the Albert-Nyanza. (4) The most important copper zone of the Congo occurs at Katanga, where a number of mines are known on both banks of the Lufila; the ore is carbonate passing into sulphide and occurring as lenticular masses in silicious schists.[†]

Cuba.—This island is supposed to have large resources in copper, mines having been worked once on a large scale at Cobre in the Province of Santiago. The copper-bearing zone of this province embraces three districts, namely Coney, Cauto, and Cobre. The first two remain unworked, although the existence of promising ore deposits in them has been proved.

Great Britain.—A copper mine has been opened in Ireland at Cappagh, on the Cork coast, about midway between Ballydehob and Schull. The ore was discovered about 40 years ago, but was not followed up effectively.

The Cape Copper Co. experienced a falling off of nearly £10,000 in the fiscal year 1897-98, owing chiefly to the coal strike in South Wales which crippled operations for several months. Notwithstanding this the company realized a total profit from its combined operations in Cape Colony, Newfoundland, and Norway of £218,345 against £215,238 in the fiscal year 1896-97.

Italy.—The copper smelting works at Leghorn have four Manhés converters and four reverberatory furnaces for copper refining. They employ about 500 men and consume 15,000 tons of fuel per annum. They treat ores from Montecatini with 36·06% Cu, 20·88% Fe, 19·80% S, and 18·25% gangue; from Mossetana with 11·21% Cu, 22·35% Fe, 24·83% S, 0·15% Pb, and 39·50% gangue; and from Boccheggiano with 11·61% Cu, 26·57% Fe, 29·18% S, 0·28% Bi, and 31·5% gangue.

Japan.—According to a recent British consular report the exports of copper from Japan in 1897 were 13,831 tons against 14,396 in the previous year. The decrease in the exportation was not due to a falling off in production, but rather to increased demand in the domestic trade. The principal mines are at Ashio in the district of Totsigi and at Besshi in Yechine. The Ashio mines are opened on veins of sulphide ore in rhyolite. The ore is smelted in six water-jacket furnaces to a matte containing 55% Cu, which is remelted in a cupola and desulphurized in four Manhés converters each of 1 ton capacity. The crude metal, containing 98% Cu, is cast into anodes and refined electrolytically at Tokio. The Besshi mines have a thicker vein of low grade copper pyrites which is smelted for

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[*] From a communication to the Saint-Etienne Section of the Société de l'Industrie Minérale, by M. Duclos, member of the Scientific and Industrial Mission sent to China by the Lyons Chamber of Commerce.

[†] X. Stanier, "The Geology of the Congo," *Transactions of the Institution of Mining Engineers*, September, 1898.

matte in circular water-jacket furnaces, partly on the spot, but chiefly at Niihama. The matte, containing 55% Cu, is roasted and resmelted in reverberatory furnaces to a final product containing 99% Cu.

Mexico.—The company said to be organized by the Rothschilds to exploit the copper mines of Inguaran in the Ario district began operations on a large scale. Among other things it will construct a railway line from Inguaran to Zihuatanejo on the Pacific. The company has a liberal concession from the government. The new Ario Copper Mining Co. is also opening mines in the Hacienda de la Zanja, municipality of Nuevo Urecho, in the district of Ario de Rosales. The Copper Queen Mining Co., operating at Nacosari, Sonora, about 190 miles south of Bisbee, Ariz., opened good bodies of silver-copper ore, and built a tramway from the mines to the smelting furnace. The company also has a standard gauge railway line to Nasco on the international boundary.

New Caledonia.—According to H. W. Edwards* the copper deposits of the northern end of the island are important, though not at present exploited. Considerable work has, in the past, been done upon them, the Balade mine having shipped some 40,000 tons of ore (said to assay 16% Cu) to New Castle, New South Wales. The ore occurs in talcose and chloritic schists, in lens-shaped deposits. The ore is chalcopyrite, oxidized in the upper portions. Some of the lenses are large and are likely to develop into important mines. The Pilon mine has been opened by a shaft and seven levels, and has some 20,000 tons of ore of all grades in stock. The Ao deposit has been cross cut 80 ft. in the oxidized zone, showing high grade ore the whole distance cut. The ore of the Pilon and several other deposits contains 6 to 16 oz. silver per ton.

Newfoundland.—The Cape Copper Co. extracted 66,507 long tons from the East mine in the fiscal year 1897-98, notwithstanding which the reserves increased from 131,891 tons to 217,000 tons. This is exclusive of the West mine, where nothing but exploration work is being done at present although a good amount of fair stoping ground has been opened with ore somewhat richer than that of the East mine. The operations of the company in Newfoundland in 1897-98 resulted in a profit, including interest on advances, of £35,725 against £28,436 in the previous year.

(By James P. Howley, Director of the Geological Survey of Newfoundland): "The Newfoundland Copper Co., Ltd., a new concern, took over the old Little Bay and Lady Pond mines in June, 1898, and in October shipped 443 long tons of ore, and about 20 tons of matte assaying 24% Cu. At the end of December it had in stock at Lady Pond 150 tons of ore and at Little Bay 130 tons. Stewart & Poole, who, previous to the formation of the new company, had a contract to work over the old dumps at Little Bay, shipped in July 220 tons of ore and 30 tons of 24% matte. The Tilt Cove mines (Cape Copper Co.) produced 66,085 tons of ore in 1898, of which 64,931 came from the East mines and the remainder from the West or old mine."

Norway.—The copper mines in Birtavarre were sold by their former owner, Herr Anker of Frederikshald, to an English company. New ore bodies averaging 8% Cu are said to have been discovered in these mines. C. C. Longridge

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* *Engineering and Mining Journal*, May 28, 1898.

described* the Sulitelma mines, situated in the Province of Nordland, in 67° 15′ north latitude. The nearest town is Bodo. The ore is shipped from Fineidet, a distance of 35 km. from the mines. The ore is copper pyrites, sometimes argentiferous. In 1897 there was estimated to be upward of 400,000 tons in sight, but there is reason to believe that the mines will yield vastly more than this. The climate is rather unfavorable, the mean temperature being 39·2° F. There is abundant water power, which is utilized. Miners, carpenters and smiths are paid 4 to 5 kroners per day, laborers 2·5 to 3, boys and women 1·5 to 2. In 1897 the cost of lump ore for export f. o. b. Fineidet was 15·05 kroners per ton, concentrates 17·87, smelting ore 14·90. In 1897 the output was 15,636 metric tons of lump pyrites (45 to 46% S and 5% Cu), 7,098 tons of smelting ore (7 to 8% Cu), and 15,232 tons pyrites fines (45% S and 3·5 to 4% Cu).

Peru.—According to British Foreign Office Report No. 2117, Annual Series, large and rich veins of copper ore have been found in the Cerro de Pasco silver mines. Ores assaying 30 to 40% of copper have been sent to Lima for shipment to Europe while those of lower grade are being stored at the mines until furnaces can be built to smelt them. The production would be stimulated by construction of a railway from Oroya, which is 65 miles distant over a level country. An output of 2,000 tons per month might then be looked for.

Philippine Islands.—According to George F. Becker copper is found in many places in the Philippines. Northern Luzon contains a district which is unquestionably valuable, the best known portion lying around Mount Data, where copper is now produced by the natives. At Mancayan, 5 miles west of Mount Data, there are said to be wide veins with sulphide ore assaying 16% Cu.

Spain.—The exports of copper and copper ores in 1898 are reported by Señor Adriano Contreras in the *Revista Minera* as follows, the corresponding figures for 1897 following in parentheses: Copper ores, 912,238 metric tons (822,570); copper matte 17,260 (15,432); copper precipitate 31,574 (36,055). The mines of Sotiel-Coronada produced 31,027 tons of ore and 1,118 tons of precipitate. In the Asturias the mine of Aramo, which has been under development for about a year past, shipped 102 tons of concentrates, which averaged 10% copper and 0·6% cobalt.

The Rio Tinto Co., Ltd., produced in 1898 1,465,380 long tons of ore, of which 820,862 were reserved for local treatment, the remainder being ore for shipment. The actual amount of pyrites shipped was 618,110 tons. These figures show considerable increases over the previous year. The amount of copper produced at the mines was 20,426 tons against 20,826 in 1897. The copper in pyrites shipped was 13,456 tons against 13,098 in the previous year. The average percentage of copper in the ore was 2·85 in 1898 against 2·81 in 1897. The sales of refined copper were 18,612 tons; copper in sulphate, 981 tons; copper in pyrites, 13,502 tons; total, 33,095 tons. The reserve heaps, from which the copper continued to be extracted at a very low cost, are now estimated to contain 114,700 tons of fine copper.

Tasmania.—According to the government statistics the exports of blister

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* *The London Mining Journal*, Sept. 10, 1898.

copper in 1898 amounted to 4,956 long tons, and copper ore, 394 tons. The Mount Lyell Co. put in operation its second plant, which consists of three furnaces 210×42 in. at the tuyères, 20 ft. in height from tapping floor to charging floor, each furnace having 40 tuyères. During the half year ending September 30, 1898, the company treated 75,865 tons of ore from the open cuts and 7,349 tons from underground, of an average grade of 0·158 oz. gold, 3·46 oz. silver, and 3·07% copper. The converter plants treated 4,728 tons of matte assaying 2·91 oz. gold per ton, 75·28 oz. silver, and 55·28% copper, producing 2,481 tons of blister copper, containing 13,206 oz. gold, 338,507 oz. silver and 2,449 tons of copper. The mining and removal of overburden cost 4s. 5·83d. per ton of ore; smelting 16s. 5·31d.; converting 2s. 5·33d.; total £1 3s. 4·47d., against £1 4s. 11·64d. for the previous half year. The total amount of blister copper produced up to September 30, 1898, was 8,352 tons, averaging 4·756 oz. gold, 102·61 oz. silver, and 98·81% copper. Dividend payments including that of October 1st amounted to £271,700. There are not yet any other important producers in the Mount Lyell district, although some promising prospects are being developed. The Mount Read and Roseberry districts are showing a good deal of zinky ore and developments are being delayed pending the introduction of a suitable method for their reduction.

Transvaal.—The occurrence of a workable vein of copper ore at Rustenburg was reported.

Turkey.—The smelting operations which used to be carried on at Tokat, with ore from the Arghama mines between Diardekir and Kharput, have now practically ceased, owing presumably to the high cost of wood.

Zululand.—According to Mr. Garrard, a deposit of copper ore has been located on the Insuzi River, Nkandhla district, to the south of the Upper Insuzi gold deposits field, where there is a quartz vein about 3 ft. thick encased in decomposed igneous rocks, in which the copper ore occurs in pockets. Samples from the outcrop have given from 10 to 20% copper and a trace of silver. Another deposit of copper occurs on the right bank of the Umhlatuzi River. An outcrop sample showing copper silicate assayed copper 32·4%, silver 2 oz. per ton. No work has yet been done on this deposit.

THE COPPER MARKET IN 1898.

There was unprecedented activity and consequently great prosperity in the copper industries in 1898. This activity was observed in all parts of the world, although the United States led in the increase both in production and consumption. The demand for copper for electrical purposes was enormous, a large number of electric tramways having been installed in England and Germany and elsewhere in Europe. The demand for engineering and ship building purposes was also unusually large, this being due to the arrears of work from the engineers' strike in England in 1897, the large appropriation for warship construction by England, Germany, Russia, and the United States and the general prosperity in the ocean carrying trade which led to large orders for merchantmen. The consumption of copper wire was very large and increased in all directions, as was the case also with brass and sheet copper. When it became evident that there was to be war between the United States and Spain it looked for a while as if the copper market would be adversely affected, but the United States government having placed large orders and Europe becoming afraid that supplies might be cut off and consequently rushing shipments, the business continued excellent and instead of declining in price copper advanced and ruled steady throughout the war.

New York.—The year opened with a fair business at 10⅞c. for Lake copper and 10⅝@10¾c. for electrolytic sorts, but the market developed strength and advanced to 11c. It would have gone higher had not the Calumet & Hecla Co. remained a free seller at that price, and the month closed with none of the other Lake companies sellers at 11c. The visible supplies at the end of January showed a decrease of 1,300 tons, which made buyers still more eager to cover their requirements, and when at the middle of February the Calumet & Hecla refused to sell further quantities at 11c., the market advanced steadily to 11⅝c. for Lake, and 11⅛@11¼c. for electrolytic. The demand was so urgent that copper sold abroad for February and March delivery had to be brought back to supply the domestic demand. Buying continued early in March, in spite of the threatening political situation, and toward the middle of the month large quantities changed hands at 12c. The demand during these winter months was unusual, buyers in former years not having taken hold in this way until the spring. April was rather a dull month. Manufacturers were well supplied, and not ready, owing to the gathering war clouds, to continue purchasing at the higher figures producers were asking. Toward the end of the month, however, some transactions were consummated at 12¾c. Early in May the large shipments from the mines at the opening of navigation began to arrive, and stocks in the hands of consumers, which had been very low, were replenished. When, however, at the end of the month the Calumet & Hecla offered to sell at 12c., for any shipment until August, manufacturers again bought heavily.

During June and July there was little new business, buyers being well covered and holding off. Prices suffered accordingly, and the market declined to 11⅞c., then to 11¾c., and finally to 11½c. At this price the Calumet & Hecla, following the previous policy of trying to keep prices down, offered freely and for any delivery over the exceptionally long period of six months. Home manufacturers, becoming frightened by this unusual proceeding, took but small quantities, and mostly for August and September shipment, while very little was bought for the later deliveries. Accordingly, early in August, the company offered abroad on that basis and readily marketed very large quantities. When this became known here, there was a scramble to cover, and in a few days the market advanced to 12c. The cessation of hostilities at the end of August had a stimulating effect on the already active business, and the still available quantities offering were eagerly competed for; and 12⅛c., then 12¼c., and at the end of August 12½c. was paid. The tremendous consumption and large exports had reduced stocks in this country to a minimum, and refiners could hardly keep up with their orders. There was not much new business done in September and October, but the undercurrent continued very strong, and spot and nearby copper exceedingly scarce. At the end of October buyers had again exhausted their supplies and under very heavy buying the market advanced to 12⅝c. In November business was limited only by the comparatively small quantities offering, and these were eagerly competed for at advancing prices, the market closing with Lake at 13c. and electrolytic at 12⅞c.

Manufacturers having covered their wants, the market during the first half of December was very dull, and prices delined to 12¾c. for Lake. However, about the middle of the month, Europe bought heavily; home manufacturers had also run down their stocks, and large quantities changed hands at constantly advancing figures. The year closed with Lake copper at 13¼c. and electrolytic sorts at 13@13⅛c.

AVERAGE MONTHLY PRICES OF LAKE COPPER IN NEW YORK.

Year.	Jan.	Feb.	Mar.	April	May.	June.	July.	Aug.	Sept.	Oct.	Nov.	Dec.	Year
	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.
1893	12·13	12·00	11·88	11·38	11·00	11·00	10·88	10·00	9·88	9·75	10·00	10·25	10·75
1894	10·13	9·63	9·81	9·50	9·80	8·94	9·00	9·13	9·40	9·88	9·60	9·80	9·56
1895	10·00	10·00	9·75	9·75	10·25	10·63	11·25	12·00	12·25	12·00	11·00	10·50	10·76
1896	9·87	10·64	11·03	10·98	11·15	11·67	11·40	10·98	10·66	10·66	11·23	11·28	10·88
1897	11·75	11·92	11·80	11·48	11·03	11·11	11·11	11·16	11·30	11·13	10·88	10·78	11·29
1898	10·99	11·28	11·98	12·14	12·00	11·89	11·63	11·89	12·31	12·41	12·86	12·93	12·03

London.—At the commencement of the year the visible supply stocd at 31,955 tons, and the market opened with English consumers holding off, but on the Continent there was a good demand, which was met principally by direct sales by American producers. Later in the month English users bought rather more freely, and the Americans stiffened in their prices. There was also rather more speculative buying, based on the probable early termination of the engineering strike. The opening values were £48 1s. 3d. cash, £48 10s. three months, and rose steadily until £48 2s. 6d. and £49 8s. 9d. were touched for these positions, but after a temporary reaction to £48 16s. 3d. and £49 2s. 6d. the tone again became firmer, and the final rates were £48 18s. 9d. and £48 6s. 3d. During the last part of the month large purchases of "put" and "call" options were made, sellers accepting 22s. 6d. per ton on the current three months' prices. A large line of single call options were also taken at 12s. 6d. per ton. India was inquiring rather more freely for brazier sheets, and the demand for the home trade was steady.

February started with a decrease in the visible supply to 29,746 tons, and prices advanced accordingly, especially for consumers' copper. The earliest values were £49 5s. for cash and £49 11s. 3d. for three months, from which point they advanced steadily to £50 2s. 6d. and £50 8s. 9d., which were the closing prices of the month. The market was greatly influenced by the strong positions in the United States, where values were considerably above the parity ruling on this side, and stuff sold by American producers was not forthcoming as the time of shipment arrived. In addition to these facts some Lake, which had been sold to consumers on the Continent, was bought back and reshipped to America. India remained out of the market.

March commenced with higher prices quoted from America, which stimulated European consumers to cover their requirements, and this demand was met by the English producers, who obtained good prices. The speculative market opened at £50 8s. 9d. spot, £50 13s. 9d. three months, rising 2s. 6d. further, but G. M. B.'s were then rather depressed, owing to profit taking on the part of weak holders. Statistics showed a further decrease of 1,259 tons in the visible supply, and this, coupled with the fact that 550 tons of Chile bars were shipped directly to the United States, caused a very bullish feeling, and G. M. B.'s were soon pushed up to £51 2s. 6d. cash, £51 10s. forward. At this point politics were considerably disturbed and caused a good many realizations, bringing the price down to £50 7s. 6d. cash, £50 15s. three months. America, however, continued to send over good advices, and this turned the tide, values again moving upward. India came into the market and bought copper and yellow metal sheets.

April commenced with a great scarcity of consumers' copper, and the margin between refined and G. M. B.'s, notwithstanding the appreciation of value in the latter sort, was wider than it had been for a long time. The visible supply at this time was 28,023 tons, which was a decrease of 1,239 tons, compared with the stock at the beginning of March. Cash metal started at £50 9s. 9d., with three months' prompt fetching £50 16s. 3d., from which point they crept up to £51 3s. 9d. and £51 10s. The prospect of the colliery strike, which might interrupt smelting operations in England, counteracted the political uncertainty that was prevailing, and after a temporary setback to £50 17s. 6d. spot and £51 5s. forward, a strong speculative feeling was evident, and on good buying values were easily carried up to £52 11s. 3d. cash and £53 2s. 6d. three months. At the end of the month, however, offers of forward copper were rather pressed by Americans, and this caused a decline, the closing rates being £52 2s. 6d. and £52 12s. 6d.

The stocks at the beginning of May showed an increase of 868 tons, and this, aided by a domestic sale in the United States of 5,000 tons of Lake at 12c., caused the G. M. B. market to break away, and prices soon fell to £51 7s. 6d. cash, £51 16s. 3d. forward. At this level things became somewhat steadier, but when it was found that other American producers had followed the lead of the Calumet & Hecla Co. and were making rather free sales a good deal of bear selling was noticeable, and values receded to £50 18s. 9d. cash, £51 7s. 6d. three months. This position suddenly changed and prices began to rise, and in a day or so there were buyers at £51 15s. spot, £52 2s. 6d. three months, but on the publication of the American production figures, which showed an increase of about 2,600 tons compared with April, holders became somewhat scared, and, with the assistance of the bears, depressed the market until £51 was accepted for cash and £51 5s. for forward. Spot stuff became very scarce, and the contango on forward prompts almost disappeared; the covering by shorts then led to a more active business, and values improved to £51 7s. 6d. cash, £51 10s. three months. The final rates were about this figure. The demand from consumers was unimportant throughout, but India bought on a small scale.

At the commencement of June stocks again showed a decrease of 1,102 tons and the price of G. M. B.'s opened at £51 2s. 6d. cash and £51 10s. three months, from which point they reacted to £50 17s. 6d. and £51 1s. 3d., but steadily improved to £51 6s. 3d. cash, £51 10s. three months. Consumers were only buying in a very half-hearted way, and reports coming to hand of a largely increasing American production caused rather a severe onslaught by the bears, and values speedily gave way until £49 7s. 6d. was accepted for cash and £49 12s. 6d. for three months, which were the lowest figures that had been seen for a considerable time. The position then—which was evidently stronger than most people thought—attracted outside speculators, and at the end of the month prices had recovered, and closed at £49 16s. 3d. cash, three months selling at £50 3s. 9d. Consumers, both in England and on the Continent, abstained from buying until English and American producers reduced their prices, which they did when almost the lowest point was reached, and a large business was done, particularly in electrolytic and tough. India remained quiet. The statistics published at the end of this month showed an increase of 312 tons, which was, however, regarded as satisfactory, and led to a continued good inquiry.

During the early part of July values steadily improved to £50 16s. 3d. cash, £51 2s. 6d. three months. There was then some disposition to realize profits, and the fact that some of the Lake producers had reduced their price to 11⅝c. and that the Calumet & Hecla Co. was selling to Continental consumers, caused holders of G. M. B.'s to sell freely, soon bringing the price down to £49 10s. cash, £49 15s. three months. Trade on the Continent also seemed to slacken somewhat at this time, but toward the end of the month the prospect of peace between Spain and America, and the reports that the copper sulphate makers intended to combine, improved that market and caused the bears of G. M. B.'s to get scared, and they, in covering their sales, drove the market up to £50 3s. 9d. cash, £50 11s. 3d. three months, at which figures the month closed. The demand for manufactured and refined copper was quiet throughout, and the prices from India were below market values.

August opened with a decidedly good tone and this quite counteracted the bad statistics, which showed an increase of 1,760 tons for the month. The opening value was £50 8s. 9d. cash, £50 15s. three months, from which point there was an advance, with minor fluctuations, to £52 2s. 6d. cash, £52 7s. 6d. three months. There were then some moderate sales, which were probably due to profit taking and possibly to some bear selling, which caused the market to decline to £51 15s. cash and £51 17s. 6d. three months. There was rather more demand during the month from the home trade, principally for shipbuilding and kindred work, and the export trade seemed to be improving. India and China were rather more disposed to buy, but it was not until the end of the month that they came up in their prices.

September opened with a visible supply of 28,397 tons, which was a decrease of 1 464 tons compared with the preceding month. The opening figures were £51 12s. 6d. cash, £51 15s. three months, and the rise was then almost uninterrupted to £52 5s. and £52 7s. 6d., respectively. There then seemed more disposition to sell American copper, and this caused values to sag off, but at £51 13s. 9d. cash, £51 17s. 6d. three months, the fall was arrested and prices improved to £52 2s. 6d. cash, £52 6s. 3d. forward, and for the remainder of the month moved up and down within a fraction of these figures. The demand from home consumers was only moderately good, and those on the Continent held off to a great extent. September closed with a decrease of 814 tons for the month, which reduction exceeded expectations.

This, in conjunction with a better demand from the Continent, caused an advance in the early part of October to £53 2s. 6d. cash and £53 6s. 3d. three months. After easing off to £52 15s. and £52 2s. 6d., values took another spurt and rose to £54 5s. cash, £54 11s. 3d. three months, which prices were practically the closing ones. Buying on behalf of speculators was very persistent and the consumptive demand was remarkably good. A good business was done in sulphate, and Eastern buyers were inquiring, although they had not yet made up their minds to meet the higher prices ruling here. English consumers bought more freely, and producers, having made good sales at high prices, withdrew.

The statistics published November 1 showed a visible supply of 25,798 tons, which was lower than the previous month by 1,785 tons. The political situation at the commencement was rather unsettled, and the improved statistical position did not have its fair share of attention. The opening rates were £54 8s. 9d. cash, £54 13s. 9d. three months, but there was a drop to £54 2s. 6d. and £54 10s. There was then some very good buying and this, coupled with strong reports and advancing prices from America, caused the prices here to bound up to £56 15s. for all positions. At this time there was a backwardation of 10s. per ton on forward metal, brought about by the continued uplifting of warrants and the free selling of three months' prompts. The close was rather easier, with buyers of cash at £56 2s. 6d. and £56 three months.

December opened with stocks showing an increase of 1,099 tons, but this was attributable to the fact that about 1,000 tons argentiferous bars had been put into stock. They were sold, but had been shipped earlier than consumers wished, and had been consequently put into public store. This surprise led to rather active selling and prices soon fell, three months' prompt being particularly pressed for sale, but after reaching £55 1s. 3d. cash and £54 18s. 9d. three months there was a sharp reaction to £55 15s. and £55 17s. 6d., respectively; but this improvement was not maintained and prices became very irregular. After touching £55 for cash and £55 3s. 9d. for three months, there was an almost gradual rise to £56 5s. spot and £56 11s. 3d. for three months.

COPPER SULPHATE OR BLUESTONE.

Copper sulphate, or bluestone, is made chiefly in the United States as a by-product by the silver-lead refiners, and producers of electrolytic copper, although there are a few concerns which make it from metallic copper. A large part of the domestic production is exported to Europe, chiefly to France, Italy and Austria. To Italy and Austria the shipments are made directly, but to France they go by way of England, in order to escape a discriminating duty. This is 4 fr. per 100 kg. (34.8c. per 100 lb.) on sulphate from the United

States, being 1 fr. more than that imported into France from England and other countries having commercial treaties with France. A considerable quantity of sulphate has been sent from the United States to Mexico in recent years, and this trade is a growing one. The exports of copper sulphate from the United States in 1898 amounted to 14,529,466 lb. ($466,244).

PROGRESS IN THE METALLURGY OF COPPER.

The following review of progress in the metallurgy of copper in 1898 is based chiefly on the technical literature of the year, although some original notes and criticisms have been incorporated.

Improvements in Matte Smelting and Converting.—W. S. Reese and E. E. Armstrong, of Natrona, Pa., patented a continuous smelting plant which has been adopted by the Pennsylvania Salt Manufacturing Co. This consists of a blast furnace with a movable settler and a series of oxidizing furnaces, placed at successively lower levels, so that the metal may be tapped from one to another for gradual enrichment. It is claimed that in this manner smelting can be performed more cheaply than by means of the ordinary blast-furnace plant with ladle and tilting converter, but for the latter it substitutes practically a series of stationary converters with the necessary and numerous concomitants of air pipes, linings, cranes, settlers, etc., and it seems to us doubtful if the economy claimed will be realized in practice.[*]

J. Colquhoun, of Clifton, Ariz., patented the use of a reverberatory furnace into which the matte and slag from the cupola are tapped and from which the matte is drawn to the Bessemer converter, this process having been employed by the Arizona Copper Co. at Clifton, Ariz., since the installation of its Bessemer plant in October, 1897.[†] The efficiency of the system is undoubted, but there is some question as to the novelty of the process, the principle of which at least has been used extensively for five years or more at the Boston & Montana works at Great Falls, Mont.

O. S. Garretson patented a cupola furnace designed to carry on in one continuous operation the three steps in the modern process of treating copper sulhide ores, namely roasting, smelting, and converting.[‡] This consists essentially of a rectangular blast furnace designed for pyritic smelting, of which the hearth is divided into unequal portions by a water-jacketed dam parallel with the short axis of the furnace. The smelting proceeds as in the ordinary pyritic or semi-pyritic process, the slag and matte accumulating in the crucible, and the slag overflowing at one end into a movable settler arranged so as to trap the blast. Besides the air blown into the furnace through the tuyères above the slag level, air is blown through the hearth of the smaller division, converting the matte therein into metallic copper, just as in the ordinary Bessemer converter, metallic copper being tapped at proper intervals from the adjacent end of the furnace, and matte flowing over the dam to take the place of the metal thus drawn off. The slags which are formed in the converting end of

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[*] United States Patents Nos. 605,451 and 605,464, June 7, 1898; English Patent No. 7,008, March 22, 1898.

[†] United States Patent No. 604,023, May 17, 1898; English Patent No. 11,180, May 17, 1898.

[‡] *Engineering and Mining Journal*, Feb. 5, 1898.

the furnace are necessarily rich, but since they flow to the opposite end of the furnace over the low grade matte and are bathed in the shower of falling matte they are claimed to give up their value to the sulphur and escape into the outside settler as ordinary clean slags. The silica required to slag the ferrous oxide formed in the conversion of the matte is obtained by charging an excess of silicious ore into the end of the furnace corresponding to the converting hearth. The furnace is run with hot blast, the air required for converting the matte being heated by passing through the hollow hearth. It is intended to build these furnaces 12 ft. long, and 4 ft. wide, with jackets 9 ft. high, and plans are now being carried out to give them a practical trial. The idea is novel and rational and has the endorsement of Dr. E. D. Peters, Jr., although he foresees difficulties in carrying it out. Other metallurgists criticize the idea of attempting to perform in one furnace three processes which for the best work require conditions so widely different.

Paul David, manager of the works of the Société des Cuivres de France, at Eguilles, has devised a special form of converter with a view to tapping off a copper bottom, in which most of the impurities that would otherwise enter the copper will be concentrated, before completing the conversion of the entire charge, whereby two grades of copper are produced in a single operation, a small part of impure and a large part of high grade metal. The David converter in principle differs from the ordinary Bessemer converter only in a detachable side pocket which is bolted to the main vessel. In operation the channel connecting the pocket with the main converter is closed, and the converter placed in an inclined position, with the pocket on the upper side, is charged in the ordinary manner. The converter is then turned upright and blown partially, tilted to pour off slag, turned upright again and blown to produce the desired copper bottom, and then tilted in the opposite direction to allow the bottom to flow into the side pocket, the connecting channel having been opened in the meanwhile by the insertion of a bar through a suitable working hole. The bottom having been tapped off the connecting channel is closed, the converter is turned upright again and the conversion of the charge is finished in the ordinary manner. The time of the operation, the percentage of the copper bottom to be made, etc., depend upon the character of the matte treated. The cost of treating 1,000 kg. of matte with 35% copper is said to be 21·70 fr., labor being reckoned at 3·5 to 5 fr. per day. Besides the special form and operation of the converter the advantages claimed are saving in time, and the greater purity of the copper furnished to the electrolytic refinery.*

F. R. Carpenter subjects silver and gold-bearing copper to an oxidizing smelting in the presence of silica to form cuprous oxide and cuprous silicate, both of which are readily fusible and absorb but little silver and hardly any gold. The cuprous oxide and silicate are skimmed off (the gold and silver concentrating in the bottom) and are used again in collecting precious metals in ore smelting. The addition of lead toward the end of the process protects the silver so that a larger percentage remains with the gold than would be the case otherwise.†

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* *Annales des Mines*, 1898; *Engineering and Mining Journal*, Oct. 22, 1898.

† United States Patent No. 897,139, Jan. 11, 1898; *Engineering and Mining Journal*, LXV., p. 193.

Improvements in Reverberatory Smelting Furnaces—Herbert Lang, of Oakland, Cal., patented* a reverberatory furnace of circular form outside, with a circular hearth and roof in the shape of a segment of a sphere, claiming advantages of strength and simplicity in construction, and facility of making repairs during operation.

Henry L. Charles, of Butte, Mont., patented† a method of tapping slag from reverberatory furnaces, which consists in blowing air over the surface of the slag toward the taphole, a suitable series of tuyères being provided and the blast being used only during the operation of drawing slag.

Electrolysis of Argentiferous Copper—William Terrill considers that the only way to insure a constant production of a good quality of copper is to discard the electrolyte when it becomes impure and replace it with a new solution.‡ He proposes to evaporate the foul liquor until the bismuth is mostly precipitated, giving a rich deposit of bismuth oxide which can be easily reduced to metal, after which the free acid is neutralized in the usual way and the copper sulphate is crystallized out. A drop of bromine water will show when the liquor has become so impure that it would be dangerous to the quality of the deposit to continue its use. This produces a white cloud of bismuth oxide in the liquor fully 48 hours before any danger has been incurred. The proximity of the changing point can be judged by the rapidity of the development of the precipitate, the white cloud appearing almost instantaneously when the point is at hand.

Cost of Copper Smelting at Butte, Mont.—F. Klepetko stated as a witness in the trial of the suit at Butte between the Butte & Boston Mining Co. and the Montana Ore Purchasing Co. in March, 1898, that the cost of producing copper at Butte was approximately as follows: Carting to dressing works $0·19, dressing $0·82, roasting $0·38—total $1·39 per ton of ore. From a ton of ore 0·42 ton of concentrates would be obtained. The average cost of roasting in 1896 and 1897 was 98·8c. per ton, and 42% of that amount gives $0·38. The roasted ore yields 78·788 of a ton of matte. The average cost of smelting was $3·04 per ton, or $2·40 per ton figured back to the original ore. The cost of converting at Great Falls was 1·4c. per lb. of copper. The total cost of treatment of a ton of ore up to the time it was made into fine copper was $3·47. This includes office, general and construction expenses. In the Butte & Boston works in 1897 the average loss of copper in dressing was 18%; in roasting there was a loss of 2·6%, and in smelting a loss of 4·2%. In converting Butte & Boston matte at the Great Falls works in 1894, 1895, and 1896 the salvage was 95·7%, consequently 73·2% of the copper originally in the ore was recovered in the form of converter bars. Basing the calculation on the ore after dressing, the salvage in roasting, smelting, and converting is 89·3%. In 1894, 1895 and 1896 the Butte & Boston matte was sent to Great Falls for conversion, and the cost including transportation to Great Falls was $1·07 on the original ton of ore.

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* United States Patent No. 604,054; *Engineering and Mining Journal*, Aug. 6, 1898.

† United States Patent No. 608,837; *Engineering and Mining Journal*, Dec. 17, 1898.

‡ *Transactions of the Institution of Mining and Metallurgy*, 1898.

At the Anaconda works in 1896-97 the cost of smelting was $1·85 per ton of ore, while converting, smelting and casting came to 85·9c. H. W. Hixon, in *Notes on Lead and Copper Smelting*, gives the loss in dressing at Anaconda as 18% copper, in smelting 9%, converting 3%, and casting 2%. Consequently in the Anaconda practice only 86·2% of the copper contained in the product of the dressing works is recovered, the loss in smelting and converting being 13·8%.

According to the official report of the Anaconda Copper Mining Co. for the year ended June 30, 1898, four new 12×100 ft. Holthoff-Wethey calcining furnaces were installed at a cost of $122,619·91. Two new reverberatory matte smelting furnaces cost $30,417·34. The converter plant now has 15 stands with a capacity of turning out 14,000,000 lb. of pig copper per month. The copper produced in the refinery was of uniformly high grade, more than 99·9% of it being above the requirements and tests for the highest grade of copper. The cost of concentration in the dressing works was 77c. per ton. Smelting came to $1·926, converting and casting 73·6c., blast furnace work 26·4c., refining 41·4c., and reduction of silver mud 5c.

The Le Roi Smeltery at Northport, Wash., has a capacity of 400 tons of ore per day. There are 86 roasting stalls, one 10×100 ft. Holthoff-Wethey roasting furnace, two 38×120 in. blast furnaces, and two Connellsville cycloidal blowers. The slag is granulated by powerful streams of water striking the hot stream from the forehearths. The low-grade matte is granulated by a similar process. The fluedust and calcines are bricked by machine before treatment in the blast furnaces. The plant was blown in January 21, 1898.

Smelting Lake Copper.—The Lake Superior copper "mineral" is smelted by the Lake Superior Smelting Co. at Hancock and Dollar Bay, the Calumet & Hecla Co. at Torch Lake, Mich., and Buffalo, N. Y., and the Quincy Mining Co. at Hancock, Mich. The Quincy and the Calumet & Hecla smelt only their own mineral, while the produce of the other Michigan mines is handled by the Lake Superior Smelting Co. In 1897 the Hancock works produced 29,644,224 lb. of refined copper; Dollar Bay, 32,130,488; Torch Lake, 31,867,192; Buffalo, 56,511,794—total, 150,153,598. The percentage of copper yielded by the mineral of the different mines is about as follows: Atlantic, 75·009; Calumet & Hecla, 70; Centennial, 75·38; Central, 69·9; Franklin, 81·107; Osceola, 84·083; Quincy, 79·489; Tamarack, 67·421; Wolverine, 86·414.

Copper Smelting at Sulitelma, Norway.—C. C. Longridge describes the practice as follows:[*] The ore containing 7% copper is roasted in heaps, whereby the percentage of sulphur is reduced from 28% to 8 or 9%. The heaps are 32×24×3 meters at the bottom, sloping to 2·5 meters in width at the top. About 6 meters at either end are built on wood, the flames being drawn toward the center by air channels. The ore is broken to about fist size, and each heap contains 1,000 metric tons. The roasting occupies eight weeks and costs 1·51 kroners per ton (the kroner is equivalent to 27c. United States currency). The roasted ore is smelted to a 45% matte in two Fraser & Chalmers water jacket cupolas,

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[*] *London Mining Journal*, Sept. 10, 1898.

36 in. and 42 in. in diameter, the charge consisting of 70% roasted ore and 30% slag, with 14% of Newcastle coke, which costs 23 kroners per ton at the smeltery. The blast is generated by water power. Smelting costs 8·64 kroners per ton of ore. The matte is run directly from the cupolas to a converter, where with a 9-lb. blast it is blown up to 99·54% copper, the converter charge being 300 to 400 kg. after relining, and increasing to 800 or 900 kg. as the lining is eaten out. The lining, which is composed of 80% quartz and 20% clay, lasts four to five blows. Poling before pouring increases the purity of the copper by 0·5%. The cost of smelting and bessemerizing come to 250 kroners per ton of copper, and including all charges for ore, etc., the cost of copper per ton is about 500 kroners.

The Welsh Direct Process of Copper Smelting.—W. Terrill, in an interesting paper on copper smelting in Wales, which does not embody sufficient new features to warrant a resumé here, gives comparative figures of the cost of producing copper by the old Welsh roaster process and the modern method called the "direct process." The treatment of 100 tons (2,240 lb.) of ore assaying 15% copper is given as 14s. 2d. per ton of ore or £4 14s. 4d. per ton of copper by the roaster process, against 12s. 7d., or £4 3s. 11d., respectively, by the direct process.*

Manufacture of Copper Sulphate from Metallic Copper.—E. Sorel finds that in place of roasting the copper, or treating it with nitrous or sulphurous compounds before exposure to the action of sulphuric acid, the addition of a small quantity of such organic bodies as oxalic acid, alkali oxalates, lactic acid, alkali lactates, tartaric, malic or citric acid, or the corresponding salts, or crude glycerine, facilitates the action of the reagent, without the added substance being itself decomposed or consumed. He proposes to apply this method in practice by charging a vertical vessel with copper turnings, which are then exposed to the action of a continuous stream of dilute (1:3) sulphuric acid, containing a few grams per liter of one of the organic substances cited, and kept at 75° to 80° C. by steam heat. Provision is made for the passage of a current of air through the apparatus. This arrangement results in the copper being rapidly attacked, and the solution, which drains off through the false bottom of the apparatus into a settling tank, is returned for use over and over again until saturated, whereupon it is cooled and the copper sulphate crystallized out. The copper and acid are renewed as required, and the wash water is utilized to dilute the latter. The crystals of copper sulphate obtained are very small, and said to be of a high degree of purity.

A New Volumetric Method for the Determination of Copper.—Richard K. Meade says† that the following method is more accurate and more widely applicable than either the cyanide or iodide method: The copper is precipitated from an acid solution by addition of ammonium or potassium thiocyanate, and the cuprous salt thus obtained is converted to cuprous oxide by warming with potassium hydrate. To this is added a solution of ferric chloride or sulphate, and the ferrous salt formed by the oxidation of the cuprous oxide is titrated with standard permanganate solution.

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* *Engineering and Mining Journal*, Dec. 3, 1898. † *Journal American Chemical Society*, **XX.**, pp. 610-613.

THE ELECTROLYSIS AND REFINING OF COPPER.*

BY EDWARD KELLER.

PART I. THE ELECTROLYSIS.

IT is not within the scope of a short treatise on the electrolysis of copper to dwell on the theories of electricity or on the mechanism of dynamo-electric machines, although all are inseparably connected. It is, however, important for the students of the subject to have a clear conception of the elementary principles that underlie the practical working of the process, and for that reason a few words should be said concerning them.

General Principles.—Wherever there is any work done by means of electricity there are three governing factors, *i.e.*, the electromotive force, the resistance, and the electrical current. The electromotive force is measured in volts, and is technically spoken of as voltage. The resistance is measured in ohms; the current in ampères, and it is spoken of as having a certain ampèrage. The relations that exist between these factors are expressed in Ohm's law, upon which, and the simple deductions therefrom, are based nearly all calculations in electrolytic work.

Ohm's law, briefly stated, is this: "The resulting electrical current is equal to the electromotive force divided by the resistance." In other words, the electromotive fôrce (electrical pressure created by the dynamo) causes the flow of the electrical current. The latter is directly proportional to the former. The resistance opposes the flow of the current. The latter is, therefore, inversely proportional to the former. The electrical power is the product of the number of volts multiplied by the number of ampères of the current. Its unit is called a watt.

From Ohm's law it follows that to double the resistance halves the current, the electromotive force remaining constant. Or if with the doubled resistance the current is to remain constant, the electromotive force, and therewith the power, must be doubled. This applied to electrolysis means that with the same current we can deposit an infinite quantity of copper, but that the power necessary must ever increase proportionally to the resistance.

Another important deduction from Ohm's law is the following: When through a given resistance the current is required to be doubled, the power must be increased four times; or in general, the resistance remaining constant the power increases proportionally to the square of the current. This applied to a practical example means that, if from a given number of depositing tanks we wish to double the output of copper by doubling the current, we must increase our power four times; or should we wish to quadruple the output the power must be increased 16 times. We therefore encounter the economic problem: Is it more advantageous to double, quadruple, etc., the capacity of an electrolytic plant as to number of tanks; or to increase four times, 16 times,

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* All analytical data given in this article, unless otherwise mentioned, are the writer's own. He takes pleasure, however, in acknowledging the courtesy of Mr. R. B. Keyser, vice-president, and of Messrs. A. L. Walker and W. H. Peirce, managers of the Baltimore Copper Smelting and Rolling Co., for contributing drawings, illustrations, and other matter, and also for the liberty extended to visit the various parts of their smelting and refining works.

etc., the power, in order to increase the output proportionally to the former? The first is a single investment; the second, a permanent expense.

The most important laws in direct relation to electrolysis are those of Faraday, who showed that a given quantity of current will always deposit the same quantity of a given element; and that the elements are deposited, or their compounds decomposed, proportionally to their equivalent weights, *e.g.*, hydrogen 1, oxygen 8, copper 31.7, silver 108, etc.

It has been demonstrated by careful experiment that one ampère of current will deposit 1.18656 g. of copper in one hour, which can be translated for technical purposes to mean that one ampère of current should deposit 1 oz. of copper in 24 hours. It is, therefore, an easy matter to keep a record of the efficiency of a plant, or to detect leakages of the current, by measuring the current, weighing the deposited copper, and comparing the latter with the theoretical quantity required. The quantity of current, *i.e.*, the number of ampères, flowing through a certain unit-surface is termed the density or intensity of the current. The greater the density of the current the greater will be the rate of deposition of the metal on a given surface. In order to deposit pure copper it is essential to have a constant current density. The copper is, however, deposited in satisfactory condition only when the current density is maintained within certain limits. To my knowledge a current density of 18 ampères per square foot of anode surface has not been exceeded in any of the electrolytic refineries in the United States. The more common density employed is 10 or 12 ampères per square foot.

Joule found that an electrical current, passing through a conductor, generates a certain quantity of heat, and he established what is known as Joule's law, namely: The quantity of heat developed in a conductor is proportional to the resistance of that conductor and to the square of the current density. When heat is thus produced in the conductors and the electrolyte it is a waste of power.

The copper plate in the depositing tank which receives the current, and is dissolved by the latter's action, is called the anode plate. The plate on which the deposition of the metal takes place is called the cathode plate, and the two are the electrodes. The solution in which they are immersed is the electrolyte.

In an electrolytic copper refinery the current must be adapted to the given condition of electrodes and tank arrangement, or the latter to a given current. For example, if with a given current, say 1,000 ampères, we wish a current density of 10 ampères per square foot of anode surface, and our anode plates have a surface of 10 sq. ft., it is necessary to let the current pass through 10 such plates connected in multiple. These would be placed in one tank with 11 cathode plates, through which the current would pass into another tank of the same description, and so on. Tanks are thus connected in series. As their number increases, the voltage necessary to carry the current through them must increase proportionally. With the increase of voltage the danger of loss of current through leakage increases. There must, therefore, be a limit to the number of tanks connected in series. The highest voltage in present practice is about 180.

The lower limit of a series is, of course, a single couple, one anode and one cathode, with an electrical pressure of a fraction of one volt, the exact value depending on the distance between the electrodes, the density of the current, the character of the electrolyte, and the composition of the anode. In actual practice the voltage between plates varies between about $\frac{1}{6}$ and $\frac{1}{3}$ volt, according to the system employed. The voltage necessary to decompose copper sulphate and to deposit the metal on the cathode is theoretically 1.16 volt. With a soluble anode this is counterbalanced by the energy evolved in the formation of copper sulphate on the anode surface.

Systems of Electrolytic Copper Refining.—There are employed in the United States two distinct systems of electrolytic copper refining, *i.e.*, the multiple and the series systems. The latter seems now confined to the modification introduced by Edward S. Hayden. The multiple system has often been described; it is the one originally introduced. The Hayden system, a later invention, has undoubtedly been much misrepresented, owing to the theoretical assumption that equal conditions regarding the anodes, etc., exist in the two systems. The distinct feature of the Hayden system is found in the character of the anode plates. They are prepared by rolling anode cakes into long sheets, about $\frac{1}{4}$ in. thick, and after cutting them into the desired lengths, two or more of the pieces are put together to form an anode, which are thus made very smooth and of uniform quality. They differ essentially, therefore, from those employed in the multiple system, which are cast from refined or blister copper, and are generally 1 or $1\frac{1}{2}$ in. thick. The dimensions employed for the anodes in the various multiple establishments do not vary widely. They are in round numbers 2 ft. wide by 3 ft. long. For the Hayden system they are about 2×2 ft.

The accompanying illustrations will be for the most part self-explanatory. Those pertaining to the Hayden system are taken from U. S. patent papers Nos. 377,487 and 459,838. Referring to Fig. 7, it should be said that two or more of the tanks are connected in multiple, and any desired number of these multiple groups may be joined in series. Fig. 4 shows a Hayden siphon. When the orifice at the top is open it serves as an overflow outlet, and is used as such in the circulation of the electrolyte. The flow, however, comes from the bottom of the tank. When the opening at the top is closed, true siphon-action is obtained, and it is made use of to empty the tank. Fig. 9 is the most widely used form of anode. Fig. 8 is the Thofern, and Fig. 10 the Peirce anode. Figs. 14 and 15 show wooden tanks with leaden lining and inner wooden protection, as well as different modes of placing the conductors and connecting them with the electrodes.

Having the difference in character and the arrangement of the anodes in the two systems in view, the other comparisons may be fairly made. The Hayden system requires a rolling mill which will consume about one-third of the power of the plant. It follows, of course, that the copper must be of such quality as to allow it to be rolled, and that the system is limited to the working of copper which conforms to this requirement. Blister copper, be it from converter or reverberatory furnace, cannot be rolled, and it must, therefore, undergo a

FIG. 4.

FIG. 4. SECTION OF SIPHON.

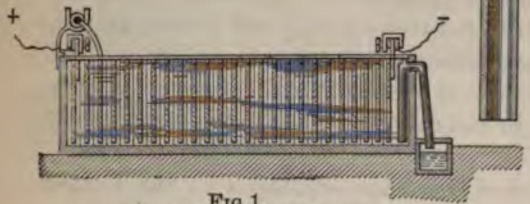

FIG. 1.
SECTION OF HAYDEN TANK.

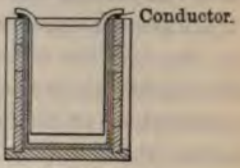

FIG. 14.

FIG. 2.
ENLARGED SECTION SHOWING ANODES WITH DEPOSITED COPPER.

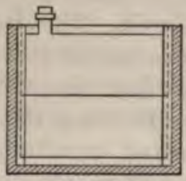

FIG. 3.
TRANSVERSE SECTION OF HAYDEN TANK SHOWING ANODES.

FIG. 5.
GROOVED SLIDES.

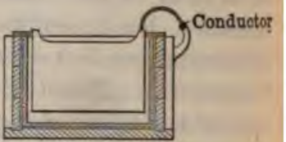

FIG. 15.

FIGS. 14. & 15. MODE OF SUSPENSION AND CONNECTION WITH ELECTRIC CURRENT.

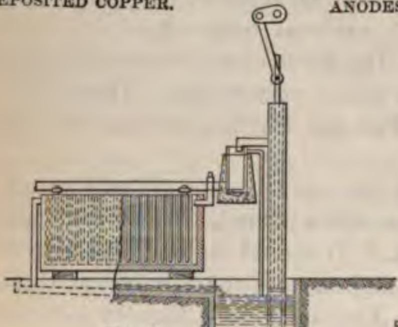

FIG. 6.

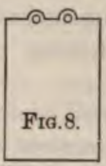

FIG. 8.

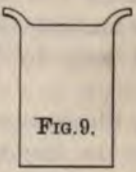

FIG. 9.

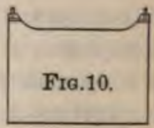

FIG. 10.

SOME FORMS OF MULTIPLE ANODES.

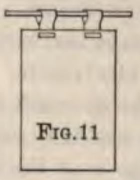

FIG. 11

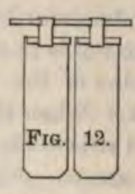

FIG. 12.

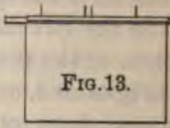

FIG. 13.

SOME FORMS OF MULTIPLE CATHODES.

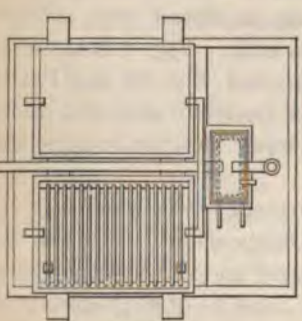

FIG. 7.

FIGS. 6. & 7. SECTION AND PLAN SHOWING PUMPING, HEATING, AND CIRCULATING ARRANGEMENT OF TANKS.

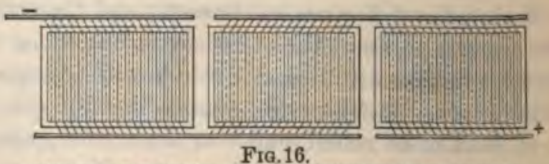

FIG. 16.
MULTIPLE TANKS SHOWING CONNECTION IN SERIES. THE WHOLE AND BROKEN LINES SHOW ALTERNATE ANODES AND CATHODES.

DETAILS OF ELECTROLYTIC APPARATUS.

process of refining. The copper produced from the ores of Butte City, Mont., by either process is well adapted for the rolling of Hayden anodes. The following figures show the impurity contents in such coppers :

Impurities in	Lead.	Bismuth.	Antimony.	Arsenic.	Se and Te.
	%	%	%	%	%
Converter copper.............	0·0082	0·0025	0·0443	0·0068	0·0071
Reverberatory copper.........	0·0093	0·0320	0·0651	0·0586	0·0098

Besides the elements above enumerated these refined coppers contain about 0·3% oxygen; probably a trace of iron; about 0·002% nickel; and about 100 oz. silver and 0·3 oz. gold per ton.

It has long been known that small quantities of lead added to copper will in many cases improve its quality for rolling. About 0·1% of lead added during the refining process seems to be the maximum quantity which exerts a beneficent influence.

The following figures show the amount of impurities in two samples of copper which would not roll until after the addition of lead:

	Lead.	Bismuth.	Antimony.	Arsenic.	Se and Te.	Silver.	Gold.
	%	%	%	%	%	Oz.	Oz.
I............	0·0069	0·0095	0·0602	0·0312	0·0527	156·30	0·48
II...........	Trace.	0·0055	0·0370	0·0255	0·0365	172·90	0·60

The following is the composition of two samples of copper which were not rollable, and with which the addition of lead proved ineffective:

	Lead.	Bismuth.	Antimony.	Arsenic.	Se and Te.	Silver.	Gold.
	%	%	%	%	%	Oz.	Oz.
I............	0·0558	0·0274	0·1245	0·1160	0·0153	100·60	0·45
II...........	0·0073	0·0340	0·1350	0·0582	0·1067	229·40	0·12

In the Hayden mill the rolls are grooved, and in rolling the cakes retain their original width throughout the several passages. When the copper is in good rolling condition only the ends of the sheets yield scrap. The amount of the latter, therefore, depends on the care taken in giving the cakes the proper size. It can, with careful management, be kept at 2 or 3%. This will be best understood by an example: Suppose we desire to have a cake from which a sheet is to be rolled and cut into 10 plates each 12 in. wide, 24 in. long, and ¼ in. thick, making 720 cu. in. After rolling, the ends of the sheet will be rounded and perhaps somewhat ragged, and must be cut square, so that 4 in. at either end, or an additional 24 cu. in. must be added to the cake for that purpose, the volume of which then would be 744 cu. in., would weigh 239 lb., and would produce 3·23% of scrap in the mill. The mill in addition produces a small percentage of suboxide scales which are taken care of with the slags from the refining furnaces.

The multiple system has the advantage of being applicable to any quality of copper; at least inasmuch as its anodes may be cast from copper, refined or blister, of any composition. However, refined copper seems to be now generally preferred. It is better than blister, because it yields a more uniform casting and a smoother surface, and consequently dissolves more evenly, and also produces much less scrap in the depositing tanks. The amount of this scrap depends, besides, somewhat on the shape of the anodes, as will be seen by the illustrations. The more copper remaining above the electrolyte the greater will be the amount of resulting scrap. The lowest figure for the latter is probably 10%. With anodes of blister copper and unfavorable shape the amount of scrap will reach 30%. This scrap requires remelting and recasting into anodes.

Since in the Hayden system one surface of the anode plate constitutes the cathode, and since the solution of the anode does not proceed with absolute evenness, it naturally follows that some parts of the anode copper will remain on the deposited cathode copper. Thus results a deterioration of the electrolytic copper by adhering anode scraps with an attendant loss of precious metals. It is, however, possible to remove all of the anode by the continued action of the current, but by doing so much of the deposited copper will become exposed, redissolved, and deposited a second time, which means waste of power. A sharp line cannot be drawn between these two disadvantages of the system. They go together, the cathode copper always carrying some anode scrap and some of the current being wasted in dissolving the former.

The vertical edges of the Hayden anodes, where protected by the wood of the grooved slides, remain intact to a certain extent. Here the cathode side should be coated with an insulating substance, so that the remaining strip of anode plate may be readily separated mechanically. The scrap resulting from this condition may amount to a small percentage.

While in the Hayden system only one cathode sheet is needed to each tank, the multiple system requires to n anodes $n+1$ cathode sheets. Formerly these were cut from rolled sheets of pure copper, but now they are generally made by electrolytic deposition in the regular tanks. Their production is, therefore, no extra expense. In some places these sheets are deposited on copper cathodes, the surface of which must be coated with paraffine and graphite, or the like, in order to enable the mechanical separation of the deposited sheet from the cathode; in others lead cathodes are used, which allow the sheet of deposited copper to be readily stripped without any special preparation of the surface. The edges of the cathode must naturally be covered with an insulating material in order to keep separate the sheets on the two surfaces of the cathode. The leaden cathodes seem to be the more advantageous, although with them is introduced an element of greater resistance than copper. The better conductivity of the latter is, however, very much offset by the coating which the character of its surface necessitates.

The cathode sheets are deposited to a thickness of 0·01 to 0·04 in. In some of the electrolytic refineries the whole anode is left to deposit on one cathode, while in others two cathodes are used; that is to say, when one-half of the anode has been deposited the cathode is removed, and a new one inserted for the deposition of the remainder of the anode. This is done to avoid excrescences on the cathode, caused by irregular and too long continued deposition and to obtain pure copper. The latter method naturally requires double tank capacity for the production of cathode sheets, as well as twice the amount of work to handle them in preparation, and in placing them in the tanks and in taking the deposited copper from the tanks to the refining furnaces.

The high voltage that exists between the end plates in a tank of the Hayden system has often been pointed out as the reason for a serious loss of current, which does not exist in the multiple system. This voltage, of course, depends on the number of plates which are placed in a tank. Supposing the pressure between plates to be ⅙ volt, and the number of plates 100, the pressure, or voltage, between the end plates would be 16⅔ volts. With 150 plates it would be 25 volts, etc.

The layer of electrolyte above and below the intermediate plates has a lower resistance than a corresponding space of electrolyte interrupted by the copper plates; hence there is a loss of current through the part of uninterrupted electrolyte, and it is of economic importance to reduce as much as possible the layer of liquor above and below the plates. With a reduction of space below, the constantly existing danger of short circuiting through the metallic residues at the bottom of the tank increases; and careful guard must be exercised not to fall from one source of loss into another equally great.

For the same reason, *i.e.*, high voltage, no lead lining may be used in the Hayden tanks, as it would at once result in a short circuit between the end plates. The Baltimore Copper Smelting and Rolling Co. uses heavy slate plates coated with tar for its Hayden tanks. The multiple system plants use wooden tanks, tarred or lead-lined. When the latter is the case it is best to have the lead protected against mechanical wear by an inner wooden lining.

Since the Hayden system employs a current of higher voltage and lower ampèrage than the multiple system, the latter requires conductors of greater cross section from the dynamos to the tanks, and from tank to tank; otherwise it will have a greater loss of power through the heating of the conductors; for from Joule's law we know that the loss of power in a given conductor increases with the square of the current density.

One of the essential differences between the Hayden and multiple systems consists in the difference of space that separates the electrodes. In the Hayden system this distance has been reduced to ⅝ in., while in the multiple refineries it still varies from 1½ to 2½ in. This reduction of resistance in the former system is a great point in its favor, which is, however, somewhat offset by the greater current density employed. To show why it is possible to place the electrodes so much closer in the Hayden than in the multiple system, it is necessary to recall that the Hayden anode plates, being rolled, have a perfectly smooth surface; are smaller in area than in the multiple arrangement; are fixed in straight-cut, grooved slides, and can, therefore, be closely set with all points an equal distance apart, thus insuring a uniform solution and deposition. The multiple system anode plates, being suspended from two points at their top or sides, often do not hang perpendicularly, leading to varying distances between them. Being cast they are not uniform. Both conditions are conducive to irregular solution and deposition, and consequent excrescences and short circuiting. This latter trouble affects only the two plates which become connected in the Hayden system, while in the multiple the whole tank becomes a sufferer. For that reason it is very important to guard against this trouble, and to be able to locate it whenever it happens. At the Anaconda Co.'s works, in Montana, an automatic short circuit recording device of ingenious construction has been put in operation. When the metallic connection between the electrodes and the main conductor is of large resistance compared to the latter, the short circuit may readily be discovered by the touch of the hand, as these connections soon become more heated than the others. The voltmeter is always a sure finder of the contact.

For the identical reasons that the Hayden plates permit of being set so closely, they allow the use of a far greater current density, which, although in itself meaning a waste of power, is the factor necessary to a more rapid turning over of the copper, therewith decreasing the amount of that metal tied up in the tanks. In actual practice the last-named points are much in favor of the Hayden system. A well-managed plant of that system is in fact a very small, or rather a very condensed, affair compared to a multiple plant of corresponding capacity.

I am not in a position to make a comparison of costs between refineries of the two systems, with equality of conditions as to management, labor, and power. It is to be pointed out, however, that the two largest electrolytic refineries in this country are run on the two rival principles, that of the Anaconda Mining Co., at Anaconda, Mont., being of the multiple system, while the new refinery of the Baltimore Copper Smelting and Rolling Co. at Baltimore, Md., has the series system of Hayden. Both are undoubtedly model plants. Each has an actual output of from 60,000,000 to 70,000,000 lb. of electrolytic copper per annum.

The Anaconda Mining Co. has recently made public the cost of electrolytic refining during the year 1897–98 at its works at Anaconda, Mont., as follows: Assay expenses and salaries, $14,699·45; electric light, $2,522·87; expense, $3,723·88; labor, $136,361·57; legal expenses, $2,046·50. Power house expenses: Coal, $132,427·39; labor, $47,429·56; machinery and repairs, $19,300·70; sundry supplies, $6,341·51—making a total of $205,499·16 for power, of which $6,000 was charged to silver mill, leaving a net expense of $199,499·16 for power for the electrolytic refinery; for recasting scrap anodes, $17,209·12; regenerating plant expenses, $11,809·39; salaries, including management, $15,611·99; stable expenses, $1,881·00; sundry supplies, $11,341·56; switching product, $870·00; taxes, $5,502·62; repairs on buildings, $5,166·82; on machinery, $7,302·63; on refining tanks, $7,963·71—making a total of $443,512·27. The output of electrolytic copper was 59,505,711 lb.; consequently the cost of production was 0·75c. per lb., or $15 per ton.

Composition of the Electrolyte.—The electrolytes used in various refineries do not, as far as can be learned, differ much in their general composition. The usual prescription seems to call for about 16% copper sulphate (bluestone) and 6% free sulphuric acid. It has been shown that a saturated copper sulphate solution presents the greatest conductivity; this, however, does not hold true in the presence of free sulphuric acid, and constant efforts must be made to prevent the electrolyte from being enriched with copper sulphate, since crystallization is another element of great disturbance. Two remedies are in use for these troubles: The periodical replacing of part of the electrolyte by water and acid, or the precipitation of the copper electrolytically therefrom in tanks with insoluble anodes, or in the series system with an insoluble anode end plate in the regular depositing tanks. With sulphuric acid it has been found that a 25% solution is the best conductor. More acid solutions than prescribed by the old formula have undoubtedly been used with good results. With too much acid, however, polarization ensues from its own decomposition by the action of the current. The latter trouble is said to set in when the copper sulphate solution has reached an acidity of about 13%, when there is a waste of electrical energy.

The following figures show the composition of an anode copper and an actual working solutions:

I. Impurities in anode copper: Lead, 0·0103%; bismuth, 0·0040%; antimony, 0·0630%; arsenic, 0·0211%; selenium and tellurium, 0·0072%.

II. Partial composition of electrolyte: Copper sulphate ($CuSO_4$, $5H_2O$), 16·3%; free sulphuric acid (H_2SO_4), 5·78%; bismuth, 0·0008%; antimony, 0·022%; arsenic, 0·035%.

Besides the elements which were determined, there are always small amounts of iron and nickel present in the electrolyte. They are never deposited with the copper in acid solutions.

Much more impure solutions than the above are successfully used. The subjoined results of analyses show the amount of impurities in an anode copper and an electrolyte after a 47 days' run. Some years ago when these analyses were made the electrolyte would be considered foul, and would be renewed or purified.

Impurities in	Bismuth.	Antimony.	Arsenic.
	%	%	%
Anode copper	0·0320	0·0651	0·0586
Electrolyte	0·0174	0·0555	0·0362

Much experimenting has been done for the purpose of finding means to maintain the electrolyte solutions within a certain degree of purity. Of some of the methods that have become public the following may be mentioned: The addition of small quantities of tin to the anode copper, for the purpose of rendering the arsenic insoluble; letting the solutions percolate through metallic copper (shot) or cuprous oxide to precipitate the impurities, the addition of common salt, the injection of air, etc. Where such means have failed or proved insufficient the whole or part of the solution is renewed at intervals, the discarded solution being treated for the regeneration of the copper sulphate.

For that purpose the solutions which show a strength of about 16° must be concentrated by evaporation to 45°. On cooling 85 to 90% of the copper sulphate is obtained by crystallization, and the remainder is precipitated as metal with scrap iron.

The copper sulphate obtained from the electrolyte is too impure to be used again for its original purpose, and must be recrystallized. Below is given an analysis of a sample of dried, impure crystals, and the percentage the impurities would constitute in a solution prepared from it. The latter values are obtained by dividing those of the crystals by 8, this bringing the copper to the proper percentage as contained in an electrolyte.

Composition of	Copper.	Iron.	Nickel.	Lead.	Bismuth.	Antimony.	Arsenic.
	%	%	%	%	%	%	%
Sulphate crystals.........	32·78	0·589	0·0496	0·0122	0·0640	0·2920	0·2470
Contents of electrolyte...	4·10	0·0736	0·0062	0·0015	0·0080	0·0365	0·0309

At many refineries it has been found advantageous to heat the electrolyte as much as the tank material, or its lining, will allow. This is because heat is the most efficient means for lowering the resistance. Furthermore, excrescence seem less frequent, and in general the copper is smoother from warm than it is from cold solution.

That the electrolyte must be kept in circulation to remain uniform in density and acidity was recognized by the founder of the commercial process of electro-deposition of copper, James B. Elkington, of England. The rapidity of this circulation must necessarily be proportional to the current density. If it be found necessary to circulate the whole solution once in six hours when 10 ampères are used per square foot, it will undoubtedly be necessary to have it circulate once in four hours when 15 ampères are used.

Composition of the Residues.—The electro-refining of copper has a double purpose: It is to separate and save the precious metals; and to make pure commercial copper. Probably no copper containing less than 30 oz. silver per ton is electrolyzed at present in the United States. In the process of electrolysis gold is not dissolved from the anode plate; neither are silver, selenium, and tellurium to any appreciable extent when present in such small quantities as in the grades of copper here under consideration. They, with the elements which are partially insoluble, drop to the bottom of the tanks as the anode is being gradually dissolved, and form there a sediment or residue, generally called slimes or mud.

When the precious metals and impurities reach a certain limit in the anode copper they become an annoying factor in the operation of the refinery. They remain wholly or partially adherent to the anode, seriously increasing the resistance, and contaminating the electrolyte and the deposited copper, also leading to loss of silver and gold. In such cases scrubbing of the anodes must be resorted to, which is expensive and dirty work. Wherever feasible it is undoubtedly far more economical to dilute such copper with lower grades to a proper working condition.

Subjoined are two analyses of copper, too impure for good work, from which the residues must be mechanically separated:

	Lead.	Bismuth.	Antimony.	Arsenic.	Se and Te.	Silver.	Gold.
	%	%	%	%	%	Oz.	Oz.
I.	0·0028	0·0208	0·3691	1·3200	0·0326	349·90	5·46
II.	0·2990	....	0·1940	0·1740	0·0117	320·70	3·14

The treatment of the residues and the ultimate refining of the silver and gold is an industry of its own. It is, however, at present generally a department of the copper refinery. Following are the results of partial analyses of two samples of residues and corresponding anode copper, I. being a reverberatory, and II. a converter copper:

	Ag.	Au.	Cu.	Pb.	Bi.	Sb.	As.	Se.	Te.	Fe.	SO_4.	H_2O.(*a*)
	%	%	%	%	%	%	%	%	%	%	%	%
I. Residues..............	53·894 (*b*)	0·2959 (*c*)	11·010	0·910	3·930	6·250	2·107	0·394	1·174	......	5·268	2·365
I. Anodes..............	0·3432 (*d*)	(*e*)	99·30	0·0093	0·0320	0·0651	0·0586	0·0025	0·0075	......	......	......
II. Residues..............	55·150 (*f*)	0·198 (*g*)	13·820	2·070	0·340	2·440	1·090	0·718	0·892	0·800	10·680	2·604
II. Anodes..............	0·3444 (*h*)	....	99·40	......	0·0035	0·0510	0·0180	0·0045	0·0056	......	......	......

(*a*) The samples were dried at 250° C.; hydrate-water and oxygen were not determined; (*b*) Equivalent to 15,718·7 oz.; (*c*) 86·3 oz.; (*d*) 100·1 oz.; (*e*) 0·51 oz.; (*f*) 16085·04 oz.; (*g*) 57·749 oz.; (*h*) 100·47 oz.

By a simple arithmetical calculation the proportion of the partially soluble elements in the original anode, gone into the residues and into solution, is found. The results are as follows:

	Cu.	Bi.	Sb.	As.	Ag.	Au.	Se.	Te.
	%	%	%	%	%	%	%	%
I. In residue..............	0·07	78·22	61·14	22·90	100	100	100	100
I. In solution..............	99·93	21·78	38·86	77·10	....	....	....	....
II. In residue..............	0·086	60·71	29·90	37·84	100	100	100	100
II. In solution..............	99·914	39·29	70·10	62·16	....	....	....	....

It will be seen that in I., where the quantities of bismuth, antimony and arsenic in the anodes are not very widely different, bismuth shows the least solubility and arsenic the greatest. In II., where antimony greatly preponderates in the anodes, that element shows the greatest solubility. The copper that has gone into solution is, of course, deposited on the cathode; but with it have been carried only small percentages of the other metals and metalloids.

The residues at the present day are treated solely for their silver, gold, and copper contents. Everything else is waste, including the selenium and tellurium contained in them. The residues would be a convenient source for these substances if there were any use for them. The copper produced at Butte alone would yield annually about 20,000 lb. of the two elements.

Behavior of Impurities in Anode Copper.—When metallic copper is dissolved in nitric acid, then converted into sulphate, and the solution subjected to the action of the electric current, all copper is deposited, using an insoluble anode, as is done in the electrolytic determination of copper, until the impurities, which now are all in solution, will be found deposited with the copper in various degrees. Subjoined are given the results of analyses of the converter copper and of the corresponding electrolytic copper; and also the percentage of the original impurities deposited:

	Lead.	Bismuth. (b)	Antimony.	Arsenic.	Se and Te.
	%	%	%	%	%
Impurities in converter copper.	0·9103	0·0040	0·0630	0·0211	0·0072
Impurities in electrolytic copper	Trace.	0·0037	0·0031	0·0009	0·0016
Impurities deposited (a).......	(?)	92·50(?100)	4·92	4·26	22·22

(a) Gold in this case was not dissolved, and silver is always completely deposited. (b) Bismuth is undoubtedly all deposited; the difference between the two analyses being well within the limits of error.

From the above figures it will be seen that of the impurities contained in the converter copper, when once in solution, bismuth is most easily deposited. When we have the converter copper as a soluble anode instead of in solution, the conditions are materially changed. What is most readily deposited from solution is now most difficult to dissolve from the anode. Selenium and tellurium do not strictly follow this rule, since they are not dissolved on the anode.

That arsenic, antimony, and bismuth dissolve in the order named when present in approximately equal quantities has already been pointed out. The exact behavior of lead is difficult to trace, because besides coming from the anode copper, it has also its source in the leaden lining of tanks and in the commercial sulphuric acid. Little of it, however, goes into solution, it being converted into insoluble peroxide on the anode and into sulphate by the electrolyte.

The most important factor in the production of pure copper in the presence of impurities is the influence of the preponderating mass of an element. That is to say, the electrical current will not deposit that element first whose compounds are most easily decomposed when another element which is more difficult to decompose vastly exceeds the former in quantity. This, however, only holds good for elements which stand close to each other in the voltaic series. A large excess of copper over the impurities in solution is, for that reason, a cardinal requirement for the production of pure copper by electrolysis. When the impurities are allowed to accumulate in the electrolyte there will come a critical point when they, or some of them, will suddenly begin to be deposited along with the copper. The practiced eye will be able to recognize that point by the appearance of dark streaks, and thus the fouling of the copper may be prevented. Absolutely pure copper is never produced on a commercial scale.

Some of the impurities in electrolytic copper may have their origin in electrolytic deposition from the electrolyte; others may have been mechanically carried there by finely divided and suspended residues. In the Hayden system, there are, in addition to the above-named sources, the small quantities of the anode copper remaining adherent to the cathode. In order to be able to apportion the impurities in electrolytic copper to their proper sources it is necessary to assume that the silver is all mechanically deposited, which, with the coppers under consideration, is undoubtedly practically true.

Composition of Anode and Cathode Copper.—In the following table are given the average results of analyses of a number of samples of anode copper, of Hayden electrolytic copper as actually put on the market, and of the same kind of copper carefully freed from all particles of adhering anode copper:

Kind of Copper.	Lead.	Bismuth,	Antimony.	Arsenic.	Se and Te.	Silver.
	%	%	%	%	%	Oz.
Anode (converter).....................	0·01030	0·00400	0·06300	0·02110	0·0072	100·00
Cathode (commercial).................	0·00047	0·00018	0·00136	0·00059	....	1·11
Cathode (freed from all scrap)........	0·00055	....	0·00094	0·00025	....	0·36

From the above figures it follows that of the total silver contents in that commercial electrolytic copper 0·75 oz. per ton is due to anode scrap adhering to it. This silver and the corresponding impurities from the anodes are, therefore, 0·75% (the silver in the anodes being 100 oz.) of their contents in the anodes; 0·36 oz. of the silver and a corresponding amount of impurities are derived from the residues. By difference we find the amount of impurities deposited from solution. The subjoined table shows the results of the calculations as to the origin of the impurities in Hayden cathode copper:

Impurities from	Lead.	Bismuth.	Antimony.	Arsenic.	Se and Te.	Silver.
	%	%	%	%	%	%
Anode scrap...........................	0·0000773	0·0000300	0·0004725	0·0001583	0·0000540	0·75
Residues	0·0000371	0·0000086	0·0000684	0·0000302	0·0000260	0·36
Solution..................................	0·0003556	0·0001414	0·0008191	0·0004015	....	....

If these minute quantities of the impurities be expressed in percentage of their respective total quantity in the electrolytic copper, they will perhaps appear more intelligible. The subjoined table shows the origin of impurities in Hayden cathode copper and percentage of total quantity present:

Impurities from	Lead.	Bismuth.	Antimony.	Arsenic.	Se and Te.	Silver.
	%	%	%	%	%	%
Anode scrap...........................	17	17	35	27	67·5	67·5
Residues	8	5	5	5	32·5	32·5
Solution..................................	75	78	60	68	0·0	0·0

The calculations for electrolytic copper from the multiple system are somewhat simpler, since no scrap must be considered. In the tables below are given the analyses and calculations of the anode copper as above, and an impure electrolytic copper of the multiple system:

Kind of Copper.	Lead.	Bismuth.	Antimony.	Arsenic.	Se and Te.	Silver.
	%	%	%	%	%	Oz.
Anode (converter).....................	0·01030	0·0040	0·0630	0·02110	0·0072	100·00
Cathode (multiple)....................	0·00231	0·0005	0·0138	0·00218	....	0·54

Source of Impurities in Cathode Copper.	Lead.	Bismuth.	Antimony.	Arsenic.	Se and Te.	Silver.
	%	%	%	%	%	Oz.
From residues	0.0000556	0.0000130	0.0001021	0.0000456	0.0000389	0.54
From solution	0.0022544	0.0004700	0.0036979	0.0021344	....	....

Proportionate Source of Impurities.	Lead.	Bismuth.	Antimony.	Arsenic.	Se and Te.	Silver.
	%	%	%	%	%	%
From residues	2	2·6	0·7	2	100	100
From solution	98	97·4	99·3	98	....	....

From the figures above given it is clear that by far the greater part of the contamination of the electrolytic copper is due to electrolytic deposition of impurities held in solution by the electrolyte.

The table next subjoined shows the composition of three cathode plates from the same tank (Hayden system). One of the plates was placed much farther from the anode than were the others, thus creating a different voltage; but the latter seems to have had no marked influence on the deposition of the impurities. The samples were carefully freed of all adhering scrap. The composition is, therefore, that of the deposited copper.

Cathode.	Lead.	Bismuth.	Antimony.	Arsenic.	Silver.
	%	%	%	%	Oz.
⅝ in. from anode	0·00110	....	0·00115	0·00019	0·48
3 in. from anode	0·00054	....	0·00102	0·00034	0·43
⅝ in. from anode	....	....	0·00065	0·00022	0·18

The purest and the impurest samples of electrolytic copper which have come under my observation had the following composition:

Description.	Lead.	Bismuth.	Antimony.	Arsenic.	Oxygen.	Silver.
	%	%	%	%	%	Oz.
Pure wirebar (tough)	....	....	0·00065	0·00022	0·01	0·18
Impure ingot (brittle)	Trace.	0·0017	0·02730	0·01040	0·05	0·17

The percentage of the impurities in the anode copper that has gone into the last described cathode coppers compares as follows:

	Lead.	Bismuth.	Antimony.	Arsenic.	Silver.
	%	%	%	%	%
In pure wirebar	(?)	(?)	1	1	0·18
In impure ingot	(?)	42·5	43	49	0·17

From what has previously been said it follows, however, that there can be no direct relation between the composition of the anode copper and that of the cathode copper. The purity of the latter depends entirely on the character and composition of the electrolyte. In the subjoined table are given the results of analyses and physical tests of converter, electrolytic, and Lake copper:

ANALYSES AND TESTS OF VARIOUS COPPERS.

Kind of Copper.	Lead.	Bismuth.	Antimony.	Arsenic.	Silver.	Sub-oxide.	Conductivity, Hard Drawn.	Conductivity, Soft Drawn.	Tensile Strength.
	%	%	%	%	Oz.	%	%	%	Pr. Sq. In.
Converter copper..............	0·00850	0·0035	0·0254	0·0290	92·00	2·07	84·70	87·20	40800
Cathode plate (not melted)....	0·00054	....	0·00102	0·00034	0·43	0·084	100·64	103·14	59800
	....	....	0·00065	0·00022	0·18	0·090	100·70	103·20	60400
Conductivity or wirebar copper, refined (Hayden)........	0·00020	....	0·00150	0·00046	1·49	0·60	98·73	101·23	61900
	0·00020	0·00030	0·00110	....	1·17	1·05	98·46	100·96	61000
	0·00165	....	0·00156	0·00110	1·06	0·66	98·70	101·20	60000
	0·00020	....	0·00098	0·00027	1·08	0·84	98·59	101·09	60200
	Trace.	Trace.	0·00110	0·00070	0·99	0·81	97·72	100·22	60600
	0·00058	0·00074	0·00190	0·00100	0·88	0·75	97·00	99·50	60000
Conductivity or wirebar copper, refined (multiple).......	Trace.	....	0·00730	0·00020	0·30	0·87	98·41	100·91	63000
	0·00116	....	0·00171	0·00031	0·19	0·42	98·00	100·50	61400
	Trace.	....	0·00400	0·00030	0·27	0·90	97·78	100·28	59700
	0·00023	....	0·00340	0·00042	0·31	0·75	97·19	99·69	61500
	0·00104	....	0·00121	0·00055	0·47	0·66	97·18	99·68	60400
	0·00035	0·00040	0·00400	0·00080	0·15	0·84	96·92	99·42	61000
Rolling or cake copper, refined (multiple)..............	0·00143	....	0·00547	0·00180	0·20	0·73	96·16	98·66	61800
	Trace.	....	0·00790	0·00030	0·27	1·02	95·88	98·38	60400
Casting or ingot copper, refined (multiple)...............	Trace.	0·00140	0·01770	0·00220	0·23	0·78	94·80	97·30	61100
	0·00231	0·00050	0·01380	0·00218	0·54	0·63	93·85	96·35	62800
	0·00160	....	0·01430	0·00510	0·19	0·99	93·35	95·85	59300
Lake copper wirebar..........	0·00140	....	(*a*)	0·00370	8·70	1·11	95·47	97·97	61200
	0·00060	....	(*a*)	0·00310	9·68	0·90	96·77	99·27	60200
	0·00065	....	0·00083	0·00520	11·78	0·60	97·00	99·50	57700
	[b] 0·00442	0·0	0·00059	0·00066	21·33	1·20	96·70	99·20	58100
Lake copper, ingot (*b*).........	Trace.	0·0	0·00082	0·00730	7·43	0·74	98·80	101·30	58500
	Trace.	0·0	0·00095	0·00013	17·60	0·50	97·20	99·50	60600
Wirebar (origin ?) (*b*)...........	0·00704	0·00163	0·02340	0·02574	3·00	1·29	87·90	90·40	51400

(*a*) These samples were not tested for antimony.

(*b*) Analysis by John Johns. This bar, taken from the open market, shows what unfortunate attempts have been made at wirebars.

For the physical tests of the samples, as shown in the accompanying table, I am indebted to A. L. Walker, manager of the Baltimore Copper Smelting and Rolling Co.'s large Hayden plant. The conductivity tests were made on hard-drawn wire. The figures for soft-drawn, or annealed wire, are derived by merely adding 2·5% to the figures found in the actual tests; this being, according to Mr. Walker's experience, the difference in the two methods. This corresponds well with the results of others. The Messrs. Roebling, for instance, make their conductivity tests on soft-drawn wire, and in their *Hand-Book of Tables for Electrical Engineers* F. A. C. Perrine gives the average conductivity of 682 Lake copper samples as 99%, with which figure those given in the above table are in perfect accord.

The Mathiessen standard, by which in technical practice the conductivity, or rather in the first place the resistance of copper to the electrical current, is measured, is to a certain extent an arbitrary figure. The standard resistance of a copper wire 0·001 in. in diameter and 1 ft. long when soft drawn is 9·720 B. A. U., or 9·612 legal ohms at 0° C. The standard resistance of a copper wire 1 mm. in diameter and 1 meter long is 0·02057 B. A. U., or 0·02034 legal ohm. Much of the electrolytic wirebar copper when soft drawn shows a smaller resistance than the standard, and its conductivity is, therefore, over 100%.

The analytical figures given above for various samples of electrolytic copper show a wide range of purity. The different purposes for which such copper is required demand that the various grades be carefully kept apart by the refiner. To that end it is necessary that all be subjected to careful chemical analyses and physical tests. The latter are the most important and most simple to perform.

When, however, the relation between chemical composition and the physical properties, especially the conductivity, has been established by a definite method* of analysis, the determination of antimony in our case becomes a safe chemical guide for the selection of the various grades of copper. When with the mode of analysis employed by the writer and others 0·006 to 0·008% of antimony is found, the desired conductivity of 99% of annealed wire is no longer a certainty. An antimony determination on 100 g. or more of copper can readily be performed inside of two days.

The electrolytic refining of copper has now arrived at such a degree of perfection that Marcus Daly, in his last annual report (1897–98) on the working of the Anaconda Mining Co.'s plant, is enabled to make the statement that over 99·9% of their electrolytic copper conformed to the highest standard.

In the table above presented it is shown that the electrolytic copper which has not been melted and refined is superior in conductivity to the refined and cast copper by over 2%. There is, therefore, a field for improvement in refining, or in making wire from electrolytic copper without a previous melting.

PART II. THE REFINING.

Copper is, from a metallurgical point of view, of fourfold origin: (1) Metallic, or native, as found at Lake Superior, which is of remarkable purity, and requires no other treatment than the ordinary refining and casting for commercial purposes; (2) metal produced from sulphides by several more or less complicated methods; (3) metal from oxidized ores, as produced by reduction in a blast furnace; and (4) metal obtained from oxidized ores by leaching and subsequent precipitation.

Comparison of Various Methods.—The processes employed to produce metallic copper from copper sulphides are the reverberatory, the converter, and the Nicholls & James, the last consisting in the fusion of copper oxides and sulphides, and being in use only at the Cape Copper Co.'s works at Britton Ferry, England. The three processes produce from the same mattes a copper of very differing purity. The accompanying table, the results of which were obtained by experiments and chemical analyses on an extensive scale, shows the merits of the reverberatory process as compared to different converters, with respect to the purity of the copper produced:

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* It is necessary to emphasize the necessity of a definite method, because results by different methods and chemists may differ widely. Such differences are, in fact, sometimes surprising on such simple determinations as those of copper and silver. On bismuth, antimony, tellurium, etc., they would undoubtedly be incredible. Uniform methods are as desirable in the copper industry as they are in that of iron and steel.

Process	Element.	Reverberatory Process. Baltimore.	Stalmann Converter, Anaconda.	New Anaconda Converter.	B. & M. Co. Converter, Great Falls, Mont.	M. O. P. Co. Converter, Butte, Mont.
Capacity			Initial Charge, 8,0000 Lb. Maximum Charge. 9,000 Lb.	Initial Charge, 7,000 Lb. Maximum Charge, 17,000 Lb.	Initial Charge, 10,000 Lb. Maximum Charge, 22,000 Lb.	Initial Charge, 2,560. Lb. Maximum Charge, 9,000 Lb.
		%	%	%	%	%
Matte	Pb.	0·590	0·590	0·5680	0·0738	1·2523
	Bi.	0·042	0·042	0·0501	0·0337	0·0418
	Sb.	0·079	0·079	0·1010	0·1010	0·0950
	As.	0·045	0·045	0·0481	0·0480	0·0634
	Se, Te.	0·015	0·015	0·0101	0·0021	0·0085
Copper	Pb.	0·0093	0·0082	0·0103	0·0069	0·0517
	Bi.	0·0320.	0·0025	0·0040	0·0029	0·0051
	Sb.	0·0651	0·0443	0·0630	0·0546	0·0533
	As.	0·0586	0·0068	0·0211	0·0156	0·0231
	Se, Te.	0·0098	0·0071	0·0072	0·0034	0·0078
Elimination	Pb.	99	99	99	95	98
	Bi.	54	96	95	96	94
	Sb.	50	66	62	73	71
	As.	21	91	73	84	81
	Se, Te.	60	71	57	19	52
Remarks	....	....	Matte remelted in cupola, Refined copper.		Matte from matting-furnace directly to converter. Blister copper.	

By laboratory experiments with the Nicholls & James' process the results were found to differ materially from those of the reverberatory and converter. Selenium was found to remain with the copper only in traces; of the lead, but a small percentage; of the antimony, from 30 to 40%; of the bismuth, from 60 to 70%; of the arsenic, nearly the whole; and tellurium was found not to be eliminated at all.

By determinations with a campaign of a blast furnace, in which highly cupriferous slag, taking the place of oxidized ore, was reduced, the elimination of the several elements was found to be as follows: Lead, 41%; bismuth, 21%; antimony, 48%; arsenic, 52%; selenium and tellurium, none.

The wet or leaching process is known to yield comparatively pure copper. Exact figures, however, of its working cannot be given.

From the foregoing it follows that the relative position of each element as to its degree of elimination from copper mattes or ores is different in each process. These positions are demonstrated in the subjoined table:

Reverberatory	Pb.	Se, Te.	Bi.	Sb.	As.
Converter	Bi.	Pb.	As.	Sb.	Se, Te.
Nicholls & James'	Se.	Pb.	Sb.	Bi.	As, Te.
Blast furnace	As.	Sb.	Pb.	Bi.	Se, Te.

Refining Black or Blister Copper.—All copper directly derived from matte or ores is known as blister or black copper, which cannot be worked, or even satisfactorily cast, without undergoing a refining treatment. This furnace refining is described in detail in every treatise on the metallurgy of copper. Up to a few years ago it had for generations been conducted practically without improvement. Its chemistry is still the same as of old, and the chemical changes in the operation are secured by the old methods.

The first operation in refining consists in subjecting the molten copper to an oxidation and slagging process, which is effected by beating and splashing the metal so as to bring a larger surface in contact with the atmospheric oxygen.

This "flapping," as it is called, is done by hand labor, and is a tedious and expensive operation; seeming entirely out of date in this age of mechanical appliances, converters, tilting furnaces, compressed air, etc. The same is true of the second part of the operation. After the slag is skimmed the copper is subjected to a reducing process. It is covered with charcoal, and by the immersion of poles of green wood is brought into ebullition. The whole mass of the furnace charge is thus brought in contact with the charcoal, the hydrocarbon gases, produced by the decomposition of the wood, also aiding the reduction. It would seem that for this second part of the refining process a blast of pure reducing gas would be far more effective.

In defense of the copper metallurgist for the long stagnancy of his art, it must be stated that in contrast with the iron and steel metallurgist, whose methods are undoubtedly more perfect, he has had to deal with very much smaller quantities, and with more limited resources, which has naturally constrained him to the retention of more primitive methods. The main reason of the former non-progressiveness in the methods of this branch of practical metallurgy, however, is to be ascribed to the over conservative policy of the smelting and refining companies, which would not seek educated technical talent, but adhered to rule of thumb management.

Extent to which Impurities can be Eliminated.—The statement has often appeared in print that the refining of copper is undertaken to remove the last traces of impurities. Nothing is more erroneous. One or more of the elements, such as sulphur, zinc, iron, etc., may be removed entirely; but the most injurious elements, such as bismuth, antimony, arsenic, selenium, and tellurium, are always only partially eliminated. The exact part of the original quantity of those elements so removed has never been determined, on account of the difficulty of sampling the copper before being charged to the furnace. The amount of the portion slagged has, however, been accurately ascertained, and these quantities expressed in percentage of those contained in the refined copper give a fair idea of the efficiency of the operation. The following figures present the results of a few such determinations:

Contents of	Copper.	Lead.	Bismuth.	Antimony.	Arsenic.	Se and Te.
	%	%	%	%	%	%
Refined converter copper		0·0103	0·0040	0·0630	0·0211	0·0072
Refinery slag	44·47	0·5936	0·0020	0·2044	0·0490	0·0026
Relative slagability (*a*)	1	129·50	1·13	7·30	5·22	0·81
Refined electrolytic copper (*b*)		0·00093		0·00164	0·00028	
Refinery slag	45·00	0·0173		0·0073	0·00089	
Relative slagability	1	41		9·97	7·13	

(*a*) Calculated from the composition of the slag in which the copper contents were 1·45% of the refined copper.
(*b*) An electrolytic copper of which 0·6% was slagged.

From these figures it clearly follows that with each percentage of copper slagged a large part of the lead is eliminated, while antimony and arsenic are removed to a much smaller extent. Bismuth, selenium, and tellurium are taken up by the slag in about the same proportion as copper. Varying quantities of impurities, large or small, in the original copper do not seem to affect materially the relations demonstrated above. These seem to prove beyond doubt that it is impossible to remove by this process all impurities from copper. The production of conductivity, or wirebar, copper from impure copper must, therefore, be accomplished entirely by electrolytic separation. If the cathode copper itself is not fit for that purpose, refining will only slightly alter its grade.

What might be attained as to elimination of impurities in a converter, by slightly overblowing the charge of metallic copper produced in the ordinary way, the following data will clearly establish:

Impurities	Lead.	Bismuth.	Antimony.	Arsenic.	Se and Te.
	%	%	%	%	%
In regularly blown charge....	0·0069	0·0029	0·0546	0·0156	0·0084
In overblown charge...........	0·0016	0·0015	0·0195	0·0072	0·0033
Eliminated by overblowing....	77	48	64	54	3

A few minutes of overblowing are undoubtedly more effective than hours of "flapping."

Slags which are produced by the refining of blister, as well as of electrolytic copper, contain generally about 45% of copper. In works where matte smelting is carried on these slags are readily reduced in the blister furnace together with the copper sulphide (regulus). Where this is not possible they must be treated in the blast furnace, there yielding a black copper. When the original copper is argentiferous, the resulting black copper will contain about 60% of the silver contained in the former, and therefore must also be electrolyzed.

Since the impurities are concentrated considerably in these slags, and some of them also in the resulting black copper, it follows that slags derived from the purest electrolytic copper will not again yield a sufficiently pure metal for conductivity, or rolling purposes, even with the most careful refining. The following table shows the composition of the products of refining argentiferous converter copper:

	SiO_2.	FeO.	Cu.	Fe.	Al_2O_3	CaO.	Pb.	Bi.	Sb.	As.	Se & Te.	S.	O.	Ag.
	%	%	%	%	%	%	%	%	%	%	%	%	%	%
Refined converter copper..........	......	......	99·25	......	......	......	0·0103	0·0044	0·0630	0·0211	0·0082	......	0·284	0·36 (a
Refinery slag metallics (1 part)...	......	......	98·95	......	......	......	0·0293	0·0044	0·0680	0·0490	0·0101	......	......	0·375 (b)
Refinery slag proper (10 parts).....	39·02	......	35·66	8·21	4·19	5·04	0·6500	0·0018	0·2180	0·0490	0·0018	0·61	6·01	0·05 (c)
Average of slag proper and metallics...........	......	......	44·47	......	......	......	0·5936	0·0020	0·2044	0·0490	0·0026	......	......	0·079 (d)
Fluedust..........	......	......	92·60	......	......	......	0·0040	0·0114	0·0650	0·0470	0·0064	......	......	....
Black copper......	......	......	97·70	0·17	......	......	0·7800	0·0035	0·2380	0.0520	0·0095	0·796	......	0·2133(e)
Blast furnace slag from refinery slag reduction..	44·43	6·43	0·69	......	4·22	43·40	0·3064	......	0·0174	0·0059	......	......	......	....

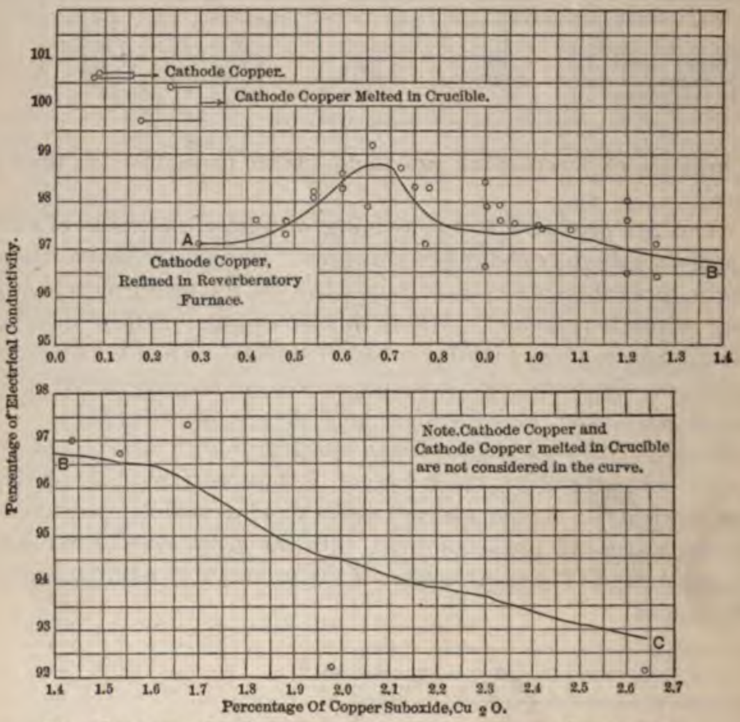
(a) 105 oz. (b) 109·4 oz. (c) 14·5 oz. (d) 23·1 oz. (e) 62·2 oz.

In the last slag the ultimate loss of copper is between 0·03 and 0·04% of the total copper refined.

It has already been pointed out that all the impurities in copper cannot be eliminated by the refining process. The final object of the latter is to bring the copper to the proper pitch, as it is termed; that is, to a point at which the surface of the copper on solidification or setting shows neither a rising nor a sinking, and at which the copper develops its best physical properties.

In the course of the oxidizing process the copper absorbs much oxygen, forming copper suboxide, which is held in solution by the metal. When in the process of poling this oxygen is not sufficiently removed, the surface of the copper on setting will sink. This is called a low pitch. When too much of the oxygen is removed the surface will rise. This is called a high pitch.

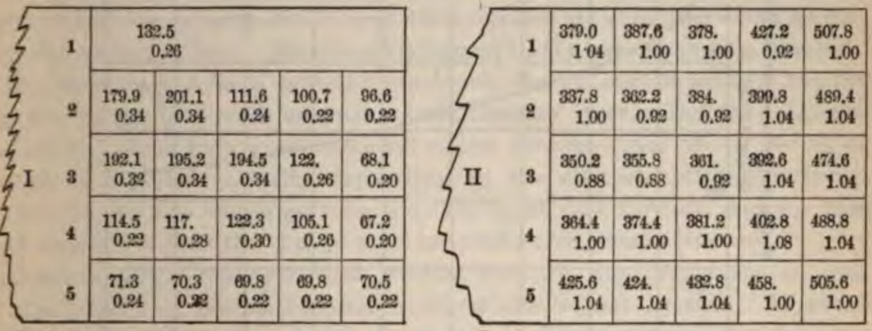
In the chemical analysis of refined copper it is customary to express, not the amount of oxygen, but the corresponding amount of suboxide of copper. Experience has taught that with refined, purest electrolytic copper, for rolling and conductivity purposes, from 0·6 to 0·8% of suboxide will cause the proper pitch and the development of the best results as to physical properties. With cakes of refined converter copper for the Hayden mill from 2 to 3% of suboxide is permissible. Mr. Walker, who has studied to some extent the influence of suboxide on the conductivity of copper, gives the accompanying results in graphic representation. Some of the irregularities are undoubtedly due to such variations in quantity of other impurities as will occur even in the purest of wirebar copper.



THE RELATION OF ELECTRICAL CONDUCTIVITY AND SUBOXIDE OF COPPER IN REFINED ELECTROLYTIC COPPER.

Metallic copper, alloyed with small quantities of other metals and metalloids, when in a molten state, shows upon solidification a segregation* of those impurities, and a concentration of them toward the center of solidification, each element, however, showing a different degree of such concentration. The latter seems to increase with the difference between the atomic volume of copper and that of the elements with which it is alloyed.

When the quantity of the impurities in the copper is increased beyond a certain limit the direction of their concentration upon segregation is reversed. They are found to be lowest at the center of solidification and highest along the surface. In the two diagrams accompanying are represented the two classes of copper alloys, showing the distribution of silver and gold along the vertical section of a block of copper 9 in. wide, 10 in. long, and 5 in thick, the upper figure of each pair being silver and the lower gold, expressed in ounces per ton. The subjoined analyses are of the samples along the vertical axis of the block:



I					
1	132.5 0.26				
2	179.9 0.34	201.1 0.34	111.6 0.24	100.7 0.22	96.6 0.22
3	192.1 0.32	195.2 0.34	194.5 0.34	122. 0.26	68.1 0.20
4	114.5 0.22	117. 0.28	122.3 0.30	105.1 0.26	67.2 0.20
5	71.3 0.24	70.3 0.22	69.8 0.22	69.8 0.22	70.5 0.22

II					
1	379.0 1·04	387.6 1.00	378. 1.00	427.2 0.92	507.8 1.00
2	337.8 1.00	362.2 0.92	384. 0.92	399.8 1.04	489.4 1.04
3	350.2 0.88	355.8 0.88	361. 0.92	392.6 1.04	474.6 1.04
4	364.4 1.00	374.4 1.00	381.2 1.00	402.8 1.08	488.8 1.04
5	425.6 1.04	424. 1.04	432.8 1.04	458. 1.00	505.6 1.00

BLOCKS OF COPPER SHOWING DISTRIBUTION OF PRECIOUS METALS.

BLOCK I.

Sample.	I.	II.	III.	IV.	V.
	%	%	%	%	%
Silver	0·454	....	0·641	....	0·248
Gold	0·00089	....	0·00109	....	0·00068
Lead	0·019	....	0·058	....	0·002
Bismuth	0·024	....	0·055	....	0·005
Antimony	0·099	....	0·157	....	0·048
Arsenic	0·074	....	0·108	....	0·034
Tellurium	0·019	....	0·027	....	0·004
Sulphur	0·047	....	0·112	....	0·040

BLOCK II.

Silver	1·299	1·158	1·201	1·249	1·459
Gold	0·00356	0·00343	0·00302	0·00343	0·00356
Lead	2·487	2·445	2·366	2·531	2·851
Bismuth	0·400	0·360	0·360	0·310	0·420
Antimony	3·438	3·073	2·820	2·789	3·929
Arsenic	0·705	0·716	0·630	0·627	0·706
Tellurium	0·892	0·768	0·814	0·856	0·882
Sulphur	0·538	0·467	0·462	0·487	0·469
Iron	4·220	5·050	4·430	5·460	3·500

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* These facts, and their important bearing on the sampling and assaying of metallic copper, were first made known by the writer in a paper presented to the American Institute of Mining Engineers (*Transactions*, Vol. XXVII., p. 106), and also published in the *Journal of American Chemical Society*, XIX., p. 243.

It may be inferred with almost mathematical certainty that there is an alloy of copper less pure than I. and purer than II. which is homogeneous, *i.e.*, in which the impurities present in a molten state on solidification do not concentrate in any portion of the body of metal.

In a block of purer copper than the above (I.) the suboxide was determined with the following results: Suboxide in portion of low concentration, 1.8%; suboxide in portion of high concentration, 2.687%.

A wirebar has not been tested as to the distribution of the suboxide in it; but it seems safe to predict that copper, even of that purity, is not homogeneous, and that it contains the greatest amount of suboxide at the center of solidification. The latter must be very close to the top, on account of the rapid chilling by the heavy mold on the two sides and bottom.

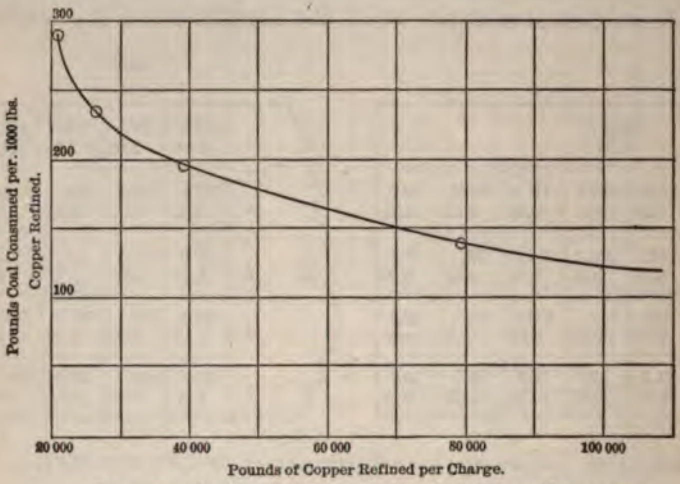

DIAGRAM OF COAL CONSUMPTION IN COPPER REFINING REVERBERATORY FURNACES.

Improvements in Methods of Refining.—The enormously increased output of copper during the past 15 years, and the reduced prices, with increased competition, naturally proved a great stimulus to the efforts toward reducing the cost of production. Naturally these efforts were first along the line of labor-saving appliances and in the economy of fuel. In connection with the latter there is to be mentioned the progressive step recently taken at the Anaconda works of utilizing the heat of the molten copper in the converter, by transferring it to a tilting furnace, where it is refined, then poured into a large ladle, and from it into the anode molds. This is the first move in the direction of combining the converting and refining process, as also of utilizing the heat of the molten blister copper. The enormous loss of caloric value through the flues and stacks of the reverberatory furnaces has also been recognized, and several of the largest establishments now pass the hot furnace gases through vertical boilers, thus generating considerable power from a source which long has been an absolute waste.

Up to about 30 years ago the reverberatory furnaces for this class of work would hold charges of only from 10,000 to 15,000 lb. of copper. As late as 1890 the capacity of the same kind of furnaces had not been increased to over 25,000 lb. W. H. Peirce, manager of the Baltimore Copper Smelting and Rolling Co.'s works, was the first to take a long step forward in increasing the size of the reverberatory for the purpose of refining converter copper. Mr. Peirce built such a furnace with a hearth area of 198 sq. ft., which, under the practical superintendency of William Griffith, now produces upward of 150,000 lb. of refined converter copper every 24 hours. At present Mr. Peirce contemplates the building of a 120-ton furnace. Mr. Griffith has also doubled the former output of the old furnaces in which electrolytic copper is refined. Such large charges can only be obtained by filling the furnace to full capacity, melting down the first part of the charge, and then filling in the remainder. With conductivity copper, however, it is not advisable to have the molten copper in the furnace deeper than about 18 in., because with a greater depth it becomes difficult to obtain a uniform pitch in the poling process. By the enlargement of the furnace Mr. Peirce demonstrated its great economic advantage in the saving of fuel. He gives his figures in the accompanying diagram.

As the furnaces had been operated previously the process of ladling out the charge was by far the most expensive task, because it required the highest-priced labor. Each man would take out of the furnace from 25 to 50 lb. of copper at a dip, the quantity depending on the character of the castings. From four to six men would perform the ladling at one furnace, and for this limited number of men the charge must naturally also remain limited.

Mechanical and semi-mechanical ladling, directly from the furnaces, was tried at a number of places, however, without any signal success. To avoid the taking out of the copper by means of ladles, the tapping of the furnaces was then resorted to. The first copper refining furnaces with tapping arrangement were built by A. F. Schneider at the works of the Guggenheim Smelting Co. at Perth Amboy, N. J., but for reasons unknown to the writer these furnaces were never put into operation. Later Mr. Peirce introduced tapping and the use of large ladles to convey the copper at his large furnaces, in which he refines converter copper with perfect success. The accompanying illustration gives a partial view of one of Mr. Peirce's large furnaces, with his ladling and other contrivances, at the Baltimore Copper Co.'s works. In the center of the picture is seen the spout, with a ladle in position to receive the molten metal, the flow of which is regulated by a man stationed at the taphole. The latter is closed with clay and iron bars in front, which are removed as the level of the metal in the furnace is lowered. During the change of ladles the flow of the metal is checked with a board coated with clay. The two ladles, which in this case have a capacity of about 750 lb., are suspended from turning-cranes. Their vertical motion, and the horizontal one along the line of the beam of the crane, is effected by compressed air and hydraulic power, governed from a station to the right, not visible in the picture. The circular, horizontal swinging of crane and ladle, and the tilting of the latter, are performed manually. The molds are arranged in a straight row along the bosh. They are dumped mechanically, from the station already referred to, in groups of two or three. The cakes are received by a mechanical conveyor and are carried to a trimming table, seen at the extreme right of the picture. The molds are made of copper, at the furnace.

Although these successful innovations by Mr. Peirce were somewhat of a revolution in the industry of copper refining, especially in the handling of the cruder grades of metal, the casting of electrolytic copper into wirebars, cakes, and ingots by the herein previously described mechanical appliances still remained unsuccessful, because splashings and cold sets, which make those castings unacceptable, seemed unavoidable except by hand ladling. It was left to A. L. Walker to devise an apparatus of considerable ingenuity and simplicity which would overcome all the difficulties that had hitherto proved to be insurmountable. Having witnessed its practical working, I consider it one of the most important inventions in the field of present metallurgical industry, and therefore I give Mr. Walker's description in full detail.

Walker's Improved Casting Apparatus.—"This was first tried at the new electrolytic refinery of the Baltimore Copper Smelting and Rolling Co., July, 1897, and soon demonstrated its great value. After making a thorough trial it was decided to build another machine at once, embodying some improvements. This also was successful, and three more were installed, making five in all, so that now almost the entire product is cast by these machines, and no faults have been developed. The cost of labor for casting copper has, by this means, been reduced to one-half of what it formerly was, since the machine can be operated by low-priced men, only one (the furnaceman in charge) receiving the wages formerly paid hand-ladlers. The capacity of the furnaces has been greatly increased, and as much as 86,000 lb. (shipping weight) of fine copper has been taken out in a single charge. This would be out of the question with hand work, even by greatly increasing the crew of ladlers. The regular capacity is 80,000 lb. per charge, which is cast in wirebars or cakes in about four hours. With furnaces of larger capacity the output could be still further increased, and this means not only a great saving in labor, but also in fuel and general supplies.

"The metal is refined in furnace (to left of drawing),* and run by means of a spout into receiving pot or ladle *B*, which is so arranged that it can be raised at the rear for pouring, and also moved backward to clear the ends of the molds when the turntable wheel *D* is revolved, and forward to bring it into position above the center of the molds for casting. These two movements are regulated by an attendant standing on platform *F* operating the two levers, one of which controls the small hydraulic lift that raises the rear of the ladle, and the other rotates the shaft on which the curved supports for the ladle are secured, thus moving the latter horizontally. The trunnions of the ladle, which rest in these curved supports, are of necessity placed near its mouth, so that the distance of the fall of metal will remain nearly constant, when the back of the ladle is raised or lowered, thus avoiding any danger of splashing or irregular pouring.

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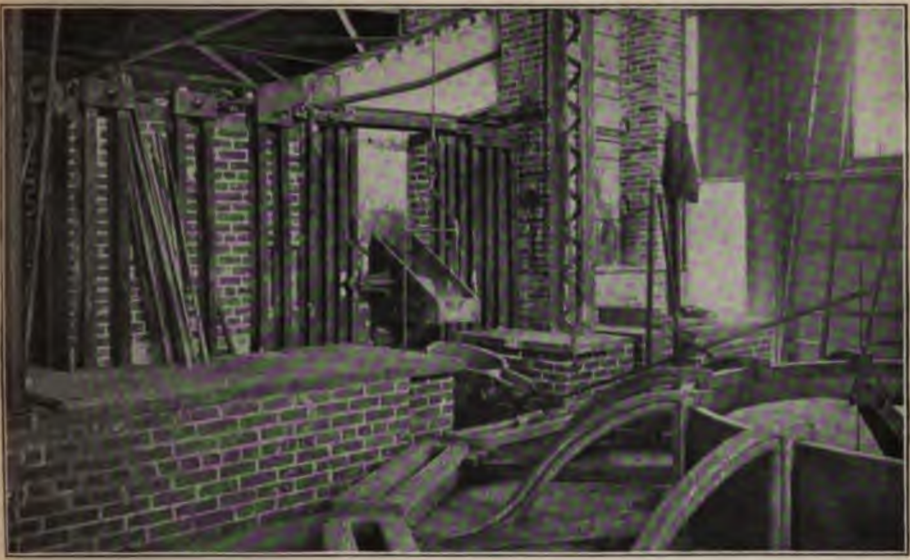
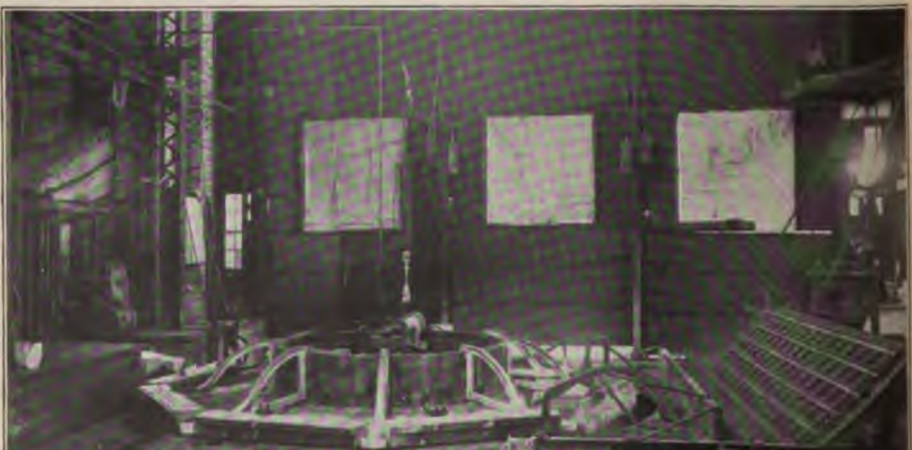
* These drawings have been furnished through the kindness of Mr. A. L. Walker, who reserves the right of independent publication. The apparatus is patented.—EDITOR.



Walker's Improved Casting Apparatus.
Showing Receiving Ladle, Deflector, Molds, and Turntable Arms.

Walker's Improved Casting Apparatus.
Showing Turntable Mechanism, and Bosh Conveyor.

PEIRCE SEMI-MECHANICAL CASTING APPARATUS, BALTIMORE COPPER CO.'S WORKS.
Showing Tapping Spout, Ladles, Tipping Molds, and Water Bosh.



WALKER'S IMPROVED CASTING APPARATUS.
View of whole system, showing Furnace, Spout, Ladles, Turntable, Molds, Water Bosh, Dumping Rig, and Conveyor.

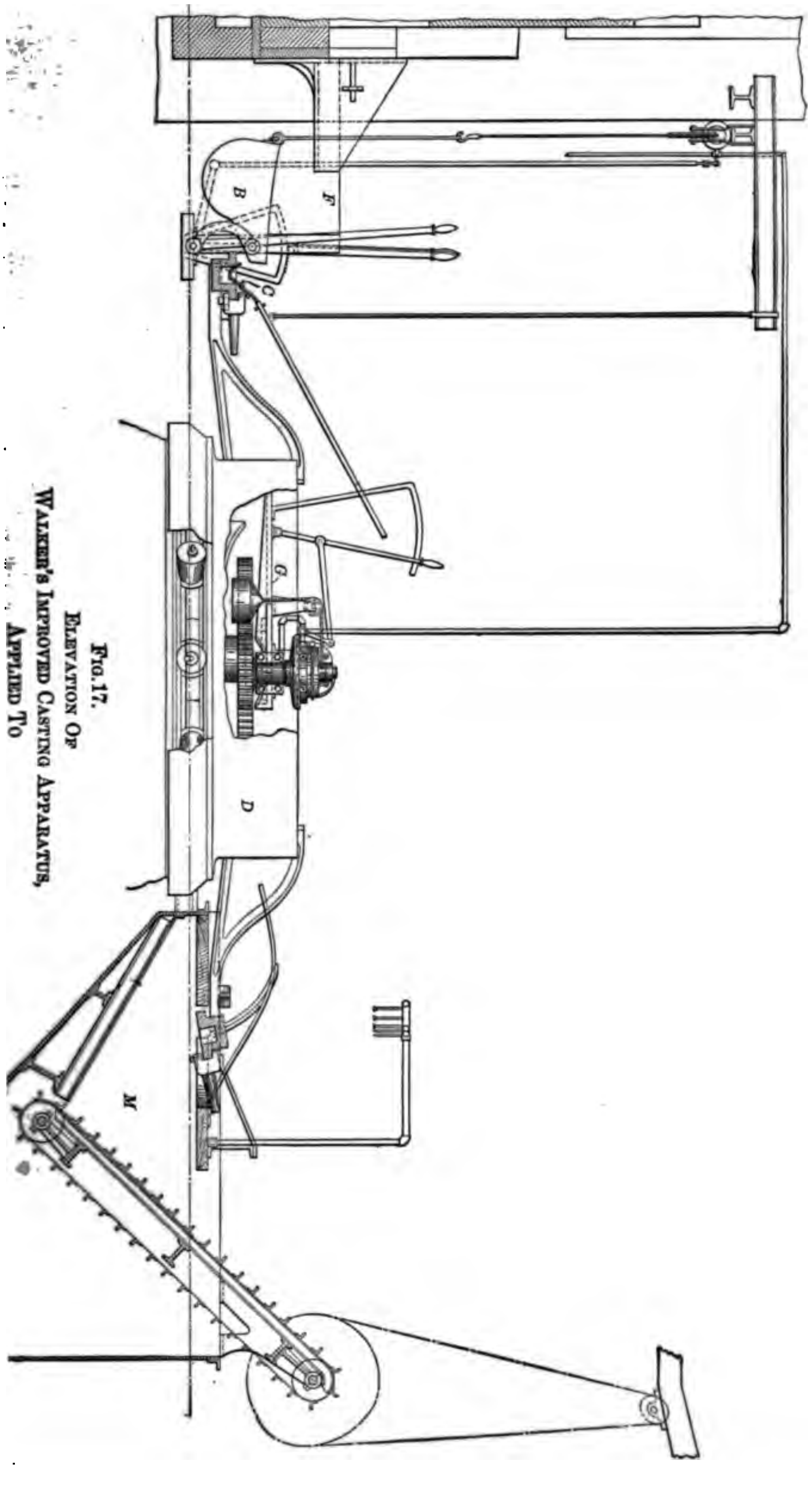


FIG. 17.
ELEVATION OF
WALKER'S IMPROVED CASTING APPARATUS,
APPLIED TO

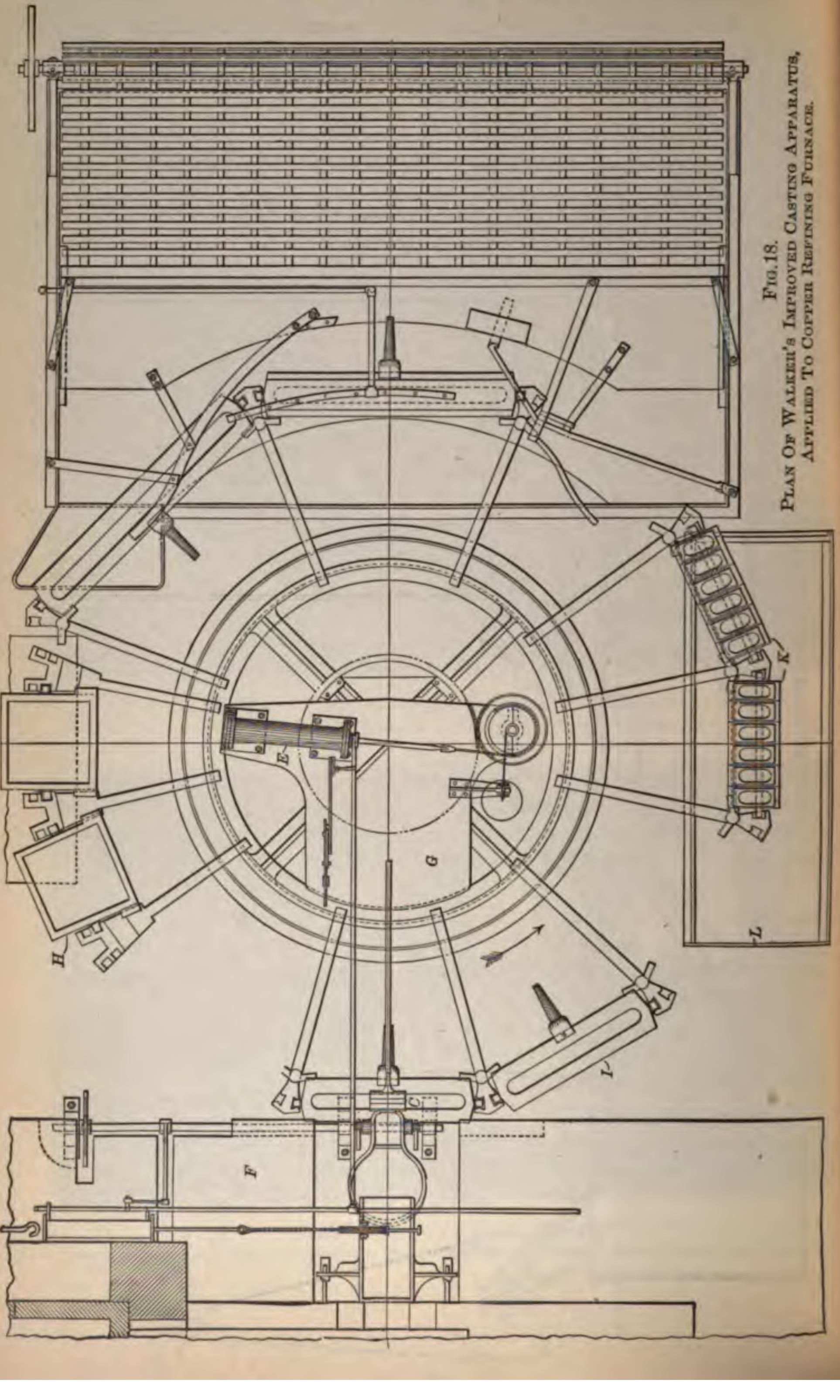


FIG. 18.
PLAN OF WALKER'S IMPROVED CASTING APPARATUS, APPLIED TO COPPER REFINING FURNACE.

"The metal from the ladle is received on deflector *C* and distributed in the mold in such manner as to prevent clinging or cold sets. This deflector is shaped as shown in the drawing for long castings, such as wirebars, but for making cakes it is more like a flat dish, and for ingots this dish is provided with pouring lips. The cake molds *H* and the wirebar molds *I*, for receiving the metal, are supported by means of trunnions fitting in recesses at the ends of the cast-iron arms, which in turn are hung from the rim of the turntable wheel, the arms being arranged so that they can be moved on the rim of the wheel to accommodate longer or shorter molds.

"The trunnions on the molds are situated a little in front of the center of gravity, so that the rear flange of the latter will rest on the crossbar projection of the arm. In case of small molds *K*, several can be put on one shaft and a crossbar placed between the projections of the cast-iron arms to act as a common rest. The mold trunnions in any case may rest directly in the recesses of the arms, which are beveled to allow for longer or shorter molds, or may rest in bushings placed in these recesses, as shown at *K* on the drawing.

"After the mold is filled the deflector is moved aside, and the turntable wheel rotated by means of a hydraulic cylinder *E*, the rod of which is connected by a flexible rope with a loose drum, revolving on a vertical shaft, which latter runs in a heavy bearing securely fastened to a fixed central platform *G*. On the loose drum are pawls which engage in a ratchet that is keyed to the vertical shaft, as is also the gear wheel below the platform. The smaller gear wheel engages in the larger one, which is securely fastened to the main turntable wheel so that the latter two revolve together.

"When the turntable wheel has revolved far enough, the water is allowed to flow from the cylinder *E*, and the piston rod of the same is drawn out, in position for another stroke, by a counter weight which acts on, and unwinds, the cord on the upper and smaller part of the loose drum, at the same time winding up the flexible rope attached to the piston rod. An attendant, standing on the platform *G*, operates the lever which controls the hydraulic cylinder *E* and the handle of the deflector *C*.

"The operation of casting is therefore as follows: Copper is allowed to run from the furnace spout till the ladle *B* is nearly full; then the attendant on platform *F* brings it into position for pouring, and raises the rear end till the metal flows from the ladle to the deflector *C*, and then into the mold. When the latter is full to the mark the flow of metal is stopped by lowering the back of the ladle; the attendant moves the deflector *C* to one side and rotates the turntable wheel until another mold is in position in front of the receiving ladle, when the operation is repeated.

"In the case of small castings which solidify quickly, the molds are dumped, after rotating 90°, in a dry pit or bosh *L*, but larger castings are dumped, after rotating 180°, in bosh *M*. In the latter case the molds are provided with tripping arms, which, in the revolving of the turntable wheel, engage on an inclined rod, held in position by a framework on bosh, raising the arm until it turns over and falls against a rest, and then down on the platform, by which time the mold is in an inverted position and the casting dumped out, falling upon the receiving plate in the bosh. On further rotation of the wheel, the arm is raised by another inclined rod, held securely in position until the mold is returned to its original position and ready to receive molten metal.

"The bosh *M* is filled with water, and the molds on being turned over are partly cooled by passing through the same. They are still further cooled by a spray from overhead playing on the bottom of the molds when they are in an inverted position.

"The castings falling in bosh *M* slide down and are caught by a conveyor (castings in bosh *L* are also raised by a conveyor, but this is not shown). This conveyor runs very slowly, so that by the time the castings reach the level of the water they are completely cooled, and when delivered at the highest point of the conveyor can be easily handled, or allowed to fall on a platform if desired. The conveyor is operated by any means, such as a motor and chain or belt.

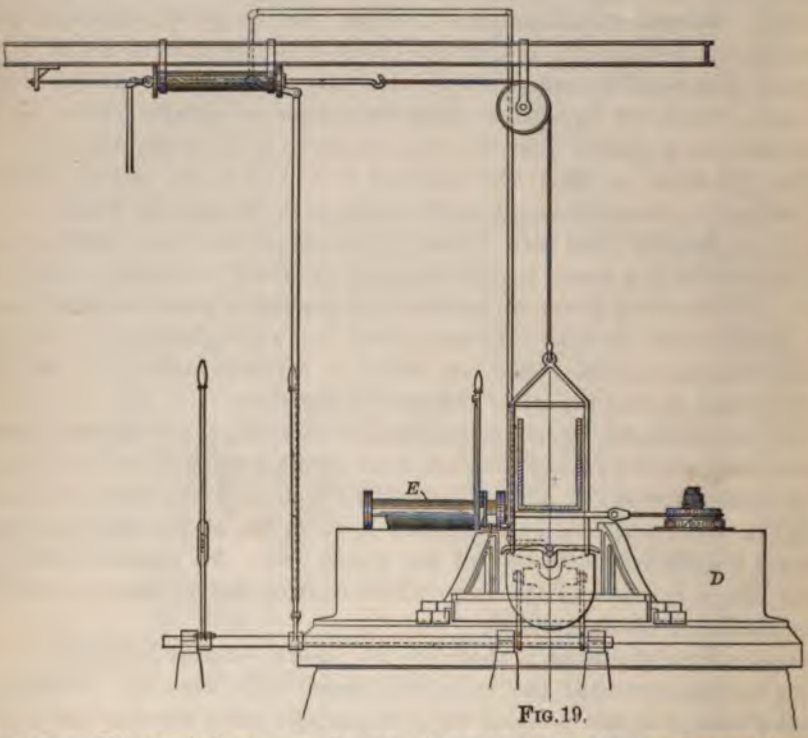


FIG. 19.

ELEVATION OF WALKER'S IMPROVED CASTING APPARATUS, APPLIED TO COPPER REFINING FURNACE.

"It will thus be seen that no hand labor is used in the operation of making castings by this machine other than the work required in supervising operations and manipulating the levers as described. The finished product is delivered at the top of the bosh conveyors ready for shipment."

On the general drawings the deflector *C* is shown as hung from an overhead support, but it can also be supported by means shown in Figs. 20, 21, 22 and 23, the small plate. In many cases the deflector need not be used at all, and d castings are obtained by pouring from the tilting ladle directly into the ds; the former being arranged so that the metal will have a minimum fall. operation is thus simplified still more.

later improvement also provides for making a number of small castings, h as ingots, at one pouring; so that this work can be carried on more idly, and the output greatly increased.

tationary vs. Tilting Furnaces.—There is no doubt that now and then the stion will arise as to the choice between a tilting and a stationary furnace the purpose of refining copper. If the time consumed in emptying a

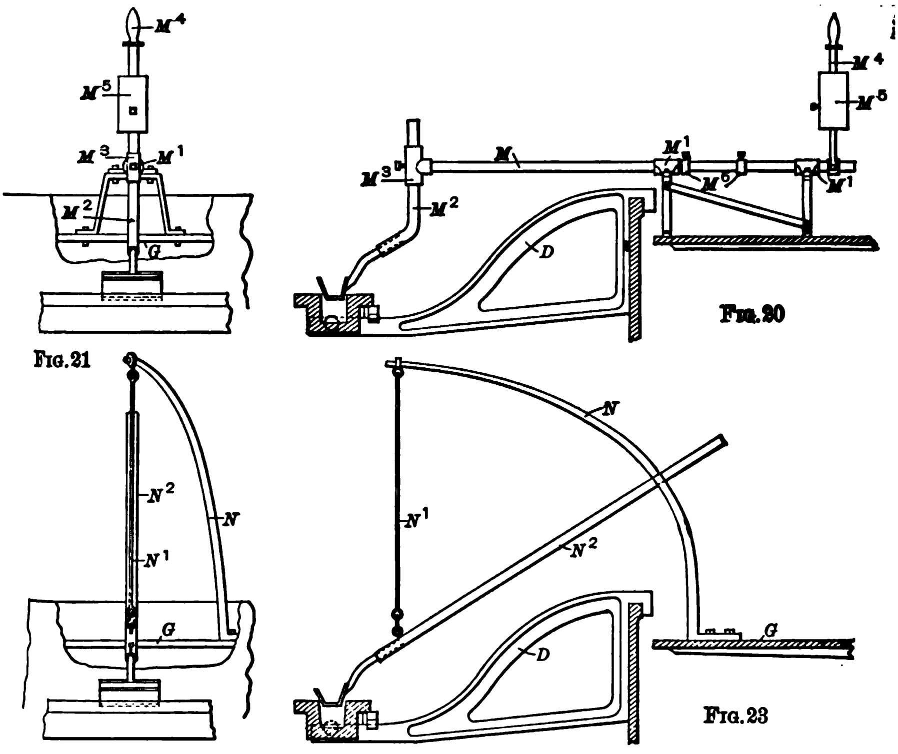


FIG. 20

FIG. 21

FIG. 22

FIG. 23

DETAILS OF DEFLECTOR, WALKER'S IMPROVED CASTING APPARATUS.

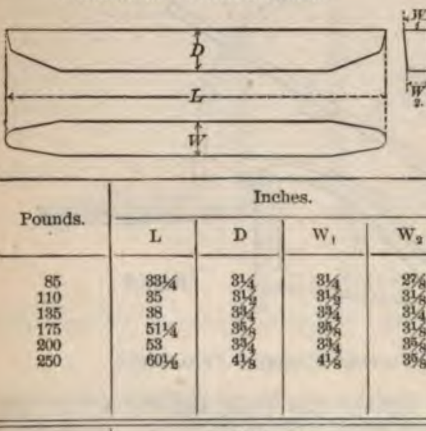
rnace were the main consideration, the former has undoubtedly a decided vantage. For all ordinary purposes, however, the taphole of the stationary rnace discharges the metal with sufficient rapidity. It has, moreover, the vantage of needing no machinery, and, size for size, allows of a far more assive construction, or heat-retaining capacity, and consequently permits eater economy in fuel.

Methods of Casting Copper.—In the practice of casting copper, iron and cop-r molds are in use. The latter are undoubtedly an improvement on the mer. They can readily be made from the material of each furnace, and en they become defective can be added at once to another charge. There is thus no waste or depreciation of mold material. The copper molds, which are always covered with a bone-ash washing, have the further advantage of producing much more perfect castings than the molds of iron; while castings from the former are generally quite smooth, they are, when turned out from the latter, usually covered with depressions and pinholes.

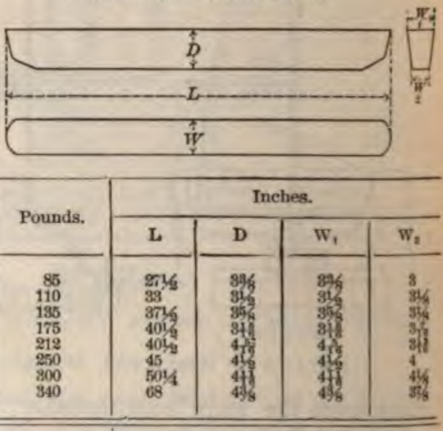
Although the color of a copper casting stands in no relation to its chemical composition and physical properties, there is nevertheless some attention given to the exterior appearance of these materials. All copper castings, if left to cool in the air, become covered with a scale of the oxides, which gives them a very dark appearance. If not removed, these oxide scales become a source of loss, as they are loosened gradually by the handling of the copper. All castings after solidification are, therefore, dumped into water. This is called the pickling process. To obtain the most desirable color, which is a lustrous one, somewhere between that of gold and that of copper, the pickling bath must consist of hot water with an addition of about 0.1% of sulphuric acid. Many shades of color are obtainable by the variation of temperature of the bath, the color becoming darker with the lowering of the latter.

At the Baltimore Copper Smelting and Rolling Co.'s works, where its own electrolytic copper, as well as that of the Anaconda Mining Co., is cast into the marketable forms, the standard shapes and sizes given in the tables below are used. Besides these, there are periodically cast solid cylinders for the manufacture of tubes; and hollow cylinders for the manufacture of the so-called bimetallic wire; *i.e.*, a copper wire with a steel core.

WIRE BARS—POINTED ENDS.



WIRE BARS—BLUNT ENDS.



Pounds.	Inches.				Pounds.	Inches.			
	L	D	W_1	W_2		L	D	W_1	W_2
85	33¼	3¼	3¼	2⅞	85	27½	3⅜	3⅜	3
110	35	3½	3½	3⅛	110	33	3½	3½	3⅛
135	38	3¾	3¾	3¼	135	37½	3⅝	3⅝	3⅛
175	51¼	3⅝	3⅝	3⅛	175	40½	3$\frac{15}{16}$	3$\frac{15}{16}$	3$\frac{7}{16}$
200	53	3¾	3¾	3⅜	212	40½	4$\frac{5}{16}$	4$\frac{5}{16}$	3$\frac{13}{16}$
250	60½	4⅛	4⅛	3⅝	250	45	4½	4½	4
					300	50¼	4$\frac{11}{16}$	4$\frac{11}{16}$	4⅛
					340	68	4⅜	4⅜	3⅞

Kgs.	Millimeters.				Kgs.	Millimeters.			
	L	D	W_1	W_2		L	D	W_1	W_2
38½	844	83	83	73	38½	698	86	86	75
50	889	89	89	79	50	838	89	89	79
61	965	95	95	83	61	952	92	92	79
79	1302	92	92	79	79	1028	100	100	87
91	1346	95	95	86	96	1028	110	110	97
113	1537	105	105	92	113	1143	114	114	102
					136	1276	119	119	105
					154	1727	111	111	98

Wire bars can be increased or decreased from the list weight about 10% by varying the depth, the other dimensions remaining unchanged.

ROUND CAKES.

Weights in pounds; sizes in inches.

Diameter. In.	Thickness.								
	1½	2	2½	3	3½	4	5	6	7
8	24	31	39	47	55	62	78	93	109
10	37	49	61	73	85	97	122	146	170
12	52	70	88	105	123	140	175	210	245
15¾	91	121	151	181	211	242			
20		195	243	292	341	390			
23⅝			340	408	476	543			
27½			460	552	644	737	921		
31½				725	846	966	1208	1450	
35½				921	1074	1227	1534	1841	2148
39⅜						1510	1887	2265	2642

Weights in kgs.; sizes in mm.

Diameter. mm.	Thickness.								
	38	51	64	76	89	102	127	152	178
200	11	14	18	21	25	28	35	42	49
250	17	22	28	33	39	44	55	66	77
300	24	32	40	48	56	63	79	95	111
400	41	55	68	82	96	110			
500		88	110	132	154	177			
600			154	185	216	246			
700			208	250	292	334	417		
800				328	383	438	547	657	
900				417	486	556	695	834	972
1000						684	855	1026	1197

SQUARE CAKES.

Weights in pounds; sizes in inches.

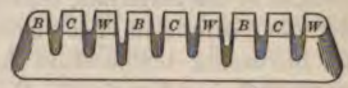
Size. In.	Thickness.									
	1½	2	2½	3	3½	4	5	6	7	8
14x17	111	148	184	221	258	295				
16x16	119	159	198	238	278	317				
18x18		201	251	301	352	402				
21x21			342	410	478	547	684			
24x24			446	536	625	714	893			
28x28				729	851	972	1215	1458		
33x33				1013	1182	1350	1688	2026	2363	
37x37					1485	1698	2122	2546	2971	3395
42x42						2187	2734	3281	3828	4375

Weights in kgs.; sizes in mm.

Size. mm.	Thickness.									
	38	51	64	76	89	102	127	152	178	203
356x432	51	68	83	101	117	134				
406x406	54	72	90	108	126	144				
457x457		91	114	136	159	182				
533x533			155	186	217	248	310			
610x610			202	243	283	233	405			
711x711				330	386	440	550	660		
838x838				459	535	612	765	916	1070	
940x940					673	769	961	1153	1346	1538
1067x1067						991	1239	1486	1740	1982

The size of cathodes is $23\frac{3}{4} \times 11 \times \frac{1}{4}$ in., or $603 \times 279 \times 6{\cdot}3$ mm.; the weight is 20 lb., or 9·1 kg.

INGOTS.



INGOT BARS.

Size of ingots: 9x3x3 in. = 228x76x76 mm. Size of ingot bars: 27x3x3 in. = 684x76x76 mm. The ingot bar is for convenience in shipping. Upon arrival at destination it can readily be broken into three ingots of ordinary size.

In comparing electrolytic copper with Lake copper it should be said that for nearly half a century the latter has ruled supreme in the markets of the world for all purposes where purity and uniformity of the material is required. For a long time it was in fact the only reliable copper on the market. No wonder, therefore, that prejudices in its favor have grown up among the practical consumers, and that its new competitor from the electrolytic refineries has had, and is still having, difficulty in securing an equal footing with the older article. The figures and facts which are presented in this paper clearly show that such a prejudice is no longer justified, and that while there is still electrolytic copper produced inferior to Lake copper, the various grades can be assorted by very simple and efficient means, either by the producer or the consumer.

For conductivity copper great purity is required. As has been shown by numerous analyses and conductivity tests, such impurities as arsenic, antimony, and bismuth begin to show a very marked deteriorating influence when present in quantities of a few thousandths of 1%. Antimony is the preponderating impurity in electrolytic copper, which, however, is about offset by the arsenic present in Lake copper. Lake copper has in its favor the absence of bismuth, while electrolytic copper shows to its credit a far smaller quantity of silver. Both contain about the same amount of lead.

Comparatively pure copper is also required in the manufacture of fine grades of brass. The influence of antimony and bismuth on the qualities of the latter has recently been thoroughly investigated by Erwin S. Sperry.* In these investigations the conclusion was reached that with an upper limit of 0·01% in the brass, the two elements cease to create a disturbing influence. According to this authority, therefore, not Lake copper alone, but all the grades of electrolytic copper presented in the table on page 15 are of sufficient purity to produce the best brass required.

In conclusion it may be of value to give a brief resumé of the extensive tests made of recent date at the Königlichentechnischen Versuchsanstalten in Berlin,† as they were undertaken to determine the following questions: (1) The influence of the chemical composition; (2) the relation between the chemical composition and the mechanical treatment; (3) the permissible stress under normal conditions (annealed) and when subjected to mechanical treatment; (4) the strength of joints, either riveted or brazed, or combination of the two; (5) the influence of heat upon physical properties.

These questions have important bearing upon cases where copper is used for the construction of apparatus subject to high temperature and pressure, as is now common in boilers, tubes, etc. The general results gathered from numerous individual tests are as follows: The mechanical treatment (hard rolling, hard drawing, and cold hammering), as already shown in former tests, increased the strength of the materials, especially at the elastic limit, accompanied by a considerable diminution of the elongation.

With increasing temperature the materials are weakened; however, up to 200° to 300° C. in a less degree than at higher temperatures. The elongation under equal stress and within the elastic limit increases with increasing temperature. The elongation is increased by cooling to —20° C.; between +20° and 300° C. it shows little variation, while higher temperatures generally diminish the elongation of annealed material and increase that of the mechanically treated. The increased strength of the mechanically treated material remains almost invariable up to 200° to 300° C., when with further increasing temperature it begins to diminish, and reaching 500° C., mechanically treated and annealed materials again have the same elastic limit and tensile strength.

The influence of chemical composition has not been conclusively demonstrated; the range of difference in such composition between the various materials was probably not great enough. It would appear, however, that the more impure copper, when mechanically treated, retains its increased strength to a greater degree than does the purer metal when subjected to higher temperatures, and treatment being equal, the impure copper seems to attain greater strength in general. If the latter conclusions be correct there will not only remain a good market for first class conductivity copper, but also one for the apparently less favored lower grades.

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* See papers read before the American Institute of Mining Engineers, Atlantic City and Buffalo meetings, 1898.

† *Mittheilungen aus der Kgl. tech. Versuchsanst*, XVI., 172.

FELDSPAR.

This mineral is produced in the United States entirely for consumption in the pottery trade, in which it is known commonly as "spar." It serves as a constituent in the body of white earthenware or porcelain, playing the part of a flux to bind the mass together, and is also one of the elements of porcelain glaze. The chief sources of feldspar in the United States are Brandywine Summit, Pa., Branchville and Glastonbury, Conn., Georgetown and Topsham, Me., and Bedford, N. Y. Some has also been obtained at various times in Delaware, Virginia, and North Carolina, the two Southern States being recent sources of supply.

The mineral occurs in veins often of large size and usually contains quartz and mica intermixed, which have to be separated by handpicking. There is much spar, however, that contains fine quartz, or quartz so interwoven with it that it cannot be separated by breaking and hand-sorting. The clean spar is shipped to Trenton, N. J., or East Liverpool, Ohio, where it is ground fine with heavy stone rollers and screened. The pottery trade requires that the feldspar shall be practically free from iron oxide, mica, and quartz, and that when subjected to the heat of the pottery biscuit kiln it shall fuse into a white opaque glass.

PRODUCTION OF FELDSPAR IN THE UNITED STATES.

State.	1896.		1897.		1898.	
	Long Tons.	Value.	Long Tons.	Value.	Long Tons.	Value.
Connecticut	5,732	$35,788	5,871	$34,231	6,696	$37,944
Maine and Pennsylvania	11,192	51.484	12,880	59,161	12,954	58,953
Maryland	2,600	13,700	1,500	7,875	None.	None.
Massachusetts	1,344	6,300	150	525	150	525
New York	900	5,300	1,500	9,600	1,550	9,725
Totals	$21,768	$112,572	21,901	$111,392	21,350	$107,147

The Maine feldspar is considered the best; that of Connecticut is of good quality but it is not easy to obtain much of it free from quartz. The Brandywine Summit spar, which is produced by a number of small quarries, is considered inferior to the Maine and Connecticut, and requires a much closer sorting. The more part of it goes to East Liverpool, Ohio.

FELDSPAR; ITS OCCURRENCE, MINING AND USES.

BY T. C. HOPKINS.

Mineralogical.—Feldspar, commonly called "spar" among the miners, is the name of a group of well-known and widely distributed minerals. The characteristics which distinguish this group from other minerals are: (1) Cleavage in two similar directions, inclined at an angle of 90° or nearly so; (2) crystallization in the monoclinic and triclinic systems, the crystals resembling each other closely in angle, habit, and methods of twinning; (3) hardness, 6 to 6·5; (4) sp. gr. varying from 2·5 to 2·9; (5) colors white or pale shades of yellow, red, green or dark. In chemical composition they are silicates of alumina combined with potash, soda or lime, and rarely baryta

The three well-defined species in the group are: (1) Orthoclase, the potash feldspar; (2) albite, the soda feldspar; and (3) anorthite, the lime feldspar. Associated with the first species are microcline and anorthoclase. The second and third grade into each other through the intermediate species, oligoclase, andesine, labradorite and bytownite, which contain both soda and lime in varying proportions—all forming the soda-lime group. From another point of view all the feldspars may be divided into two groups: (1) Orthoclase, the acid or potash feldspar; and (2) plagioclase, the basic or soda-lime feldspars. To the manufacturer of china ware there are the potash feldspar, the soda feldspar, the lime feldspar, and the soda-lime feldspars, which differ in (1) chemical composition, (2) specific gravity, (3) in their fusing point, and (4) in their different fluxing action in his mixes. For comparison these properties are grouped in the following table:

PHYSICAL PROPERTIES OF FELDSPARS AND THEORETICAL COMPOSITION.

Name.	Hardness.	Fusibility.	Specific Gravity.	SiO_2.	Al_2O_3.	CaO.	Na_2O.	K_2O.
				%	%	%	%	%
Orthoclase	6	5·0	2·57	64·7	18·4	....	....	16·9
Albite	6-6·5	4·0	2·624	68·7	19·5	....	11·8	....
Oligoclase	6-7	3·5	2·64	65·7	21·5	2·4	10·4	....
Andesine	5-6	....	2·69	57·3	27·1	8·9	6·6	....
Labradorite	5-6	3·0	2·71	53·0	30·1	12·3	4·6	....
Bytownite	....	....	2·74	46·6	34·4	17·4	1·6	....
Anorthite	6-6·5	5·0	2·76	43·2	36·7	20·1	....	....

The table shows an increase in specific gravity from orthoclase at one end to anorthite at the other. The fusing point lowers to 3 at the middle of the series and rises to 5 at each end. Likewise the hardness decreases from each end toward the middle.

The composition of the orthoclase, albite and anorthite given in the table are theoretical rather than actual analyses, as these minerals rarely occur absolutely pure in nature. Orthoclase nearly always contains some soda, by the increase of which it grades into anorthoclase, an intermediate species between orthoclase and albite. The accompanying table of actual analyses will show the variations from the normal composition of the different species:

ACTUAL ANALYSES OF FELDSPARS.

Name.	Locality.	SiO_2.	Al_2O_3.	CaO.	Na_2O.	K_2O.
		%	%	%	%	%
Orthoclase	Embreeville, Pa	67·46	21·76	1·67	5·96	4·28
Orthoclase	Bolton, Mass	65·23	19·26	0·42	2·98	11·80
Orthoclase	Brandywine Summit, Pa	65·61	16·92	0·16	2·11	12·92
Orthoclase	Brandywine Summit, Pa., 1898	65·76	18·94	....	1·80	12·68
Orthoclase	Glen Hall, Pa	70·36	19·34	1·56	4·68	3·00
Orthoclase	Bedford, N. Y	65·95	18·00	1·05	2.11	12·13
Orthoclase	Saint Genevieve, Mo	64·80	18·00	·60	....	15·90
Albite	Unionville, Pa	66·65	20·79	1·47	8·86	1·36
Albite	Mineral Hill, Pa	66·34	20·72	1·85	9·44	0·98
Oligoclase	Wilmington, Del	64·75	23·56	2·84	9·04	1·11
Oligoclase	Sanford, Me	56·65	25·56	8·25	6·18	1·34
Labradorite	Labrador	56·00	27·50	10·10	5·00	0·40
Anorthite	Mt. Somma, Italy	43·98	35·30	18·98	0·47	0·40

Occurrence.—Orthoclase, or potash feldspar, is the variety that is used most extensively in pottery, as it occurs more abundantly in a relatively pure form than the others. The basic feldspars are nearly always associated with dark-colored minerals, such as hornblende, augite and biotite, generally so intimately associated that they cannot be separated economically in commercial quantities. Orthoclase is the principal feldspar in granites, syenites, gneisses and mica schists, and in some localities occurs coarsely crystallized with other minerals in veins or dikes intersecting these rocks. The dikes are called pegmatite (giant granite, granitic vein stone),* or sometimes binary granite, and are composed of orthoclase, quartz, muscovite, and, less abundantly, other minerals. Where other minerals occur in these veins they are usually segregated in such a manner that they can be separated by a process of hand-picking which is followed at all the quarries. Hornblende and augite occur rarely in these dikes in large quantities, biotite sometimes and muscovite frequently. But both the micas occur in large crystals which can be broken out by hand. The large quartz segregations are also picked out, but most of the quartz which is deposited in the cleavage planes of the spar, in the form known as graphic granite, is not removed, hence a commercial analysis often shows a higher percentage of silica than the analysis of a selected sample of spar free from quartz. Within reasonable limits the higher percentage of quartz is no objection to the spar for pottery work, provided it is uniform in composition. The mix for china ware contains both spar and "flint," and if the spar carries a high percentage of "flint" it is only necessary to add more of it, and less "flint," to get the same proportions as before.

The occurrence of the spar is not limited to the granitic rocks, as large deposits of it occur in the serpentines of southeastern Pennsylvania, in Chester County. While feldspar is known to exist in serpentine in other localities, so far as known to the writer the Chester County quarries are the only places where it is worked or occurs in workable deposits in that rock. In the Chester County serpentine spar quarries quartz is absent; but biotite, muscovite, hornblende and inclosed serpentine occur, thus necessitating a close hand

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* The term pegmatite was first used to designate that unique intergrowth of quartz and feldspar now known as graphic granite or binary granite, but the meaning was later extended to include all those coarsely crystallized segregations of feldspar and other minerals found in igneous rock masses.

picking. Quartz occurs in large quantities in the serpentine in the region of the spar quarries, but none was observed in or at the spar quarries.

Uses.—Feldspar is used extensively in the china and porcelain factories, where it serves as a flux in combining the different substances of the ware. The common ironstone china ware is made of a mixture of kaolin or china clay, ball clay, flint, and feldspar. These are mixed in different proportions according to grade, kind of ware and recipe of different potters. One of the largest concerns in Pennsylvania uses the materials in the following proportion in making ironstone china: kaolin or china clay, 35%; ball clay, 16%; quartz ("flint"), 35%; and feldspar, 14%. The ball clay gives the mass plasticity, so that it can be molded into the desired form. The kaolin furnishes a refractory body, while the flint is to prevent excessive shrinkage in the drying and burning, and the feldspar serves as a flux to unite all the ingredients into a partially vitrified material.

Feldspar is also used as one of the principal constituents in making the glaze for chinaware and is an essential ingredient in porcelain. It is used in making glazed tile, where it serves the same purpose as in china making, and in certain kinds of glass. More than 500 tons of feldspar was used in the United States in 1898 by soap makers and for the preparation of polishing materials. It is used in dentistry in small quantities.

Mining.—The spar in some places extends to the surface, but in most localities the upper part has disintegrated to a depth varying from a few inches to several feet. Scattered through this disintegrated material fragments of sound spar remain, which are generally separated from the other material by screening in the same way that gravel is removed from sand. Where the material is too wet to screen it is washed. The pick and shovel are all the tools necessary in many of the mines of the United States.

When the mines extend to a depth beyond the action of the weathering agencies, blasting is resorted to in order to loosen the material. After working too deep to throw out the material with a shovel, a small derrick is erected and the material is lifted in tubs. Steam power is used at some of the quarries, but horse power is employed at many quarries, as the output is too small to justify the expense of a steam hoist. Most of the quarries are worked from the surface in open pits, but in several places underground work has been done where the upper part of the vein would not pay for removal.

Preparation for Market.—After the material is removed from the vein to the surface it is carefully sorted by men with hand hammers, who break out and separate the grosser impurities, particularly the dark-colored minerals. It is then carted to the railway and sent to the spar mill to be ground. There are two of these mills in Pennsylvania, which receive much of the product from the Pennsylvania quarries, but some of the spar is shipped in lump directly from the mines to the potteries and is there ground. There are spar mills at the two great pottery centers, Trenton, N. J., and East Liverpool, Ohio, to both of which spar is sent from the Pennsylvania quarries.

The grinding is a twofold process, the first part consisting in crushing the spar in a kind of Chilean mill, with heavy stone rollers running in a shallow pan of small diameter into which the spar is fed by shovel. The general plan is about the same as the dry pan for grinding clay and shale, except the larger stone rolls replace the smaller iron ones of the dry pan and they run in a smaller pan. The crushed material is then put in a revolving drum together with a quantity of Norway pebbles and revolved rapidly for two or three hours, which reduces the material to an impalpable dust. It is then ready for use by the potter or the glassmaker.

Prices.—The prices vary somewhat with the grade of the spar, and like other commodities fluctuate with the supply and demand. At the Pennsylvania quarries the price for the crude spar unground f. o. b. varies from $3 to $4·25 per long ton, most of it bringing about $4. The prices at Trenton range from $5 to $7. The price for ground spar is of course a little higher. The Maine feldspar is of purer quality and commands a better price. In 1898 lump feldspar from Auburn, Maine, is said to have brought $9 per ton on board ship at Bath.

DISTRIBUTION OF FELDSPAR IN THE UNITED STATES.

Since the feldspars are limited in their occurrence to the igneous or highly metamorphic rocks, the localities where they do not occur are fairly well defined. Although present in nearly all the igneous rocks and in such metamorphic rocks as mica schist, gneiss, granite and serpentine, it is only in limited areas where it is sufficiently segregated to be obtained in commercial quantities.

The source of supply in the United States is mainly the long narrow strip of schistose and gneissoid rocks extending along the Atlantic seaboard east of the Allegheny Mountains. The States which produce feldspar for the market are Maine, Connecticut, New York, Pennsylvania, Maryland, North Carolina, and Missouri. But the other Atlantic border States may become producers at any time. Of the States named, Pennsylvania and Connecticut are the largest producers. Feldspar occurs in large quantities in the granitic areas in the Western States, but the freight rates are so high that at the present low price of the mineral it cannot be shipped to the Eastern potters in competition with the Eastern product.

Carolinas, North and South.—Feldspar occurs as pegmatite veins in gneiss in several localities in both North and South Carolina, but none of the deposits have been developed as yet.

Connecticut.[*]—Connecticut has long been one of the principal feldspar producing States of the Union. Quarries, large and small, have been opened at various times in and near the towns of Middletown, Portland, Haddam, Chatham, and Glastonbury, the last being one of the most productive localities. C. H. Hall, of that place, has been in the business for 30 years, and for the last 15 years has operated a mill for grinding the spar. His quarry is said to have produced about 200,000 tons of feldspar and is still active.

At different points in the area there have been a great many small quarries

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[*] The writer is indebted to Prof. W. N. Rice, of Middletown, and to C. H. Hall, of South Glastonbury, for information concerning the Connecticut feldspars.

operated for a short time and then abandoned. It is not possible to obtain records of most of these, nor can it be ascertained with any accuracy how much spar is or has been produced in this way. Within the last few years a spar quarry has been operated in the northern part of Portland and another in the northeastern part of Haddam. The last one has derived considerable profit from the sale of green tourmaline obtained incidentally in mining the spar.

In all the quarries the spar has been obtained from pegmatite dikes occurring in gneisses and schists. Many of these dikes run parallel with the foliation, but some cut across it. The feldspar is chiefly orthoclase, but there is some associated albite. In most localities the characteristic graphic granite occurs, that is to say the intergrowth of quartz and feldspar, but in some localities there are large masses of nearly pure spar. There are many associated minerals, the most common of which are tourmaline, garnet and beryl.

Delaware.—Feldspar has been quarried near Wilmington and Hockessin, but none has been produced there for several years.

Maine.—Auburn is a productive locality, from which 500 tons of feldspar and a large quantity of quartz was shipped in 1896, about 800 tons of feldspar in 1897, and a still larger quantity in 1898. The product was shipped to East Liverpool and proved so satisfactory that an East Liverpool company bought and now operates the quarry. The quarry at Auburn was first worked for quartz which was shipped to Lynn, Mass., for making sand paper, and has been worked for feldspar only since 1896. The spar is said to occur in pockets rather than in a continuous vein. It is pure and remarkably white. There is considerable associated albite and much quartz, while some remarkably fine crystals of beryl have been obtained, one of which is said to have been 2·5 ft. in diameter and 15 ft. long. The quarry is situated on the east side of a hill known as Mt. Apatite, about 3 miles from Auburn. The feldspar costs $1·25 per ton to mine and 75 cents to haul to the railroad.* A Trenton company operates a spar quarry at Brunswick, but its product is small. Feldspar occurs at other localities, but so far as known is not quarried now. Feldspar has also been quarried at Georgetown, Topsham and North Turner.

Maryland.—This State has productive spar mines, which occur in the same belt of mica schists that contains the Pennsylvania quarries.

Massachusetts.—Feldspar occurs in several localities in Massachusetts, and small quantities have been marketed in previous years.

New Hampshire.—Feldspar is common in large masses between Surry and Easton, but so far as known it is not quarried for the market at present.

New York.†—There is a productive feldspar area in the vicinity of Bedford, Westchester County, about 40 miles north of New York City, near the Connecticut State line, where the spar occurs in pegmatite veins in an area of "Augen" gneiss, the strike of the veins corresponding to the general strike of the gneiss. The spar occurs in some places in comparatively pure masses, at others it is associated with much quartz, sometimes as large masses and sometimes as an intimate pegmatitic intergrowth. The feldspar

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* Private communication from N. B. Tracy.

† The writer is indebted to P. H. Kinkel, of Bedford, N. Y., and to an article by Drs. McI. Luquer and Ries in the *American Geologist*, for information regarding the Bedford feldspar.

varies from dark red to almost white in color, and the quartz is white, rose, and smoky. There are considerable quantities of muscovite and biotite segregated in different parts of the veins. Black tourmaline is common and seems to occur mostly in the quartz near its contact with the feldspar. The feldspar is orthoclase high in potash, as shown by chemical analyses as follows: Silica, 65·85%; alumina, 19·32%; ferric oxide, 0·24%; potash and soda, 14·10%; lime, 0·56%; magnesia, 0·08%. Feldspar has been quarried here since 1878. There were three quarries operated in 1896, and two in 1898, one of which operates a mill for grinding the spar.

*Pennsylvania.**—Pennsylvania is one of the leading States in the production of feldspar. There were 10 companies and individuals operating 15 quarries during 1898, and there are three other quarries that have been operated within the last few years. There are two mills in operation for grinding the spar, one of which is located at Brandywine Summit, in Delaware County, and the other at Toughkennamon, in Chester County, both on the Philadelphia, Wilmington & Baltimore division of the Pennsylvania Railroad.

The spar quarries are located as follows: Two active and three idle at Chester Heights, one idle near Glen Mills, one active near Elam post office, one active near Chelsea, all of which are in Delaware County; four active and two idle at Avondale, one active near Chatham, three active and several idle near Unionville, one active at Embreeville, two active near Sylmar, one near Fairville, and one near Chadd's Ford, all of which are in Chester County. The active quarries employ from three to ten men each, and produce from 1 to 10 tons per day each. They are nearly all worked as open pits, but in a few instances underground tunnels have been employed.

At Chester Heights the spar occurs apparently in a series of parallel veins having a N.E.-S.W. trend, in a mica schistose rock. The spar has been quarried from six different openings. Three openings made along one of the veins show it to have a width of 8 to 20 ft., narrowing with depth, the quarries extending to a depth of about 40 ft. The spar is in the pegmatite or binary granite, quite variable in character, being almost free from quartz in places and the quartz segregated in huge masses elsewhere. The material is shipped to the spar mill at Brandywine Summit.

The feldspar quarry known as the Sharpless quarry, about a mile southeast from Glen Mills, has been idle for several years. The spar is mixed with large quantities of muscovite, which occurs in large crystals, from 5 to 10 in. across the sheets, and quartz which occurs segregated in large masses.

The feldspar quarry southwest of Elam post office is the largest one in the State. It has been in operation for 15 years and it is estimated that not not less than 100,000 tons of feldspar have been removed from it in that time. The vein has a N.E.-S.W. course, and the open working shows the vein to be 60 or 70 ft. wide. It has been worked to a depth of 100 ft. There are some underground workings extending 300 ft. or more along the vein. The spar

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* The data on the Pennsylvania feldspar quarries is based on my field investigations during the summer of 1898, under the auspices of The Pennsylvania State College, and is here published in advance of the College Report with the consent of the president of the college.—T. C. H.

occurs in pegmatite together with considerable quartz and some mica in places. There are large segregations of quartz near the bottom of the opening. A steam hoist is used to lift the spar from the lower workings, but near the surface it is hauled out in carts on an inclined roadway, and is carried by wagon to the mill at Brandywine Summit.

Near Boothwyn Station, in Delaware County, is a feldspar quarry which has been in operation several years. The strike of this vein appears to be N.W.-S.E., and it dips about 60° S.W. The vein is about 30 ft. thick, and the open quarry (about 150 ft. long) has been worked to a depth of 40 ft. A steam hoist is used to lift the materials from the quarry. The product of the Delaware County spar quarries is controlled by the Brandywine Summit Kaolin and Feldspar Co., which also grinds some spar from Chester County.

The feldspar quarries near Unionville, which are controlled by companies outside of the State, are situated north and east of the abandoned corundum mines, and consist of numerous small openings scattered over a considerable area. Some are in the mica schists and some apparently in serpentine. None of the openings show a very decided vein structure in the spar, which appears to occur rather in somewhat irregular bodies, interspersed in some place with much quartz, and in some places with streaks of serpentine and patches of biotite and hydromicas. The work is all done by hand in surface openings, no hoists being used. The spar is shipped from Glen Hall station on the Wilmington & Northern Railway.

The spar quarry at Embreeville is on the Wilmington & Northern Railway, about 300 ft. north of the station, and occurs in a vein about 10 ft. thick. The spar is nearly snow-white, slightly stained by dendritic markings in places. It occurs in mica schists, which are very garnetiferous just south of the quarry.

The four active and the three idle feldspar quarries near Avondale are all in pegmatite veins of considerable extent. The veins are in mica schist, with limestone outcropping in proximity to some of the quarries. The spar has a more characteristic graphic granite structure than that in any of the other quarries. The spar is hauled by wagon to the railway station at Avondale.

The feldspar quarries near Sylmar* have a peculiar scientific interest in that they occur within the borders of a large serpentine area. Some of the Unionville quarries are apparently in serpentine, although the relation of the spar to the surrounding rock is not clearly shown, but the serpentine area is a small one and the quarries are so close to the mica schist that it might well be the source. But in the Sylmar locality the serpentine area is a large one and the quarries well within its borders. The spar is not a graphic granite, as there is no quartz present. It varies from flesh color to nearly snow-white and is stained with dendritic coatings in many places. The spar contains considerable green hornblende, some muscovite, a very little biotite, some hydromica, some talc, and some inclosed serpentine. In one of the quarries the vein walls and the contact of the spar and serpentine are clearly shown. Feldspar quarries are operated at Fairville and near Chadd's Ford, in Chester County.

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* Sylmar post office is in Maryland, the railway station is on the State line (hence the name), and the spar quarries are in Pennsylvania, nearly two miles northwest of the station.

FLUORSPAR.

The chief producer of fluorspar in the United States in 1898 was the Fluor Spar Co., of St. Louis, which operates mines in Kentucky at Crayneville and Annora and vicinity. This company, which erected a new grinding mill at Crayneville and increased largely its production, offers to deliver spar with a guarantee of 98·5% calcium fluoride. It has opened eight mines and reports that it has large bodies of mineral in sight. The Rosiclaire mines of Illinois, were closed down during part of the year and their output was considerably less than in 1897. The mines of Porter, Hudson & Co. in Kentucky were turned over to the Kentucky Fluorspar Co., which operated them during the year. A new company, called the Eagle Fluorspar Co., was organized to operate in Kentucky, and expects to be a producer in 1899.

PRODUCTION OF FLUORSPAR IN THE UNITED STATES. (IN SHORT TONS.)

Year.	Tons.	Value.	Per Ton	Year.	Tons.	Value.	Per Ton	Year.	Tons.	Value.	Per Ton
1887....	5,000	$20,000	$4·00	1891.....	6,320	$38,000	$6·00	1895	4,000	$24,000	$6·00
1888....	6,000	30,000	5·00	1892.....	9,000	54,000	6·00	1896.....	6,000	48,000	8·00
1889....	9,500	45,835	4·82	1893.....	9,700	63,050	6·50	1897.....	4,379	36,264	7·65
1890....	8,250	55,328	6·70	1894.....	6,400	38,400	6·00	1898.....	12,145	86,985	7·16

PRODUCTION OF FLUORSPAR IN THE PRINCIPAL COUNTRIES. (IN METRIC TONS.)

Year.	France.	Germany.					Spain.	United Kingdom.	United States.	Total.
		Anhalt.	Bavaria.	Saxe-Weimar.	Saxony.	Schwarzburg.				
1896...........	1.940	5.600	5.218	330	805	1,218	3	400	5.443	20,957
1897...........	2,722	7,000	4,904	278	592	641	2	302	8,142	24,583

[a] From the official reports of the respective countries except the United States, for which the totals are based on direct returns of the producers, and for Anhalt, Saxe-Weimar and Schwarzburg-Sonderhausen, which are due to the courtesy of Herr von Scheel, director des Kaiserlichen Statistischen Amts.

The production of fluorspar is not reported in the mineral statistics of the German Empire, nor are there any statistics for Prussia, although that kingdom is a producer. A large fluorspar mine is operated in the Duchy of Anhalt, of which the production is reported by the Chamber of Commerce at Dessau. In Saxe-Weimar and Schwarzburg-Sonderhausen fluorspar and barytes are mined in connection with manganese ore.

USES OF FLUORSPAR.

Fluorhydric Acid.—The consumption of fluorspar in the United States in the manufacture of fluorhydric acid increased considerably in 1898, this acid fast supplanting other acids for cleaning iron castings, and the business is growing so rapidly that it is expected to assume large proportions in the near future. Fluorhydric acid has an advantage over sulphuric acid inasmuch as it dissolves the sand on castings and has comparatively little effect on the metal, while sulphuric acid loosens the sand by dissolving the iron under it. Moreover fluorhydric acid is claimed to be a better solvent for rust or black oxide of iron than either sulphuric or chlorhydric acid. The usual strength of the acid used is 20 to 25 parts of water to one of acid, the latter being a 30% solution of the gas. This will clean ordinary castings in 30 minutes to one hour. On removal from the tanks the castings are washed in hot water, if it be desired to keep them bright, otherwise cold water may be used.

Fluorhydric acid is made in three different grades, containing 30, 48 and 52% gas. The first is shipped in wooden barrels containing about 300 lb., or lead carboys boxed in wood, containing about 100 lb. As soon as received at their destination the barrels should be emptied into lead lined, or asphaltum coated, tanks. Care is required in handling strong fluorhydric acid, since it causes a painful inflammation whenever it comes into contact with the human skin. If this happens the skin should be washed immediately with water and dilute aqua ammonia, which will prevent any injury, but it is better to avoid danger by using rubber gloves in handling the acid.

Fluorspar as a Flux in Smelting.—The advantages of fluorspar as a metallurgical flux have been pointed out again by Dr. Foehr, who asserts* that it is especially useful (1) in smelting ferrosilicon and ferromanganese, (2) in the basic open hearth steel furnace, where it makes the slag more fluid, and helps to carry off the phosphorus and sulphur, and (3) in the foundry cupola, where it makes the iron more fluid, permits the use of greater quantities of low grade pig and scrap and reduces impurities.

In smelting, fluorspar takes the place of lime, than which it is more energetic, the reaction in each case being shown by the following equations:

(*I.*) $2CaF_2+3SiO_2=2CaSiO_3+SiF$. (*II.*) $3CaCO_3+3SiO_2=3CaSiO_3+3CO_2$.

Dr. Foehr claims that fluorspar forms only two parts of slag where limestone forms three, and it probably forms fluorsilicates, whereby heat is liberated. Ferrosilicon with 10% Si may be obtained by smelting silicious ores in an ordinary blast furnace if these be fluxed with fluorspar and the slag is strongly basic. The spar first reduces the silica, forming fluorsilicon, which is in turn reduced to silicon by the hydrogen of the furnace gases and possibly also by coke. In the basic open hearth steel process the fluorspar serves only to render the slag more fluid, not only on the hearth but also in the ladle and mold. In the foundry cupola fluorspar accomplishes all that limestone does in fluxing the ash of the fuel, and also has an important effect on the iron, keeping it gray and soft. In foundry work the best results are obtained with 0·33% fluorspar, while with more than 0·5% the results are inferior.

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* *Chemiker Zeitung*, 1898.

FULLERS EARTH.

As in 1897 the domestic production of fullers earth in 1898 was derived chiefly from Florida, the deposits of that State being the only ones in the United States which have so far proved to have important commercial value. The output of Florida has been derived heretofore chiefly from the vicinity of Quincy, but in 1897 a peculiar chocolate brown earth was discovered near Ocala which is claimed to be superior as a filtering and clarifying agent for oils to all other American and English fullers earth. This material, which resembles fullers earth in chemical composition, differs from it by carrying persistently a small percentage of aluminum phosphate, while it is rendered impure by a content of sand and a small amount of organic matter, its brown color being due probably to the latter. So far there has been no production from these deposits. Samples have been submitted to the trade and there has been some doubt expressed as to its suitability for certain of the purposes for which ordinary fullers earth is employed.

In general the quality of the fullers earth from Florida is very good and it is said that this product has practically superseded bone black in the clarification of mineral oils, in which work it is said also to be more efficient than the imported earth, although it is unable to compete with the latter in the clarification, deodorization and bleaching of animal and vegetable oils, or in the fulling of wool. There is very little earth used for the last purpose by American woolen manufacturers, although the superiority of English woolen goods is said by some to be due largely to the free use of it, and the demand for fullers earth from the American woollen trade is likely to increase.

The use of fullers earth in clarifying oils was discovered in America and for a while mills were in operation here to prepare the powdered mineral, but these were gradually closed down, as it proved cheaper to import the powder from England. Under the Dingley bill fullers earth in lumps pay a duty of $1·50 a ton and ground earth $3. In purifying and bleaching the yellow cottonseed oil resulting from the first refining process between 2 and 3% of fullers earth is used.

A new discovery of fullers earth was made in 1898 at Poso Creek, 17 miles north of Bakersfield, Cal., where a large deposit was opened. About 100 tons were shipped to different packing houses in California, who pronounced it an earth of good quality for filtering and bleaching purposes. Plans are being made to work this deposit in 1899.

GARNET.

THE production of garnet in the United States in 1898 was 2,882 short tons ($82,930) against 2,261 ($66,353) in 1897. Of the production in 1898 the mines at North Creek, in the Adirondacks, N. Y., furnished 1,686 tons against 1,050 in the previous year, the remainder in each year coming from Chelsea, Pa., and Roxbury, Conn. The average value of the product in 1898 was $28·78 per 2,000 lb. against $29·35 in the previous year. The number of producers was increased in 1898 by the reopening of the Moore mine at North Creek, N. Y., which had lain idle for a number of years. The output of garnet in the United States could be increased easily, but the limited demand for the mineral, which is used only as an abrasive, inferior to corundum and emery and superior to quartz, would soon bring prices down to a point where mining would be unprofitable. Our statistics of production do not include a certain amount of garnet which is produced in North Carolina in connection with the corundum of that State, some of the producers there shipping a garnet-corundum mixture. This product, which is small in amount, is included in our statistics of corundum production.

The Adirondack garnet deposits were described in THE MINERAL INDUSTRY, Vol. VI., to which reference should be made. Prof. T. C. Hopkins, of State College, Pa., contributes the following notes concerning the mines in Pennsylvania. These are situated about one mile west of Chelsea, Delaware County. There are two mines in active operation, one owned and operated by Herman Behr, of New York, and the other owned by Mr. Behr but operated by a lessee who sells his output to the lessor. In both mines the mineral is almandite, the common iron garnet, which occurs in imperfect crystals richly impregnating a mica schist. Near the surface the rock is soft and the garnet can be separated readily from it, but at a depth of 20 ft. the rock becomes solid and the garnets so firmly imbedded in it that they break to pieces without separating from it. The garnets form about 75% of the rock mass.

The mineral is washed as it comes from the mine and is then crushed and rewashed, the garnet being separated from the gangue by virtue of its greater specific gravity. It is screened at the mine into four sizes, which products are shipped to works in New York where they are crushed again and screened into 28 sizes, which are employed in making different grades of garnet paper. Each mine employs six to eight men. There are many places in Delaware and Chester counties, Pa., where garnet is abundant in the mica schist, but Chelsea is the only locality where the mineral is exploited.

GEMS AND PRECIOUS STONES.

The only features of interest in the mining of gems and precious stones in the United States in 1898 were connected with the production of ruby and turquoise, which are engaging considerable attention. Elsewhere in the world the main supply of diamonds continued to come from South Africa and ruby from Burma, while Persia remained the chief source of turquoise.

Amber.—Amber, the mineral succinite, is produced chiefly in Germany, where the production in 1896 was 440 metric tons, which was about 100 more than in 1895. The mines and business which were formerly in the hands of Stantien & Becker have now been taken over by the Prussian government. A small amount of amber is obtained from Myitkyina in the Hukong Valley of the Bhamo district of Upper Burma, where it is otained by mining at depths of 70 to 80 ft. Lumps weighing as much as 1,000 lb. are found sometimes.

Diamonds.—The De Beers Consolidated Mines, Ltd., reported for the year ended June 30, 1898, receipts of £3,647,874 13s. 11d., from diamonds sold, and an expenditure of £1,870,079, including amounts written off for depreciation of machinery, redemption of debentures, etc. Dividends amounting to £1,579,582 were paid and £748,488 6s. 7d. was carried forward. The average yield per load for the De Beers and Kimberley mines was 0·80 carat, worth 21s. 21d., or 26s. 62d. per carat; for the Premier mine the yield was 0·27 carat, worth 5s. 8·22d., or 20s. 9·3d. per carat. The price received for diamonds showed a slight decrease as compared with the previous year. The amount of blue ground and lumps on the floor, June 30, 1898, was 3,619,945 loads, including 729,039 at the Premier mine. At the annual meeting of the company at Kimberley, December 19, the Right Hon. C. J. Rhodes, quoting Gardner Williams, the general manager, as to the condition of the mines, said: "Although there was a falling off in the yield in 1898 this was due only to carelessness of the contractors in the mine in separating waste and blue gravel; and the fault has been remedied. At the De Beers mine an area of blue ground 730×200 ft., which should yield 1 carat per load is being dug on the 1,200-ft. level. Besides this there is almost an equal area in the west end of the mine which has been called poor heretofore, but will probably pay, and is to be tested thoroughly. The Wesselton mine is yielding nearly 32 carats per

100 loads and paying between £300,000 and £400,000 per annum. The position of the De Beers Co. is highly satisfactory. The net return for the six months ending December 31, 1898, was about £924,000. Expenses are from £150,000 to £160,000 per month, of which £100,000 is for mining and the remainder for sinking fund, interest and other charges. The company is receiving 20s. 8d. per carat for Kimberley and De Beers diamonds and 20s. 6d. for Wesselton, but will get a better price in 1899. The entire product is sold to the syndicate which controls the diamond-cutting industry. This syndicate imposes upon the company the condition of not putting diamonds unnecessarily upon the market, wherefore the company has no intention of increasing its output."

The production of De Beers Company in 1897-98 was 2,753,000 carats, from 3,260,000 loads (725 kg. per load) against 2,769,000 carats, from 3,011,000 loads in the previous year.

The De Beers Co. keeps a close watch upon all new discoveries of diamonds and considers that through its agents it has better opportunity than any outsider to secure control of new and valuable finds. During 1898 an examination was made of the Rivas mine in the Orange Free State, which proved to contain diamonds, but in too small quantities to make the undertaking attractive. Reports were also received as to the Schuller mine, near Pretoria, and its competition is not feared, although, it may be remarked, the feeling between Mr. Rhodes and the Transvaal government are not of a nature to permit close relations toward the investigation or acquisition of a rival diamond producer in the South African Republic.

The Jagersfontein mine, in the Orange Free State, produced in 1897-98 232,433 carats from 2,421,503 loads (725 kg.), against 220,212 (from 2,147,427 loads) in 1896-97, and 205,000 (from 1,905,000 loads) in 1895-96. The Rivas diamond mine, in the Orange Free State, also made a small production in 1898.

Upward of 10,000 carats of diamonds from the Schuller mine were exhibited in Johannesburg in December, which were described as small and of poor color. The mine is said to show two pipes, 2 furlongs apart, one 630 ft. in diameter, yielding 63 carats per 100 loads, and the other 550 ft. yielding 42 carats per 100. The mine, known also as the Rietfontein, and owned by Lewis & Marks, has been tested by drilling to a depth of 500 ft. The stones are usually covered with a reddish film, which is removed by boiling in acid. Previous to November the largest stone that had been found was of 57·75 carats. About a mile west, J. B. Robinson has a mine, called the Montrose, on which he is erecting machinery. About 5 miles north the Bynestpoort Exploration and Developing Syndicate has tested an alluvial stratum about 2·5 ft. thick, which is said to yield an average of 3·5 carats per load. The diamonds are not of very high quality the average produced being worth only 15 to 17s. per carat; a plant of 50 loads per day capacity is being erected It is not yet two years since the first diamond was discovered in the Pretoria district. The Transvaal government has already promulgated a new law, considered rather onerous, to regulate the new industry.

The total production of the Schuller diamond mines in 1898 was registered as 10,823 carats, valued at £9,192. This production was obtained from 15,000 loads of red soil, representing the top stratum about 2 or 3 ft. thick on the easternmost of the two pipes mentioned above. This quantity of gravel was treated by one machine, having a capacity of 300 to 500 loads per day, according to the nature of the gravel. Below the top stratum there is a stratum of yellow soil, 30 ft. thick, and below the latter is the blue ground of the pipe. Blue ground is now being "floored" preparatory to washing. The actual production of the mine in 1898 was larger than reported, a good many stones of the better class being known to have been stolen.

For a description of the practice in washing diamondiferous gravel in South Africa the reader is referred to the paper on "Progress in Ore Dressing and Gold Milling," by R. H. Richards, elsewhere in this volume. The diamondiferous rocks of Monastery, in the Orange Free State, 30 miles south of Winburg and 140 miles from the Jagersfontein mine, and those of Griqualand, were described by A. Lacroix in the *Bulletin de la Société Francaise de Minéralogie*, 1898, Vol. XXI., pp. 21–29.

An exploration party sent recently by the Portuguese government to the country lying between the Tembe and Umbeluzi Rivers, in Portuguese South Africa, has located diamondiferous ground over a large area. The announcement of this discovery caused considerable excitement at Lourenço Marques. Considerable interest was excited in 1898 by some new finds in the Nullagine district, situated at the headwaters of the Congoon River and not very far from the Pilbarra gold field in northwest Australia. Diamonds were first discovered in this district in 1891, in connection with the gold ores of Pilbarra. A small consignment was shipped in 1897. The diamonds are said to occur in a conglomerate in which the stratification resembles the South African banket reefs. Diamond fields have been located in the Province of Shantung, China, and it is said that they are worth development.

Although Brazil was once the most important diamond-producing country in the world, and the excellent quality of the stones makes them highly valuable, diamond mining has been comparatively neglected of late in that country, being carried on only by individuals or small associations working in a crude manner. The production at present is probably about 40,000 carats a year.* The larger part of the stones found in Brazil at the present time is cut in that country either at Diamantina, Gouvea, or Rio de Janeiro. A new company the (Boa Vista) has lately been organized in Paris with a capital of 2,000,000 fr. to carry on systematic mining on a large scale in the neighborhood of Diamantina. This company intends to exploit what is believed to be the original deposit from which the diamonds were washed into the Santa Maria River, an affluent of the Jequitinonha, the most famous of all the diamond rivers of Brazil.

Diamond Cutting.—About 50% of the world's production of diamonds goes to Belgium for cutting, the remainder being done chiefly in Amsterdam. The Belgian and Dutch cutters buy the rough stones in London, whither they are

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* British Consular Report, Miscellaneous Series, No. 494, February, 1899.

shipped from South Africa. The value of the rough diamonds purchased by the Belgian cutters and the wages paid to their employees since 1893 have been as follows, the wages being given in brackets after the value of the rough stones: 1894, $7,750,000, ($772,000); 1895, $9,650,000 ($965,000); 1896, ($1,061,500); 1897, $10,615,000 ($1,061,50).

Carbonado, or bort, the black diamond which is used in diamond drills and for other abrasive purposes, is found commercially only in Brazil. The occurrence and industry there are described by H. W. Furniss, United States Consul at Bahia,* who reports that they are found in the interior of the State of Bahia, in a district which is reached, from Bahia by boat to St. Felix and thence by rail to Bandeira de Mello, the edge of the diamond region. The carbonado is always found near the gem diamonds. The most productive region is further up the Paraguaçu River, which is reached by a mule-back journey of two days. It is thought that diamonds and carbonado occur all through this section, but on account of the rude methods of mining only the bed of the Paraguacu and its tributary the San Antonio and the side of the Serra das Lavras Damantinas are worked. The carbons are found in a kind of gravel called "cascalho," which occurs in the river beds beneath the silt and on top of a stratum of clay; on the mountain sides beneath a stratum of rock and above the same stratum of clay; and in the surrounding country beneath several strata of earth. The carbons are obtained from the river beds by planting a pole in a place not more than 20 ft. deep, where the current is not too rapid, up and down which native divers work, removing all the gravel down to the clay by means of sacks. This work is carried on during the six months of the dry season, being abandoned necessarily at the beginning of the rains on account of the strong current and greater depth of the river. During the rainy weather the gravel previously got is washed and examined for carbons and diamonds. The land deposits are worked by short adits, the accumulation of gravel being washed during the rainy season. These are more productive, being more accessible, than the river bottoms. Carbons vary in size up to 975 carats, which was the weight of one discovered in 1894; it sold in Paris for 100,000 fr. The most useful are those weighing from 1 to 3 carats. Larger stones have to be broken, in which there is always a loss, since they have no cleavage planes. About two years ago there was a local combination to keep up prices, but this failed, and since then the price has been maintained solely by the large demand, small supply and laborious method of mining. The small supply is due to the crude methods. Frequently two men obtain only three or four carbons in six months' work. The carbons are bought by agents of the exporters in Bahia.

QUARTZ.—In THE MINERAL INDUSTRY, Vol. VI., p. 256, reference was made to the discovery of large quartz crystals in the Green Mountain mine, Chile Gulch, Calaveras County, Cal. During 1898 a sphere entirely free from flaws 5·5 in. in diameter was cut from one of these crystals by Tiffany & Co. This is the largest perfect sphere ever made from an American crystal. It is valued at $3,000.

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* United States Consular Reports, December, 1898.

RUBY AND SAPPHIRE.—Gem mining was prosecuted actively in 1898 at the ruby mines of the American Prospecting & Mining Co., in Macon County, N. C. A good many inferior sapphires and rubies were found there, and one pigeon-blood ruby weighing five carats. The New Mine Sapphire Syndicate was organized to exploit sapphire-bearing gravel along Yogo Creek, Mont.

The world's chief supply of ruby continues to come from Burma, where the most important of the deposits (described in THE MINERAL INDUSTRY, Vol. V.) are worked by the Burma Ruby Mines Co. Ltd. In 1895 this company produced 56,108 carats, valued at 2,91,357 rupees. In 1896 the outturn yielded 6,87,537 rupees, the expenses of the company being 4,31,892 rupees. In 1897 the yield was 8,02,452 and the expenses 6,78,934 rupees. Some excitement arose early in 1898 over the discovery of sapphires in the Muang Long district, on the right bank of the Me Yom River, in Siam.

Rubies have long been esteemed as the most valuable of all gem stones. At the present time rubies under half a carat in weight cost £4 to £10 a carat, while rubies of four carats fetch £400 to £450. Stones of greater weight than four carats are of such exceptional occurrence that they command fancy prices. The two most important rubies ever known in Europe were brought from Burma in 1875, one weighing 37 and the other 47 carats. The smaller stone, it is said, brought £10,000 and the larger one £20,000.

Artificial rubies were regularly on the market in London and Paris in 1898. Some gem experts said that they could not distinguish them from the natural stone; others, including Mr. Claremont, who is the contributor of a paper in this volume, said that there was no difficulty in making the distinction. The stones are made by a company known as the Maiche, Ltd., exploying a process invented by Louis Maiche, which claims to be making a profit of £37,000 per annum. If this be true it is obvious that a large number of the artificial gems, which are said to be beautiful, have already been marketed.

TURQUOISE.—A recent report of the British consul-general at Meshed, Persia, says that the turquoise mines near Nishapur in Khorassan are at present held by the Malik-ut-Tujjar (head of the merchants) of Khorassan at a yearly rental of 24,000 tumans (about $24,000). There is great uncertainty about the continuity of the contracts, which naturally prevents any good work or improvements in the mines. The production can only be estimated in a very vague way, as there are no records whatever.

There was considerable activity in turquoise mining in Arizona in 1898, an especially important development being made at Turquoise Mountain, in Mohave County, where ancient Aztec or Indian workings were reopened. According to a report by A. B. Frenzel, veins of turquoise showed from the commencement of operations, increasing in width as the work progressed. Near the surface the stone was of poor color and quality, but as depth was gained the rock became firm and the turquoise solid and of better color, giving evidence of permanency. There is said to be a good deal of stone here which can be worked up to advantage in mantelpieces, columns, bases for clocks and bronzes, pedestals, mosaics, etc. This material is the country rock intersected by veins of turquoise. The mines are 18 miles from Kingman.

THE IDENTIFICATION OF GEMS.

BY LEOPOLD CLAREMONT.

IT often happens that a dealer in gems, who has gained his experience by constantly handling large quantities of precious stones in the rough and cut states, is able to settle in a few moments some important question of dispute which it would take a clever mineralogist as many hours to decide, and it is surprising to what a degree of keenness the senses can be educated by the continual concentration of the faculties upon minute details, enabling one not only to discriminate between gems of different kinds, but to pick out, say, from a parcel of cut diamonds weighing several thousands of carats, a single stone which owes its origin to a different mine from the remainder of the parcel. It is, however, impossible to overestimate the value of the methods of testing precious stones which have been placed at our disposal by the science of mineralogy, by the application of which the keen judgment of the expert should be indorsed and made doubly sure.

Before applying any test at all, the gem to be examined—unless it be a turquoise or an opal—should first be carefully studied with a view to discovering if possible the form in which it crystallized, since many gems which somewhat resemble each other in many other respects occur in forms appertaining to different systems of crystallography. Unfortunately it is not always possible to discover more than a trace of the natural geometric formation, owing to the fact that precious stones are frequently found in more or less broken or waterworn fragments, which, especially in the case of highly modified forms, renders them not easily recognizable. At the same time careful note should be made of the fracture, indications of cleavage, and direction and nature of striæ, all of which help in the diagnosis.

Specific Gravity.—The specific gravity and the hardness should then be tested, after which the optical properties, which form a study in themselves. Following will be found a list of some of the gem stones, with their respective specific gravity, viz.: Diamond, 3·5; ruby, asteria and sapphire (blue, yellow, purple and green), 4·0; chrysoberyl, alexandrite, cymophane and catseye, 3·7; spinel, ceylonite and "Balas ruby," 3·6; topaz, 3·5; emerald, beryl and aquamarine, 2·7; jargoon, jacenth, hyacinth and zircon, 4·6; pyrope and almandine garnet, 4·2; essonite, 3·66; tourmaline, 3·1; amethyst, cairngorm and rock crystal, 2·6; chrysolite, peridot and olivine, 3·3; turquoise, 2·7; opal, 2·2; hiddenite and spodumene, 3·2; iolite, 2·6; phenakite, 2·9.

All the mechanical methods formerly used for taking the specific gravity of precious stones by means of balances, etc., are too laborious and troublesome to be of much assistance, and cannot be compared with the simple method of immersing the doubtful stone in different liquids of known degrees of density. A liquid suitable for this purpose is methylene iodide, which can easily be diluted by the addition of benzine, thereby diminishing the density, and hence by a careful adjustment of their relative proportions the specific gravity can be regulated to a nicety, so that the density of even the most minute stone can be ascertained in a few moments; for instance, if it be desired to discover if a certain doubtful gem is chrysoberyl or beryl, all that is necessary is to place it in a tube of methylene iodide together with a small fragment known to be beryl, to serve as an index. If it be beryl (2·7) it will float, if it be chrysoberyl (3·6) it will sink immediately. In the former case, if the fluid be diluted and stirred until the index is exactly suspended, the doubtful gem will also neither float nor sink, but remain poised beside it. The great drawback to this method is that many of the gem stones are heavier than methylene iodide.

Another liquid used for determining the specific gravity of precious stones, and one which I find most convenient, is known as "Sonstadt's solution." It consists of a solution of mercuric iodide and potassium iodide, and should be made up in several different vessels to different degrees of density, in which the gems will either float or sink, according to their relative weight, viz.: Solution A=3·17, solution B=2·9, solution C=2·67, and solution D=2·63. In solution A, spodumene, diamond, diopside, topaz, spinel, peridot, zircon, chrysoberyl, garnet, ruby, sapphire (and all other varieties of corundum gems) sink; and tourmaline, phenakite, turquoise, labradorite, beryl, emerald, quartz, iolite and adularia float. In solution B, tourmaline, turquoise and phenakite, which float in A, sink. In solution C, labradorite, beryl and emerald sink; and in D, quartz and iolite sink, but adularia floats. "Sonstadt's solution" should be handled with great care, as it is highly corrosive and very poisonous. It will be seen that this solution also is not sufficiently dense to float some of the heavier stones, but this difficulty has been removed by the discovery of a colorless solid compound which melts, at a fairly low temperature, to a clear liquid five times as dense as water, and therefore sufficiently dense to float all known precious stones. This compound is a double nitrate of silver and thallium, and possesses the remarkable property of mixing in any desired proportions with warm water, so that by dilution the specific gravity can be easily regulated.

Preparation of Specific Gravity Solutions.—Sonstadt's solution is easily prepared by making a saturated solution of potassium iodide in distilled water and adding to it mercuric iodide until no more is dissolved. Portions of the dense fluid thus obtained may then be diluted with water till they have the specific gravity required. Three fluid ounces (or say 100 c.c.) is a convenient minimum quantity for each solution, and assuming that four are to be prepared, take 360 c.c. of distilled water and add 415 g. of potassium iodide, and 567 g. of mercuric iodide, which should be previously powdered and mixed together. Allow to stand for some days, shaking at frequent intervals; then filter the liquid to remove any suspended particles, and ascertain if the specific gravity exceeds 3 by dropping in a gem, such as chrysoberyl, of about that density, which should float. If it does not, small portions of potassium iodide and mercuric iodide previously mixed and powdered in a mortar must be added till the required gravity has been attained. The whole bulk of liquid is then divided into four portions, each of which must be separately adjusted to the particular density required. This is a somewhat tedious and troublesome process, especially if any great degree of accuracy is desired; but since gems of the same nature may vary somewhat widely in density, an error not exceeding ·005 may be disregarded. The most accurate way of adjusting the solution is by means of a Sprengel tube, or a specific gravity bottle, preferably the former, on account of the high coefficient of expansion of the liquid and its great weight, which makes it desirable to work with a smaller quantity of material than most pyknometers are made to hold. Having ascertained the density, and assuming that it is too great, the proportion of water which must be added to get the density wanted may be calculated with approximate accuracy by means of the following formula:

Let x=the number of parts by volume of the liquid which must be added to reduce the gravity to the required extent. Let s'= the specific gravity of the solution, and let s''=the specific gravity to which it is wished to reduce it. Then $x=(s''-1)\div(s'-s'')$.

Having diluted in accordance with the above calculation, take the density once more. After one or perhaps two further adjustments the gravity required will be obtained with all necessary exactness. However, instead of directly ascertaining the specific gravity of the solutions, they may be adjusted with all reasonable accuracy by dropping in gems of known specific gravity, and adding water, as in the case of the silver-thallium compound about to be described. If the gems of the right density for each solution are at hand, this is a much quicker method than that previously described, but it generally necessitates taking the gravity of a large number of stones before suitable ones can be found.

Sonstadt's solution, though not so subject to loss by evaporation as the silver-thallium preparation, and not acted upon by organic matter under the influence of light, nevertheless slowly changes in density by keeping, however well stoppered the bottles in which it is kept may be. It is therefore desirable to have some rapid means of ascertaining whether it is up to strength. For this purpose a pair of gems may be kept in each solution, one which just floats and one which just sinks when the respective solution is at its correct strength. If no such gems are available any maker of scientific apparatus will, if supplied with solutions of correct gravity, prepare glass bulbs weighted with mercury for use instead of the stones. By careful adjustment it is possible to make them so that they neither sink nor float, but remain in almost any position in which they may be placed when the liquid is at the right temperature; but of course any considerable departure from the temperature at which the densities have been adjusted will determine the sinking or floating of the bulb.

The double thallium-silver nitrate may be made from the nitrates of silver and of thallium, both of which can now be purchased in sufficiently pure condition. For actual working purposes one fluid ounce of each solution is a sufficiently large quantity, but if expense is no great object more than this should be prepared, as the liquids rapidly lose water owing to the high temperature (70–100° C.) at which they have to be employed, and the density continually increases, but of course less rapidly with a large than with a small bulk.

Assuming that two solutions of about 60 c.c. each are to be made, having a density of 4·0 and 3·5 respectively, weigh out carefully 170 g. of silver nitrate and 265 g. of thallium nitrate, dissolve separately in distilled water, mix the two solutions in a porcelain basin and evaporate on a water bath, keeping the liquid protected as much as possible from both light and dust. The concentration should be continued until a stone (*e.g.*, a topaz or a garnet) having a carefully determined sp. gr. of 3·5, or a very little below, just floats. A little more than half the total volume should be set aside and the remainder further evaporated until a gem, such as a jargoon, with an observed density of 4 or a trifle below, rises to the surface.

Since so high a temperature has to be employed it is absolutely necessary to frequently test the density of the liquids, and this is best done by keeping in them a pair of gems as recommended for Sonstadt's solution. For convenience in working the latter is best kept in very wide-mouthed stoppered bottles, with no constriction at the neck. A pair of steel or ebonite forceps may be used to remove the gems, and a spare bottle containing distilled water should always be at hand in which to dip both gems and forceps after they have been withdrawn from the solution. Of course both should be wiped dry before being immersed in a fresh solution, otherwise the latter would be diluted and so altered in density by the adherent moisture. The water containing the washings should of course be used over and over again, as it can be filtered and evaporated down to the right density to replenish the solutions when they require it.

The silver-thallium solution is best kept in little flasks with wide and short necks, kept covered by a watch glass. When not in use each flask with its watch glass should be put into a wide-mouthed stoppered bottle with a little cotton wool at the bottom to prevent it shaking about. Before use the flasks and contents must be warmed to a temperature a little over 70° C., until the crystals have entirely dissolved, and the clear liquid must be made homogeneous by gently shaking the flask so as to impart, without splashing, a circular motion to its contents. The heating is best carried out in a small hot air or water oven with a false bottom, on which the flasks can rest, and two glass doors on opposite sides, so that the solutions are easily seen. For greater convenience there should be circular holes (with copper lids) in the top, sufficiently wide to admit a flask with the watch glass resting on it. When the solutions are quite clear and homogeneous the gems may be introduced through the top of the oven, removing the little lid and then the watch glass, the latter always being replaced immediately. As a cold stone dropped into a solution is liable to cool it down and to become coated over with crystals, it is as well to let the gems to be examined remain in, or on the top of, the oven a little while before they are tested. A pair of steel forceps should be used, and the coating of silver-thallium nitrate should be rinsed off the forceps and gems, as in the case of Sonstadt's solution. This is best done in warm water contained in a small flask similar to that in which the solutions are held. On account of its greater cost the washings are of course even more worth keeping than in the case of Sonstadt's solution. Great care must be taken to avoid getting the liquid on the hands, since no washing will remove the black stains it produces.

Hardness.—By the hardness of precious stones is meant the degree which they possess of resisting abrasion. Many of the hardest minerals cannot be scratched, although they can be easily broken, fractured or chipped. Emeralds, diamonds and jargoons, for instance, have often been injured by a blow or a fall. This quality of hardness forms another very valuable means of identifying precious stones, which, however, must be applied with discretion and care. There are 10 convenient standards of hardness used by mineralogists, to which specimens under examination may be compared, as follows: Diamond 10, sapphire 9, topaz 8, quartz 7, feldspar 6, apatite 5, fluorite 4, calcite 3, rock salt 2, talc 1. Small pieces of these minerals cut into points and suitably mounted in handles are applied in succession to the doubtful gem, with the object of attempting to scratch it.

When the gem neither scratches nor is scratched by any member of the scale the two stones may be considered of the same hardness. When it is scratched by the harder and scratches the softer of two test stones a good idea of its position between them may be gained by drawing all three stones with slight pressure across the surface of a fine, clean, hard file, and carefully noticing the different degrees of resistance to abrasion and the sounds produced. It requires some practice to acquire the delicacy of touch necessary to successfully perform this operation. The tools used in the process of cutting and polishing precious gems enable experienced gem cutters to appreciate the quality of hardness to a nicety, and thus dispense altogether with the system of scratching specimens with test stones. The following list shows the relative position of various gems in the scale of hardness: Diamond 10, sapphire 9·0, ruby 8·8, chrysoberyl 8.5, spinel 8·0, aquamarine 8·0, topaz 8·0, zircon 7·8, emerald 7·8, tourmaline 7·5, phenakite 7·5, iolite 7·3, almandine garnet 7·3, essonite 7·0, amethyst 7·0, peridot 6·3, adularia 6·3, green garnet 6·0, opal 6·0, turquoise 6·0, lapis lazuli 5·2.

Optical Properties.—These form most valuable means of identifying precious stones, and there are several scientific methods of a more or less practical nature which can be applied with the utmost advantage in this connection. These tests depend in the main part upon the refraction, single and double, and the absorption of light during its passage through the gems under examination. It will be obvious that the natural broken and scratched surface of a rough stone is very detrimental to the successful use of some of the instruments about to be described; therefore it is in connection with the discrimination of cut and polished gems that they are introduced into this paper.

Refraction.—The extent to which light is refracted by a translucent precious stone is a characteristic property most useful in determination. In the case of a facetted gem there is no need whatever to cut the stone into a prism, as was formerly supposed to be necessary, which, if the stone were of value, would be absolute madness. It is always possible to find two of the facets forming a convenient angle, and after carefully painting over the remainder of the gem to trace the ray of light passing through these facets, and thus to measure by means of the goniometer, not only the refraction, but the double refraction of the gem, no matter how great its refractive power. The little instrument known as the reflectometer is also useful for the purpose. It consists of a hemispherical glass lens viewed by an eyepiece containing a graduated scale

It only requires to be pressed against the gem upon which has been previously placed a drop of liquid possessing a higher refractive power than itself, such as monobromonapthalene. On looking into the eyepiece a shadow is seen over half the field of view, and its edge crosses the scale at a point which gives the exact refractive index of the stone. This method, however, is only useful to examine gems of low refractive power.

The results of the measurements of the indices (for the yellow ray) of refraction of a few transparent gems are here given: Diamond 2·75, jargoon 1·95, ruby 1·779, white sapphire 1·75, phenakite 1·675, topaz 1·621, aquamarine 1·598, rock crystal 1·547.

Pleiochroism, or the property possessed by colored, doubly refractive gems of appearing of different colors, and different degrees of density of color, when viewed in different directions of the crystal, is most easily recognized by the dichroiscope, although the effect is often observable to the unaided eye. The instrument consists of a cylinder 2 in. long and 1 in. in diameter, containing a cleaved crystal of Iceland spar. When the instrument is held to the eye two images of the square opening of the eyepiece are to be seen, which appear of different colors, or of identically the same color, according to the nature of the precious stone.

If a gem under examination with the dichroiscope shows two images of the square opening of the instrument identical in color, no matter in which direction it is viewed, the stone is singly refractive and may be a garnet, spinel or diamond, but cannot be a ruby, sapphire, topaz, chrysoberyl, aquamarine, emerald, or any other doubly refractive gem, all of which show twin colors of distinctly different hues from each other when viewed at right angles to the principal axis of the crystal.

This is a very convenient and easy means of distinguishing gems occurring in the cubic form from those belonging to the other systems of crystallization, and is also a test which can be quickly applied to a doubtful specimen, either in the rough or cut state, enabling one often at a glance to decide between gems of the greatest value and others which are comparatively worthless. The characteristic twin colors of a few doubly refractive gems will prove of interest, the prime or natural color being mentioned first and the twin colors immediately following in parentheses: Ruby, red (crimson and carmine); sapphire, blue (dull green and blue); tourmaline, green (chartreuse green and bluish green); tourmaline, brown (dark orange and greenish yellow); tourmaline, red (salmon and rose pink); tourmaline, brownish red (mahogany brown and red); tourmaline, blue (green and blue); emerald, green (yellowish green and bluish green); peridot, green (brownish yellow and sea green); topaz, yellow (sherry yellow and pink); aquamarine, green (straw and grayish blue); chrysoberyl, yellow (golden yellow and greenish yellow); iolite, violet (drab and blue).

The so-called interference figures seen on looking through a doubly refractive translucent stone, by means of a polarizing microscope, are very characteristic, and this method deserves to be used as a test for precious stones much more than it is at present. It is necessary to look through the stones in one given direction (parallel to the principal axis), and as there sometimes exists a great difficulty in finding the exact direction in a cut and polished gem, which might possibly be from one projecting angle to another, this test is not often applied. This difficulty, due to refraction, may, however, be surmounted by immersing the gem in some liquid having nearly the same refractive power as itself, such as oil or glycerine, contained in a spherical glass bottle, whereby the stone may be easily held in almost any position, and the interference figure can always be seen.

The spectroscope may be brought into use in distinguishing gem stones, as there are two gems which give different absorption bands across the spectrum, namely, jargoon and almandine garnet or carbuncle, and the effect can be seen even with a pocket instrument.

Electrical Manifestations.—If a crystal of tourmaline be warmed it is electrified, one end becoming positive and the other negative. Advantage has been taken of this fact to make a very pretty test for the stone. If a mixture of powdered red lead and sulphur be blown through a sieve the particles become electrified by mutual friction, and if it then be shaken over a tourmaline which is being warmed the positively electrified end of the crystal attracts the negatively electrified yellow sulphur and the negatively electrified end of the crystal attracts the positively electrified red lead; thus one end of the stone becomes red and the other yellow. This ingenious test can of course be applied either to a rough crystal of tourmaline or to a cut and polished gem mounted as a jewel.

Color.—In the discrimination of gems it is always advisable to apply as many different tests as possible to every doubtful gem, and to be guided by the results of them all, thus avoiding the possibility of being misled by any slight error which might occur by the application of a single test. It is unwise to place much reliance on the color of the precious stone with regard to its identity, since there is no property of gems so unreliable and variable as this. The color of a precious stone is nevertheless of the greatest importance in regard to its market value, as even a slight variation in depth or tone often makes the difference between an immensely valuable and a common quality gem. The color of the ruby most highly prized is described as "pigeon blood," that of the sapphire "royal blue" or "cornflower" blue. The finest emeralds are spoken of as of "velvety green" richness, and fine white diamonds are termed "blue white," while a fine alexandrite should be pistachio green by daylight and a rich raspberry red by artificial light. Although every gem has a certain tone of color which is most highly prized, specimens occurring exactly of the desired tint are extremely rare; consequently those stones which present an appearance most nearly approaching it are of corresponding value.

Turquoise.—In the foregoing remarks I have been concerned almost entirely with methods of identifying gems which occur in nature in crystalline form, but there are two precious stones which are cryptocrystalline or amorphous, namely, turquoise and opal.

Calliate, or the true turquoise, is a sky-blue opaque colloid stone, consisting of aluminum phosphate and hydrate associated with a hydrated phosphate of copper, and containing small quantities of phosphate of iron and manganese.

Its hardness is 6, and sp. gr. 2·7, which, however, must not be taken with the specific gravity solutions, since the stones may change color. The color of the turquoise is due to the presence of copper phosphate. Powdered turquoise becomes dark-blue when moistened with strong ammonia.

There are two other substances which somewhat resemble the true turquoise; one is odontolite, a fossil bone or ivory colored with phosphate of iron, which is more opaque than turquoise and much softer; also its bony structure can always be seen under the microscope. There is also the mineral known as variscite, sometimes called callais or callainite, which has a hardness of 4, sp. gr. of 2·55, and is bluish green in color.

Opal.—Among the various mineral species known as opal there is only one kind of value as a gem. This is the variety called "precious opal," a beautiful translucent gem, which by transmitted light appears milky and cloudy, but by reflected light exhibits most brilliant prismatic colors reflected in patches of orange, red, blue and green. In composition opal consists essentially of silica, differing from rock crystal in being vitreous and containing combined water. Its sp. gr. is 2·2; hardness, 5·5 to 6. There is really no stone which resembles in any way the precious opal, and any specimen of fair quality can be easily recognized by its fiery play of colors.

The common and valueless variety of opal, which does not exhibit the prismatic colors, occurs in tints of pinkish brown and yellow, and often shows agatoid and dentritic markings. Hydrophane is a variety which readily absorbs moisture, and though not naturally transparent, becomes so (and sometimes prismatic) on being immersed in water. It is of little or no value. Cachalong is the name given to these specimens of hydrophane which adhere to the tongue.

Nomenclature of Gems.—Many of the difficulties frequently experienced in distinguishing precious stones are due to the nomenclature of gems being so very faulty and confusing; for instance, a diamond is always called a diamond, no matter in what color it occurs, be it green, red, blue or yellow, but corundum has a different name for every color in which it is found. When red it is called ruby; the blue variety is called sapphire; the green, oriental emerald; the yellow, oriental topaz; the purple, oriental amethyst; and the asteroid variety is known as asteria. In the beryl family of gems the bright green is called emerald; the blue, aquamarine; while only the sage green is known as beryl. The variety of chrysoberyl which has the property of changing from green by daylight to red by artificial light is called alexandrite, and the cats-eye is another variety of the same stone possessing a brilliant shimmering white line when cut *en cabochon.*

All the many different kinds of zircon, or jargoon, receive the same name with the exception of the red one, which is called jacinth. Indicolite and rubellite are the blue and red varieties respectively of the tourmaline. Olivine is called peridot when it occurs green and chrysolite when it occurs yellow, but the topaz is always called a topaz, no matter if it be blue, white, pink or yellow. Spinels occur in every imaginable shade of color; the bright red ones are called Balas rubies, and the others are known as red, blue, green and purple spinels, as the case may be. Amethyst is a purple form of quartz crystal, which also occurs yellow, being known then as cairngorm or Scotch topaz.

It should be borne in mind also that the same name has become associated with several absolutely distinct gems. No less than three different gem stones are known as topaz: (1) There is the "Oriental" topaz (corundum), (2) the true topaz, and (3) the Scotch topaz (yellow crystal); they can, however, be distinguished by their hardness and specific gravity. The beautiful golden-red stone called essonite is frequently misnamed jacinth, which is the correct designation of jargoons of the same color.

There are two or three varieties of quartz which, when properly cut and polished, display a ray of light somewhat resembling the true catseye. Although not approaching that costly and exquisite gem in luster or beauty of coloring, they have become known by the name of quartz catseyes. The emerald, amethyst and topaz must not be confounded with the Oriental emerald, Oriental amethyst and Oriental topaz, which are varieties of corundum, and are not infrequently known as "fancy sapphires."

It is greatly to be regretted that ignorance and the desire of irresponsible venders of gems to deceive the unwary has caused absolutely fictitious and misleading names to be indiscriminately applied to certain stones, such as "Parisian diamonds," "Cape ruby," "Rhine stone," "Cornish diamonds," "Evening emerald," "Chinese catseye," "Brazilian sapphire," etc.

GOLD AND SILVER.

THERE was a large increase in the world's production of gold in 1898, the details of which are given in the subjoined table. This was due chiefly to the gains in three districts, namely, the Witwatersrand, Western Australia, and the Klondike. Of these the gain in the last was largely fortuitous. The gold from this source is derived entirely from alluvial deposits which are quickly worked out, and there is not consequently the same prospect of a permanent increase in production as there is in the Witwatersrand and Western Australia, where the mines are opened exclusively on lodes. However, the increase in production in 1898 is not wholly accounted for by these districts, and in the other parts of the world where the statistics show an advance, especially in the United States, British Columbia, Mexico, Queensland, Russia and British India, the increase must be attributed to the

PRODUCTION OF GOLD IN THE UNITED STATES.

State or Territory.	1895.		1896.		1897.		1898.	
	Fine Ounces.	Value. (*a*)	Fine Ounces.	Value. (*a*)	Fine Ounces.	Value. (*a*)	Fine Ounces.	Value. (*a*)
Alaska	78,140	$1,615,300	99,444	$2,055,700	130,624	2,700,000	136,430	$2,820,000
Arizona	95,072	1,965,300	124,770	2,579,000	130,624	2,700,000	135,462	2,800,000
California	722,171	14,928,600	737,036	15,235,900	725,689	15,000,000	720,851	14,900,000
Colorado	648,074	13,525,300	719,264	14,867,971	947,249	19,579,637	1,138,584	23,534,531
Idaho	86,088	1,779,600	104,263	2,155,300	96,759	2,000,000	99,178	2,050,000
Michigan	2,075	42,900	1,800	37,200	(*c*)	(*c*)	(*c*)	(*c*)
Montana	198,405	4,101,400	209,207	4,324,700	217,534	4,496,431	253,890	5,247,913
Nevada	75,088	1,552,200	116,620	2,410,538	145,138	3,000,000	145,138	3,000,000
New Mexico	23,810	492,200	23,017	475,800	22,738	470,000	23,222	480,000
Oregon	42,972	888,300	59,313	1,226,000	65,534	1,354,593	58,862	1,216,669
South Dakota	187,187	3,869,500	237,978	4,919,000	256,410	5,300,000	276,730	5,720,000
Southern States (*b*)	15,026	310,600	12,785	264,300	12,082	249,737	12,731	263,153
Utah	66,419	1,373,000	91,908	1,899,900	89,305	1,845,938	114,777	2,372,442
Washington	16,980	351,000	19,626	405,700	21,754	449,664	29,028	600,000
Other States	1,693	35,000	1,413	29,200	3,136	64,795	3,760	77,722
Total domestic	2,265,612	$46,830,200	2,558,433	$52,886,209	2,864,576	59,210,795	3,148,642	65,082,430
Foreign	217,234	4,490,227	409,315	8,461,023	584,983	12,091,599	1,065,552	22,024,960
Grand total	2,482,937	$51,320,427	2,967,737	61,347,232	3,449,559	71,302,394	4,214,194	87,107,390
Total domestic—kg	70,468	....	79,576	....	89,092	....	97,933	....
Total foreign—kg	6,761	....	12,731	....	18,194	....	33,142	....
Grand total—kg	77,229	....	92,307	....	107,286	....	131,075	....

(*a*) 1 oz. gold = $20·67; 1 kg. = $664·60. (*b*) South Carolina, North Carolina, Georgia and Alabama. (*c*) Included with other States.

GOLD PRODUCTION OF THE WORLD.

Countries.	1896.			1897.			1898.		
	Fine Ounces.	Kilo-grams.	Value.	Fine Ounces.	Kilo-grams.	Value.	Fine Ounces.	Kilo-grams.	Value.
North America:									
United States......	2,558,433	79,576·0	$52,886,209	2,864,576	89,092·4	$59,210,786	3,148,642	97,932.9	$65,082,430
Canada............	134,491	4,183·1	2,780,086	291,583	9,068·6	6,027,016	662,796	20,613·9	13,700,000
Newfoundland.....	3,000	93·3	62,010	3,000	93·3	62,010	3,000	e 93·3	62,010
Mexico............	305,216	9,493·2	6,309,181	344,518	10,715·0	7,121,189	398,487	12,393·5	8,236,720
Central America ..	24,127	750·4	498,450	25,399	e 789·9	525,000	25,399	e 789·9	525,000
South America:									
Argentina....	15,235	473·8	314,907	15,235	e 473·8	314,907	15,235	e 473·8	314,907
Bolivia............	3,144	98·0	65,000	16,076	e 500·0	332,300	16,076	e 500·0	332,300
Brazil.............	58,033	1,805·0	1,199,603	70,736	2,200·0	1,462,120	122,481	3,809·3	2,531,687
Chile..............	68,096	2,118·0	1,407,544	68,102	e 2,118·0	1,407,623	68,102	e 2,118·0	1,407,623
Colombia..........	174,165	5,416·8	3,600,000	188,679	e 5,868·2	3,900,000	179,003	e 5,567·3	3,700,000
Ecuador	6,430	199·9	132,900	6,430	e 200·0	132,900	6,405	e 199·2	132,400
Guiana (British)....	107,773	3,351·9	2,227,662	101,504	3,156·9	2,098,098	184,526	5,739·0	3,814,150
Guiana (Dutch).....	23,523	731·6	486,233	32,982	1,025·8	681,748			
Guiana (French)....	82,115	2,553·9	1,697,332	59,860	1,861·7	1,237,310			
Peru	5,639	175·4	116,571	5,788	c 180·0	119,628	9,958	c 309·7	205,827
Uruguay...........	6,880	213·9	144,600	1,863	57·9	38,506	1,863	e 57·9	38,506
Venezuela.........	39,384	1,224·9	814,067	39,384	e 1,224·9	814,067	39,384	e 1,224·9	814,067
Europe:									
Austria........	105,397	3,278.2	2,178,556	2,174	67·6	44,927	2,174	e 67·6	44,927
Hungary..				98,645	3,068·0	2,038,993	98,645	e 3,068·0	2,038,993
France.............	10,513	327·0	217,304	8,874	276·0	183,430	8,874	e 276·0	183,430
Germany..........	a 79,960	2,487·0	1,652,773	89,417	a 2,781·0	1,848,253	91,539	a 2,847·0	1,892,116
Italy	8,832	274·7	182,557	10,160	316·0	210,014	10,160	e 316·0	210,014
Norway............	500	15·5	10,335	498	15·5	10,301	498	e 15·5	10,301
Russia..............	1,499,947	46,653·2	31,005,717	1,042,017	32,408·2	21,538,490	1,196,634	37,217·0	24,734,418
Spain...............	....	....	....	13,279	413·0	274,480	13,279	e 413·0	274,480
Sweden............	3,682	114·5	76,107	3,643	113·3	75,299	3,643	e 113·3	75,299
Turkey.............	387	12·0	8,000	386	12·0	7,975	386	e 12·0	7,975
United Kingdom...	1,353	42·1	27,967	1,354	42·1	27,980	1,354	e 42·1	27,980
Asia:									
China..............	321,296	9,992·8	6,641,190	321,296	e 9,992·8	6,641,190	321,293	e 9,992·8	6,641,190
India (British)......	296,494	9,221·4	6,128,530	353,147	10,983·4	7,299,554	375,704	11,684·9	7,765,807
Japan	34,509	1,073·3	713,300	34,509	1,073.3	713,300	34,509	e 1,073·3	713,300
Korea..............	34,918	1,086·0	721,765	52,927	1,646·1	1,094,000	52,927	e 1,646·1	1,094,000
Malay Peninsula...	25,000	777·6	516,750	25,000	777·5	516,750	25,000	e 777·5	516,750
Borneo.............	3,628	112·9	75,000	4,838	e 150·5	100,000	4,838	e 150·5	100,000
Africa:									
Witwatersrand.....	1,832,329	56,988·0	37,874,240	2,511,544	78,112·6	51,913,607	3,564,487	110,860·6	73,677,936
Other dists., S.A.R...	193,544	6,013·5	4,000,556	232,466	7,230·0	4,805,072	212,522	6,609·7	4,392,825
Rhodesia...........	*Nil.*	*Nil.*	*Nil.*	*Nil.*	*Nil.*	*Nil.*	20,981	652·5	433,682
West Coast	39,585	1,231·0	818,225	48,363	1,504·1	999,653	34,845	1,083·7	720,248
Madagascar........	19,351	601·8	400,000	19,352	e 601·9	400,000	19,352	e 601·9	400,000
Australasia, 7 cols..	2,092,446	65,070.9	43,250,853	2,539,491	78,981·8	52,491,279	3,013,763	93,732·3	62,294,481
Totals..........	10,219,355	317,831·6	211,242,081	11,549,095	359,193·1	238,719,755	13,988,767	435,075·9	289,147,779

(*a*) Production of refiners. (*c*) As reported by the Lima mint. (*e*) Estimated.

causes cited in The Mineral Industry, Vol. VI., for the increase in 1897, *i.e.* the cheapening of production, by improvements in mining methods and metallurgical practice; the diversion of attention from silver mining to gold mining, on account of the decrease of profit in the former; the increased profit in gold mining in countries whose finances are on a silver basis; and the growing tendency of capital to seek investment in gold mining as an industry which aims at the production of a metal that, nominally at least, is not subject to market fluctuations.

The world's production of silver in 1898 decreased. Attention should be called to the fact that the totals for recent years have been largely overestimated in all statistical publications with the exception of The Mineral Industry, Vol. VI., owing to the lack of precise statistics for Bolivia. The silver production of that republic was reported in The Mineral Industry, Vol. VI., as 10,500,000 oz., which was modified but little by the revised returns received later in the year. Other statisticians who have estimated the Bolivian production, with insufficient data, have largely overshot the mark.

SILVER PRODUCTION OF THE WORLD. (*a*)

Countries.	1897.			1898.		
	Troy Ounces.	Kilograms.	Commercial Value.	Troy Ounces.	Kilograms.	Commercial Value.
North America:						
United States	56,457,292	1,756,004·0	$33,755,815	58,763,127	1,827,723·0	$34,670,245
Canada	5,558,446	172,885·6	3,323,395	4,434,085	137,913·3	2,616,110
Mexico	54,052,647	1,681,212·0	32,318,078	56,859,076	1,768,501·0	33,546,855
Central American States	1,623,575	*e* 50,500·0	970,735	1 623,575	*e* 50,500·0	957,909
South America:						
Argentina	328,262	*e* 10,210·0	196,508	328,262	*e* 10,210·0	193,675
Bolivia	10,723,790	333,609·9	6,411,754	11,000,000	*e* 342,188·5	6,490,000
Chile	4,613,974	*d* 143,514·0	2,758,095	4,613,974	*e* 143,514·0	2,722,245
Colombia	1,646,080	*e* 51,200·0	985,191	1,646,080	*e* 51,200·0	971,187
Ecuador	7,716	*e* 240·0	4,610	7,716	*e* 240·0	4,553
Peru	1,876,590	*c* 58,368·0	1,121,013	1,876,590	*e* 58,368·0	1,107,188
Europe:						
Austria	1,286,836	40,026·0	769,399	1,286,836	*e* 40,026·0	759,233
Hungary	861,298	26,790·0	515,071	861,298	*e* 26,790·0	508,166
France	2,583,285	80,351·0	1,544,547	2,583,285	*e* 80,351·0	1,524,134
Germany	14,405,834	448,068·0	8,613,248	15,450,583	*a* 480,578·0	9,115,744
Greece	1,099,376	*f* 34,183·0	657,317	1,303,136	*f* 40,533·0	768,850
Italy	1,456,812	45,313·0	871,028	1,456,813	*e* 45,313·0	859,520
Norway	151,748	4,720·0	90,796	151,748	*e* 4,728·0	89,531
Russia	280,132	8,713·0	167,495	278,515	8,663·0	164,324
Spain	4,211,781	*b* 131,000·0	2,518,224	7,362,350	*b* 229,000·0	4,343,786
Sweden	71,315	2,218·2	42,639	71,315	*e* 2,218·2	42,176
Turkey	49,029	1,525·0	29,314	49,029	*e* 1,525·0	28,927
United Kingdom	249,156	7,750·0	148,970	249,156	*e* 7,750·0	147,002
Asia:						
Japan	2,508,067	78,009·0	1,499,580	2,508,067	*e* 78,009·0	1,479,759
Australasia	17,077,248	531,174·1	10,210,487	17,179,683	534,360·0	10,136,013
Totals	182,080,914	5,663,301·8	109,866,521	190,641,163	5,929,619	$112,478,287

(*a*) Production of refiners. (*b*) Reported by the *Revista Minera*; it is doubtful whether the silver contained in ores and silver-lead is included in the report for 1897. (*c*) As reported by the Lima mint. (*d*) Silver bullion exported. (*e*) Estimated. (*f*) The production of Greece is mostly refined in other European countries and consequently has not been included in the totals.

PRODUCTION OF SILVER IN THE UNITED STATES.

State or Territory.	1895.		1896.		1897.		1898.	
	Troy Ounces	Commercial Value (*a*)	Troy Ounces.	Commercial Value. (*a*)	Troy Ounces.	Commercial Value. (*a*)	Troy Ounces.	Commercial Value. (*a*)
Alaska	74,616	$48,724	150,000	$100,650	250,000	$149,475	250,000	147,500
Arizona	859,739	561,174	2,000,000	1,342,000	1,332,292	796,577	2,750,000	1,622,500
California	463,910	302,933	600,000	402,600	757,300	452,790	750,000	442,500
Colorado	17,891,626	11,687,150	22,500,000	15,097,500	21,278,202	12,722,227	23,502,601	13,866,535
Idaho	8,425,653	2,236,951	5,400,000	3,623,400	6,000,000	3,587,400	6,284,744	3,707,999
Montana	15,046,400	9,825,315	15,720,000	10,548,120	16,807,346	10,049,112	14,818,662	8,743,011
Nevada	807,230	527,120	1,200,000	805,200	1,500,000	896,850	1,400,000	826,000
New Mexico	409,549	267,430	700,000	469,700	350,000	209,265	650,000	383,500
Oregon	11,686	7,631	61,100	40,998	84,802	50,703	128,326	75,712
South Dakota	70,226	45,858	450,000	301,950	500,000	298,950	600,000	354,000
Texas	529,974	346,073	525,400	352,543	600,000	358,740	600,000	354,000
Utah	6,579,043	4,296,115	8,842,810	5,933,526	6,689,754	3,999,804	6,570,256	3,876,451
Washington	98,523	64,336	274,900	184,458	242,781	145,159	350,000	206,500
Others	57,411	37,496	64,600	43,347	64,815	38,753	108,538	64,037
Totals	46,331,235	$30,254,296	58,488,810	$39,245,992	56,457,292	$33,755,815	58,763,127	34,670,245

(*a*) The average value in 1896 was 67·1c per oz., 59·79c. in 1897, and 59c. in 1898.

UNITED STATES.

The statistics of gold and silver production presented in the accompanying tables are compiled as to totals from returns by all the American refiners and exporters of silver-bearing material and reports from the Bureau of Statistics of the Treasury Department as to gold and silver imported in ores and bullion, but in the distribution of the totals the statistics compiled by H. A. Lee, Commissioner of Mines of Colorado, have been used for that State, while those of E. B. Braden, United States Assayer at Helena, Mont., have been taken for the States of Montana, Oregon and Utah. The carefulness and experience of these officials render their work worthy of credence.

Alaska.—The gold output of this territory was larger in 1898 than in 1897, but the new stamp mills on Douglas Island and near Juneau, not having been completed so early in the year as was expected, there was not so much of an increase as was looked for. The new mills were ready for operation in October. They give Douglas Island a total of 880 stamps, with a capacity for crushing 2,500 tons of ore per day. With the large increase in the milling capacity of Douglas Island and the Juneau district there is reason to expect in 1899 twice as large an output of gold from southeastern Alaska as in 1898. The ore reserves of the Douglas Island mines are said not to show any diminution. Placer mining in Alaska in 1898 was rather disappointing, although experienced men did well. Operations were hampered somewhat by excessive water in the creeks.

Arizona.—Gold mining was prosperous in Arizona in 1898, but silver mining proper continued to decline, the only concerns producing silver bullion being the White Hills in Mohave County and the Commonwealth Mining Co. in Cochise County. Otherwise the silver output of the State was obtained in connection with gold, lead and copper. The Fortuna mine in the Gila Range, near Yuma, continued to be a large producer; a new shaft was sunk which cut the vein at a greatly increased depth; cyanide works are projected for the treatment of the stamp mill tailings. The Commonwealth mine at Pearce increased its large production, and the width and richness of the vein disclosed by new exploration work proved astonishing. The discovery of a large lode near Willcox in the Graham Mountains, somewhat resembling the Commonwealth, caused some excitement. A promising discovery was made in the Allison mine in the Babioquivari Mountains, west of Tucson. The King of Arizona and Crowned King mines were operated successfully. Work was resumed in the Head Center and Tranquillity mines at Tombstone, with satisfactory results; 25% of the product is gold. There are 14 cyanide plants in Arizona which, in the aggregate, treat about 400 tons of ore per day.

California.—The gold production of this State decreased in 1898 on account of the excessive drought which prevailed during the year, rendering placer mining impossible in many districts and compelling many of the quartz mill men to hang up their stamps; in Nevada County about 200 stamps were idle at one time; in December there was an unusually heavy rainfall in the valleys and snowfall in the mountains, wherefore an abundant supply of water is assured for 1899. Outside of the old mining districts of the State, which maintained their usual output except when interrupted by lack of water, considerable attention was paid to various mining districts in Del Norte, Siskiyou and Shasta counties, where exploration work has been done on quartz veins, which as a rule are small in comparison with those in the line of the "Mother Lode." In the southern section of the State the mines of Randsburg began to turn out well, mining in this district having settled down to a solid basis. The Vanderbilt mines were important producers. River dredging was continued in 1898 and is expected to become a more prominent feature of mining operations in the State during 1899, a number of new enterprises having been outlined for operation on the upper reaches of the Sacramento and other northern rivers.

Colorado.—Gold mining in this State in 1898 experienced an unparalleled degree of prosperity, while the production of silver, which is obtained largely as a by-product, or at least as an accessory product, also showed an increase. This is attributed by H. A. Lee, State Commissioner of Mines, especially to the readjustment of economic questions to meet existing conditions. Transportation facilities were improved by the construction of a number of wire tramways and railway switch extensions, while the power of the mountain streams is being utilized by electrical transmission. A noteworthy feature of 1898 was the reopening of a large number of old mines, which was made possible by improvement in the mining and metallurgical methods. In the following county reviews the statistics are taken from Mr. Lee's report for 1898.

Clear Creek County.—The production of gold in this county fell off from $782,649 to $605,528; the production of silver increased from 1,442,583 oz. to 1,569,012. The silver came chiefly from the mines of Silver Plume and Georgetown and Idaho Springs; the gold from the lower section of the county. There were few developments of importance during the year. Empire camp came somewhat to the front and the opening of the mines near Idaho Springs by deep cross-cut adits received considerable attention.

El Paso County (Cripple Creek).—The gold production of this county increased from $10,131,855 in 1897 to $13,507,349 in 1898; consequently the Cripple Creek mines furnished considerably more than one half the total gold output of the State. The tonnage of ore produced was 381,550, of which 107,750 was smelted and the remainder treated by chlorination or cyanide lixiviation, with the exception of a few tons crushed by the local stamp mills. Of the cyanide and chlorination works the following were in operation: Metallic Extraction Co., El Paso Reduction Co., Colorado-Philadelphia Reduction Co., Gillett Reduction Co., Brodie Gold Reduction Co., Colorado Ore Reduction Co., and Kilton Ore Reduction Co. The Economic Gold Extraction Co. commenced a large chlorination works near Victor. The Cripple Creek District electric railway was completed, but during 1898 was used only for passenger traffic. When ready for freight traffic it will probably improve conditions a good deal in the district, since the comparatively high cost of wagon transportation has been a drawback hitherto. Another new improvement in 1898 was the introduction of power transmitted electrically by the Colorado Electric Power Co. from Canyon City, about 25 miles from Cripple Creek; numerous electric hoisting engines, etc., are now in use in the district. The principal producing mines were the Portland, Gold Coin, Independence, Strong, Legal Tender, Vindicator, Lillie, Victor, Isabella, Union, Zenobia, Free Coinage, Modoc, Last Dollar, Half Moon, Anchoria-Leland, Moon-Anchor, Gold King, Anaconda, Raven and Elkton. There were no new discoveries of great importance during the year and but little progress in extension of the producing district. Litigation began between the Portland and Strong mines. The Por land paid $570,000 in dividends; Victor, $350,000; Elkton, $220,000; Vindicator, $177,625; Moon-Anchor, $165,000; Lillie, $153,950; Gold Coin 120,000; Anchoria-Leland, $72,000; Golden Cycle, $60,000; Matoa, $25,000 Montreal, $7,500; total, $1,921,075. This does not include the profits of the Independence, Strong, Raven, Modoc and other private companies.

Gilpin County (Black Hawk).—The gold production of this county fell off from $2,086,471 in 1897 to $1,983,514 in 1898; the silver production also decreased slightly, but the total is unimportant. However, mining was generally prosperous, although there were no new discoveries during the year. The grade of the mill ore averaged about the same as in 1897. The Topeka, Lillian, East Notaway and some other mines are reputed to have paid dividends, but their aggregate was not large.

Lake County (Leadville).—The production of gold in this county in 1898 was $2,073,036, against 2,063,858 in the previous year; the production of silver increased from 5,451,317 oz. to 7,068,727. The tonnage of ore raised is estimated at 425,000, a large part of which was silver-bearing manganese-iron ore. The increase in the output of silver was due chiefly to the mines on the sulphide ore shoots between Carbonate and Iron Hill and the reopening of some of the former large producers, including the Louisville, the A. Y. & Minnie, and the mines of the Iron Silver Mining Co. The mines under the city of Leadville, which had been idle since the summer of 1896, were unproductive in 1898, but the Leadville Pumping Association resumed operations in October and it is expected that this district will make an output in 1899. The gold production of Leadville as in the previous year was made chiefly by the Ibex Mining Co., which is said to have produced upward of 100,000 tons of ore, of which 60% was sulphide. The extension of railway branches to this mine and some others in the Leadville district increases the shipping facilities. A new discovery of gold ore was made in Lake Park, a few miles west of Leadville.

Mineral County (Creede).—The gold production of this county is insignificant. The production of silver increased from 3,070,576 oz. to 4,177,184. The Commodore mine was the principal producer, and paid large dividends. This mine is opened to a great depth by means of three adits, one of them a cross-cut. The Last Chance and Amethyst mines were worked only in their upper levels, the lower being flooded with water.

Pitkin County (Aspen).—The gold production of this county fell off from $164,430 to $71,001 and silver from 4,599,946 oz. to 3,977,270. Aspen produces partly dry silver ore and partly silver-lead ore, though there is none in which the percentage of lead is very high. A good deal of the ore is calcareous. The silver contents range from 20 to 40 oz. per ton; lead from 5 to 10%; while some of the ores carry from 30 to 45% lime. For the tenor in lime the smelters pay 10c. per unit, which often offsets completely the charge for smelting. On account of the higher prices paid for these calcareous ores and the reduction of freight rates and smelting charges it is now possible to ship a much lower grade of ore from Aspen than in former years, when the price of silver was higher, but the decline in the output shows that mining must still be unprofitable in a good many cases or the mines are being exhausted.

San Miguel County.—The gold production of this county, made chiefly in the vicinity of Telluride, increased from $1,458,145 to $1,572,677; the production of silver increased from 869,079 oz. to 2,129,082. The Smuggler-Union Consolidated Mining Co. was the chief producer of silver. The mines of this company are opened to a great depth on a vein of remarkable persistency. The Tom Boy gold mine, which was closed down in 1897, was reopened in May, 1898, and during the remainder of the year was a large producer.

Idaho.—There were few developments of importance in the gold mining industry of this State in 1898. The production of silver, which is made chiefly by the silver-lead mines of the Cœur d'Alêne, increased about in correspondence with the increase in production of lead. Developments in this district are referred to especially under the caption "Lead," elsewhere in this volume.

Montana.—There was considerable increase in the gold production of this State in 1898, without a single development of special importance. Mention should be made, however, of the new discovery of ore on the 700-ft. level of the Drumlummon mine and the successful treatment of the stamp mill tailings at that mine by the cyanide process. The decrease in the silver production of Montana was due especially to the falling off on the part of the Anaconda Mining Co., the largest producer, for an account of which reference may be made to the paper on "Copper," elsewhere in this volume. Some of the mines worked for silver alone, which have been idle for several years, were reopened in 1898. The Granite Mountain and Bi-metallic pooled interests and started up under one management. At Butte the Alice mine was worked by the tribute system, while the Moulton was exploited under lease. The production of gold and silver in Montana, classified as to origin, reported by E. B. Braden, assayer in charge of the United States assay office at Helena, is given in the following table:

Classed as—	1897.—Ounces.		1898.—Ounces.	
	Gold.	Silver.	Gold.	Silver.
Placer bullion	33,418·923	3,098·48	31,662·452	4,021·33
Mill bullion	66,135·432	663,643·85	80,590·965	1,028,619·10
Cyanide process	9,993·117	33,937·50	19,223·130	78,006·95
From copper ores	51,463·070	10,710,317·02	51,408·020	9,495,911·71
From lead ores	8,529·345	4,175,792·56	8,369·027	2,763,991·44
From smelting ores	47,974·959	1,248,556·65	62,602·143	1,448,111·45
Total	217,514·846	16,887,346·06	253,867·787	14,818,661·98

Nevada.—The De La Mar mine was the largest producer in this State, as in the previous year. Arrangements were made to rework the accumulations of tailings on the Comstock lode by the cyanide process, and in 1899 an increase in the production of gold and silver may be expected from this source. The De La Mar mine is situated at De La Mar, Lincoln County; it was discovered in 1893. It is opened on a strong vein in quartzite, which is worked through adit levels. The ore is a quartzite breccia hardly distinguishable from the country rock. It is treated by the cyanide process by which 94 to 96% of the gold and 80 to 84% of the silver are recovered. The ore averages ab $28·50 per ton in gold and 2·5 oz. silver. Mining costs $2 per ton, milli $4·25, and general expenses $1.

North Carolina.—Several of the old gold mines in this State were reopen in 1898, both in the mountains and in the midland counties.

Oregon.—The gold production of this State decreased for the same reas as in California. The chief center of the mining industry is Baker City, the vicinity of which are the Union-Companion, Eureka & Excelsior and C umbia mines, all large producers. A large mill was erected at the Golcon mine in the Cracker Creek district, where there are said to be large depos of low-grade ore.

South Dakota.—The increase in the gold production of this State was d to the greater activity in general through the Black Hills and especially to extension of operations by the Homestake Co. Several old mines were reopen and the exploitation of some promising prospects, which have been await capital, was undertaken. The extension of the rich mining region to Spear Canyon was a material advance. During the year rich silicious ore found near Custer in the southern Black Hills. A phenomenon in this reg was the Holy Terror mine at Keystone, the 10-stamp mill of which yiel $70,000 in one week. A similarly rich vein was struck near Hills City in the year. The Deadwood & Delaware smelting works at Deadwood w rebuilt and operated on a large scale, a large part of the ore treated be pyrites concentrates from the Homestake and other mills. The D. & D. wo comprise four cupola and three reverberatory furnaces. The D. & D. Co. mining interests of its own in different parts of the Black Hills, especiall the Ruby Basin, where the Union mine owned by it is a large producer. Golden Reward Co. kept its chlorination works at Deadwood in operation chi on ore from the Tornado mine at Terry. The chlorination works at Pluma chiefly on ore from the Mogul, Golden Sands and other mines at Bald Mount The old chlorination works at Rapid City were started up again.

Utah.—In this State the Camp Floyd or Mercur district continued to be largest producer; its output showed a considerable increase, some of the cyanide works which were in course of construction in 1897 having been pu operation in 1898. The output of the district was 50,363 oz. gold against 739 in 1897. The Mercur mine produced 24,200 oz. in 1898. La Cigale Overland mills were not started up in time to contribute to the 1898 yi and although the De La Mar mill began operations early in the summer up to end of the year it had not settled down to regular work, numerous difficult especially in connection with the roasting furnaces, having appeared. The Chloride Point and Daisy mills began operations. There is a prospect of ther additions to the mill capacity of the Mercur district. The pan ama mation mills in the Tintic district (Eureka Hill, Mammoth and Sioux) produ about 16,400 oz. gold and 1,468,500 oz. silver, besides which a good dea ore was shipped to Salt Lake City smelting works. At Park City the Ma mill (Russell process) was run on tailings from the Ontario mill dump, treat 80 tons per day at a fair margin of profit; 72,471 oz. silver were produced.

Daly and Ontario mines remained idle and are likely to continue so until silver reaches a point considerably above 60c. per oz. The Silver King mine of Park City was a large producer of silver-lead ore, and paid large dividends.

Wyoming.—Numerous gold discoveries were reported in this State, but none produced enough to merit special attention. The Carissa mine was equipped with new machinery, including a stamp mill which was started in September. The exploitation of some dry placers by means of steam shovels was undertaken.

GOLD AND SILVER MINING IN FOREIGN COUNTRIES.

Abyssinia.—According to Arthur Griffiths (in *The Fortnightly Review*, 1898), the existence of a great gold-bearing field in the S. E. extremity of the Soudan, in the country of the Beni Shengol, has long been known. This is believed by many to be the real site of King Solomon's mines. The territory belongs to Abyssinia.

Australasia.—The seven colonies of Australasia made a large increase in gold production in 1898, for although there was a falling off in some this was a good deal more than offset by the gains in Queensland and Western Australia.

Colonies.	1897.			1898.		
	Crude Oz.	Fine Oz.	Value.	Crude Oz.	Fine Oz.	Value.
New South Wales.........	292,217	256,342	$5,296,197	341,722	292,940	$6,055,070
New Zealand...............	231,644	230,782	4,770,256	280,176	254.416	5,258,779
Queensland..................	807,928	601,394	12,423,584	918,106	647,487	13,383,556
South Australia............	33,900	28,260	584,134	....	22,377	462,583
Tasmania......................	60,646	53.538	1,106,640	....	68,624	1,418,258
Victoria........................	812,766	765,345	15,819.677	837,258	788,429	16.296,390
Western Australia........	674,993	603,830	12,481,173	1,050,180	939,490	19,418,712
Totals	2,934,094	2,539,491	$52,481,661	....	3,013,763	$62,203,298

New South Wales.—This colony, in which is situated the Broken Hill district, continues to be the most important silver producer of Australia, the output of the other colonies, indeed of the other districts outside of Broken Hill, being comparatively insignificant. A new discovery of silver-lead ore was made in 1898 at Burragorang, about 30 miles from Pictou and 40 miles from Camden on the Southern Railway line. According to late reports this find is turning out well, considerable shipments having been made to the Illawarra smelting works, although the long wagon haul is a serious handicap. The production of gold in the Colony in 1898 was checked by the severe and protracted drought. The increase in the output occurred in spite of this, coming from the older and well developed districts. There were no new finds of importance in 1898. The Broken Hill Proprietary Co. produced 2,691,546 oz. of silver during the half year ending November 30, 1898, against 2,873,684 oz. in the previous half year. This was a large falling off from 1897, the output in the year ending November 30, 1897, having been 6,584,557 oz.

New Zealand.—The gold production of this colony showed a small increase in 1898, due chiefly to the Waihi Gold Mining Co., which produced £242,871 against £144,000 in the previous year. The Hauraki, Ohinemuri and Thames districts showed increases in production; Coromandel a decrease. The Opitinui district is expected to be productive in 1899.

The Coromandel and Thames districts of the Hauraki peninsula are the most important gold producers of this colony. Their characteristic features are narrow veins of quartz, frequently in disturbed country rock, which carry gold irregularly in rich patches here and there. When a number of these rich patches have been found within a small area production and profits have been temporarily large. Apparently, however, these rich patches become much more infrequent below 300 ft., since most of the mines which have been worked below that depth have failed to continue profitable. In the Coromandel district there are 30 mines, most of which are well equipped. The Hauraki was worked successfully down to 200 ft. but has hardly paid its way below that depth. The Kapanga is opened to 1,000 ft. but has had no pay ore below 400 ft.*

The Upper Thames or Ohinemuri district has veins frequently from 10 to 20 ft. wide, wonderfully free from disturbance, regular in occurrence and containing the gold, whether in payable or unpayable quantity, fairly regularly distributed through the ore. This district is by far the most important mining center in New Zealand. It is in this district that is situated the famous Waihi mine, which has several lodes aggregating 30 ft. in thickness and opened for a length of 2,000 ft. along the strike, showing an immense reserve of payable ore. At present the company has 190 stamps at work, which crush about 7,000 tons of ore per month. The ore is stamped dry through a 40-mesh screen. The introduction of wet crushing is now being considered.†

Queensland.—The gold production of this colony increased slightly more than 10% in 1898. Its resources in gold are immense, as has been frequently pointed out in preceding volumes of this series, and a further increase in production from the colony may be looked for. The principal centers of lode mining, going from south to north, are: (1) Gympie, (2) Mt. Morgan, (3) Ravenswood, (4) Charters Towers, (5) Etheredge, (6) Croydon. From Gympie in the south to Croydon on the Gulf of Carpentaria is a distance of 800 miles, the whole country being covered with gum trees. There are extensive areas which have never been prospected and a good many known deposits of low grade ores which can be worked with improved facilities. In the Charters Towers and Ravenswood districts there are said to be dikes hundreds of feet wide which assay $4 to $5. Water is generally procurable and there is no doubt that some day many of these will become productive.

The Gympie gold field lies about 100 miles north of Brisbane. Gold was discovered there in 1867, since which time the mines have been worked regularly. The occurrence of gold in this district is very peculiar. Several parallel beds of slate dip across the country at an angle of about 20°. In the opposite direction, dipping at a steep angle, is a series of white quartz reefs; these are usually poor in gold contents, except where they cross through the slate beds, and when this happens they are rich in gold. The widest of the

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* *Economist*, Jan. 14, 1899.

† *Ibid.*, Jan. 21, 1899.

slate bands is about 200 ft. thick, so that every reef passing through this usually carries 200 ft. of valuable "backs." When one reef is exhausted in the neighborhood of the slates, the management at once leaves it as worked out, and proceeds to crosscut in the hope of finding another reef passing through the slate further on. It is only recently that any deep explorations have been made. They have been entirely successful. At depths of from 800 to 1,400 ft. a further series of reefs are found passing through the slate, and these where opened have so far proved as good as any of the previously worked reefs which passed through the slates at a shallower depth. There are fully 20 mines, most of them deep levels, which to all appearance have most favorable prospects. On the strength of this the Gympie field is attracting a good deal of attention in Queensland and New South Wales.*

The Charters Towers gold field is situated about 800 miles north of Brisbane and 80 miles from the port of Townsville. The system of lodes, of which there is a great number, lies within an area of a few square miles. They are fissure veins, filled with white quartz intersecting granite, but wherever they are rich in gold they are also highly mineralized with pyrites and galena. The lodes are irregular in size and gold contents. Between 60 and 80 mines are now being exploited. The principal reefs are the Brilliant, Day Dawn, Victoria, Queen and Victory. The first two are the largest and richest. Neither appears to have much lateral extent, and since they are being worked not only by the outcrop mines but by several rows of deep levels they are becoming exhausted rapidly.

The Brilliant reef dips about 45° and averages about 4 ft. in thickness. Apparently it has never been found near the surface. The shallowest depth at which it has been worked was about 300 ft. (in the Victory mine) and the deepest about 2,500 ft. vertically (in the Brilliant and the Brilliant Deep Levels), this being the deepest working in Queensland. There is a well-defined channel of rich ore about 800 ft. wide going down with great regularity, while outside of this the reef, so far as explored, is valueless.

The mines of the Charters Towers district are controlled entirely by local capitalists who have perpetuated the old-fashioned and costly methods of working in vogue when mining was first begun on the field. The underground work is on the whole efficient, but the surface handling of the ore is extravagant. Few of the mines possess batteries of their own. The ore is carted to the mills in wagons. The cost of crushing and amalgamating is 14s. per ton; collecting concentrates comes to several shillings a ton extra. The tailings belong to the mills unless they are specially carted away. The mill owners realize a profit of 6s. to 12s. a ton. The directors of mining companies are frequently owners or part owners of mills. Within the last two years about 70 cyanide plants have been erected to treat the old tailings which previously had been allowed to run to waste. Although cyanide gold will swell the output of the district for some years to come the opinion is expressed that the maximum for the district has been reached at 40,000 oz. per month and from now onward it will tend to decrease slowly.†

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* *Economist*, Oct. 29, 1898.

† *Ibid.*, Nov. 12 and 19, 1898.

The Charters Towers mines produced 729,000 tons of ore in 1898, yielding 458,000 oz. gold, of which the stamp mills crushed 210,000 tons for a yield of 233,000 oz. and the cyanide works, 513,000 tons for 214,000 oz. while the remainder was recovered by smelting and chlorinating or from alluvial working. The value of the output was £1,160,000.

The Ravenswood gold field, which is about 50 miles from Charters Towers, has been for many years a disappointing one owing to the refractory character of the ores and the impracticability of erecting the smelteries necessary to treat the ore profitably. The reefs although narrow and often badly faulted are in some places particularly rich, and several mines which are shipping their ores to outside smelters are doing very well. Doubtless these reefs will be worked more systematically and profitably some time in the future.*

The ore at Mt. Morgan holds out with depth, it having been proved that at the lowest point yet reached, namely, 650 ft. from the original summit of the mountain, the deposit has become hardly any smaller than at the top, but the average gold contents are becoming less. The entire top of the hill, equal in area to about 250 yards long, 200 wide and 100 deep, has been or will be dug away almost entirely. Below this the sides of the hill slope away from an outer shell of barren rock, down the center of which the great mass of gold ore continues. The reports of the Mt. Morgan Co. for the last four semesters are summarized as follows:

Half-Year Ended—	May 31, 1897.	Nov. 30, 1897.	May 31, 1898.	Nov. 30, 1898.
Ore treated (tons)	48,946	73,610	79,687	107,860
Yield (oz.)	82,925	85,627	84,699	84,087
Average, per ton	1·7	1·1	1·0	0·7
Value of gold, etc	£341,768	£352,487	£346,008	£334,194
Expenditure, ordinary	133,631	148,502	120,709	167,240
Expenditure, special	25,591	22,664	38,656	30,260
Profit	192,546	181,321	187,643	136,694
Dividends paid	150,000	150,000	150,000	175,000

Tasmania.—Almost the whole of the gold output of Tasmania is obtained from the Tasmania and New Golden Gate gold mines and from Mt. Lyell. The Tasmania and New Golden Gate mines have paid dividends of £680,000 and £200,000, respectively. It is said that they both have large ore reserves and if worked by modern methods would yield much more than at present. From Mt. Lyell about 2,000 oz. of gold per month is shipped in the form of blister copper and its value does not appear in the statistics of the gold output of the colony.†

Victoria.—There was a small increase in the gold production of this colony in 1898. The mines of Bendigo produced 214,205 oz. gold in 1898 and paid £225,774 in dividends. The record of the remaining districts of the colony was as follows: Gippsland, 106,793 oz. (£108,840 dividends); Ballarat, 186,696 (£99,918); Castlemaine, 71,778 (£75,227); Beechworth, 111,703 (£50,630); Maryborough, 54,339 (£34,262); Ararat, 38,906 (£20,983). Bendigo, Castlemaine, Ballarat and Ararat made increased outputs over the

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* *Economist*, Nov. 26, 1898.

† *Ibid.*, Dec. 10, 1898.

previous year. The number of men engaged in gold mining in Victoria, December 31, 1898, was 30,804, or 2,016 less than 12 months previous.

At Bendigo, which has been the most important field in the colony, there are 12 mines which have been worked below 2,700 ft. deep. The New Chum Consolidated is opened to a depth of 2,553 ft., but the vein has not been payable below 2,000 ft. The peculiar nature of the reefs at Bendigo, however, bids the management never give up hope. Recently the Garden Gully United mine, thought to be almost worked out, struck a valuable patch of reef, and during 1898 became a large producer. As regards the management of Victorian mines, the most distinctive feature is the economy with which they are worked—small shafts, little machinery, and hired batteries. In the case of some of the small mines this method is to the benefit of the shareholders, but in the case of many good mines with proved long lives, this system is a distinct mistake. There is no doubt that the good Victorian mines could be made better by the introduction of modern labor-saving machinery, by a careful system of analysis of accounts and costs, and by the erection of heavy stamp mills. Generally speaking, the management must be considered bad, especially in the reduction of the quartz, but it is so more from pure conservatism than from dishonesty or incompetence.*

Western Australia.—The remarkable increase in the gold production of this colony was the chief feature in Australian mining in 1898. Nearly 100 companies reported yields of gold. A large number of new stamp mills were put in operation and a good many more are in course of construction. There were 22 dividend payers on the list against 21 in 1897. The Great Boulder Proprietary paid £250,000, Lake View Consols £187,500, Associated Gold Mines £90,000, Golden Horseshoe £75,000, Ivanhoe £50,000, and Hannan's Brownhill £41,250. The total dividend payments of the year amounted to £802,000 against £489,000 in 1897, these statistics not including dividends paid by exploration and finance companies. An important development of the year was the passage of the Gold Mining Amendment Act, based on the recommendations of the recent Royal Mining Commission. This act provides for the total abolition of dual titles, retrospective and future, and will prevent the possibility of future collision between lease holders and alluvial miners.

The gold deposits of Western Australia may be divided into two classes: (1) Impregnations of the country rock along fissure planes or crushed zones, with but little deposition of gangue minerals, and (2) normal quartz fissure veins. The deposits at Kalgoorlie and in a few isolated instances in outside fields such as the Sons of Gwalia, Ltd., at Mt. Leonora, fall under the first class, to which 10 of the 16 important mines of the colony belong.†

The first mentioned deposits may be briefly described as zones of crushing and fissuring in diabase and highly altered slates, from which channels impregnation and some replacement of the country rock have taken place. The dynamic phenomena are strongly evidenced by the frequent walls (which,

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* *Economist*, Dec. 3, 1898.

† Herbert C. Hoover, "The Superficial Alteration of Western Australian Ore Deposits," American Institute of Mining Engineers, Buffalo Meeting, October, 1898.

however, do not represent lateral extensions of the ore), and by the not uncommon brecciation and crushed zones. The lodes vary up to 100 ft. in width; and while many are continuously profitable for long distances, the common characteristic of ore occurrence is in lenticular masses or lenses along a general line. The character of the ore is indicated by the following determinations of free silica from average samples of oxidized ore: Hannan's Brownhill mine, 18·21%; Lake View Consols, 28·31; Boulder Main Reef, 31·06; Sons of Gwalia, 40·01. There are no other gangue minerals of consequence, the remainder being almost all kaolin, showing that the oxidized ore is but slightly different in major composition from the country rock. In the unoxidized ores more quartz is present; and it becomes evident that some replacement of the country rock by quartz and the metalliferous minerals has taken place; but, as a whole, the deposits are not structurally complex, although they are so chemically. The values often diminish gradually from a central core or source of impregnation into the surrounding rocks, the limit of the "ore" being regulated by the working costs. Below the zone of oxidation there is a wide range of accessory minerals: sulphides of iron, lead, zinc, mercury, arsenic and antimony; tellurides of gold, silver, mercury and bismuth, with many rare minerals, among them natural amalgam. The gold occurs in minor proportion as free gold, but in major value in the tellurides and sulphides, which are, as a rule, very finely disseminated through the ore, although the tellurides occur occasionally in large patches or stringers.

The mines outside of Kalgoorlie, with a few exceptions, are of the normal fissure vein order. The subsidiary minerals are fewer and more common than at Kalgoorlie, consisting of sulphides of iron, lead, zinc, arsenic and copper. Tellurium has been rarely found in quartz veins. The gold in these veins is altogether different from that of Kalgoorlie, being usually coarse, and occurring often in wonderfully rich patches or pockets. The superficial decomposition offers one general difference from that of other regions, which is in the widespread superficial enrichment. There are many quartz mines of value, the exploitation of which has proved the values to continue in depth; but the bare fact that, of 161 mines "promoted" on quartz veins in these districts, only seven or eight have proved successful, is evidence on this point.

The main gold field of Western Australia covers an area of about 800 miles north and south by 500 miles east and west. Of 580 mining companies promoted in this field not more than 18 may be termed large mines. These 18 mines produced up to June, 1898, a total of 642,284 oz. of gold from 295,096 tons of ore. The average value of the ore has therefore been 2·15 oz. per ton, to which may be safely added an average of 1 oz. lost in the tailings, for few mines had during the period instituted tailings treatment. There are on the fields 1,787 stamps distributed in 186 mills, or less than an average of 10 stamps to a plant. The following plants are in course of erection: Associated, 600 tons daily; Lake View Consols, 400; Hannan's Brownhill, 250; Ivanhoe, 300; and Sons of Gwalia, 300. Several others are either in contemplation or awaiting a lead in metallurgical practice. The highest daily tonnage at present is Lake View Consols, 200. The present daily average of the 18 mines referred to is 80 tons. It seems definitely settled that Kalgoorlie treatment will involve three main principles, viz., dry crushing, roasting and cyanidation. All plants in construction are mechanical variations upon this basis, yet there are many difficulties to be overcome, the separation and treatment of sufficient slimes to give a leachable product and the securing of the gold too coarse for cyanidation being two leading ones.*

There is no surface water on the gold fields, and it is secured entirely from subterranean sources. Most of this water contains a variable percentage of salts up to the saturation point, but in that portion of the field east of Cue and north of Menzies fresh water is met with occasionally, not, however, in large enough quantities to permit of any relaxation of efforts toward economy. There is no deep-seated water, and the zone of surface decomposition of the rocks acts as a reservoir for such rain as falls—about 4 in. annually; below the line of oxidation water rapidly decreases until the workings become almost dry. In localities where mining is much congested, as at Kalgoorlie, and this reservoir is heavily drained, a diminution of water may be expected. Salt water must be distilled for domestic and steam purposes; the cost of such distillation varies from 2c. to 6c. per gal. Salt water costs from 75c. to $7·50 per 1,000 gal., varying with the locality. About 400 gal. per ton of ore treated is considered the average mill, steam and domestic loss with careful conservation, or a cost of from 25c. to $3 per ton of ore. When tailings and slimes treatment are added, a greater percentage of water is saved; in this case only about 250 gal. per ton are lost. The salt water is detrimental to amalgamation and cyanidation, and, moreover, the devices to which it is necessary to resort for its conservation represent a large additional cost for equipment and a constant cost for labor, the amount depending on what proportion of water it is advisable to save. The maximum cost necessitated by scarcity of water is experienced in Menzies, where at one mine it amounts to $3 per ton of ore, and this does not include the loss of gold in treatment due to the water containing from 15 to 32% of salt. A solution of the problem is being attempted by the government in the construction of a pipe line 400 miles long from the Darling ranges, at a cost of $15,000,000, from which it is proposed to supply Kalgoorlie and Coolgardie with 5,000,000 gal. daily of fresh water at a cost of about 60c. per 1,000 gal. An auxiliary supply may be obtained by the construction of reservoirs on the fields to catch the torrential rains, but this cannot be depended upon, nor does the country offer topographical features for any considerable storage.

High wages and inefficient labor are great drawbacks to mining in Western Australia. A careful comparative record shows that the miners accomplish about two-thirds the amount of work of a Californian miner, and only about 40% more than the Kaffirs of the Rand. Single-hand work is used in but few mines, those managed by Americans being almost the only ones. The men work 47 hours weekly and receive $15 to $22·50 and water free. In some mines Italians have been tried and found to accomplish about one-half more

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* This account of mining and milling conditions in Western Australia is an abstract of a paper by H. C. Hoover, in the *Engineering and Mining Journal*, Dec. 17, 1898.

work, but they possess certain disqualifications. The high freight and absolute unproductiveness of the country make living high, and it is not therefore so much in reduction of wages that reforms must come, as in increased efficiency of the labor.

Suitable timber for mine work and surface construction is very scarce. The native timbers are brittle, double the weight of pine, shrink greatly, and are very difficult to work. Sawn woods, either native or foreign, along the railway, cost from $3 to $3·75 per 100 ft., board measure. In the mulga bush, upon which many mines away from the railways have to depend, sticks 8 ft. long and 6 in. in diameter are very difficult to find, and cost about 6c. per running foot. In the southern portion of the fields eucalyptus may be secured, but costs about 12c. per running foot for logs 10 in. in diameter.

The railways are owned by the government, and from an American standpoint freights are very high, but the roads, like the mines, are run under severe conditions. Freight on machinery to Kalgoorlie is 8·6c per ton mile, the distance being 390 miles from the coast. The mines outside of the railroad points are greatly scattered, and transportation is conducted by wagon and on camels. By the former it averages about 24c. per ton mile, and for such goods as the latter can carry, about half this amount. Since some important mines are as much as 200 miles from the railroad, these freights amount to an enormous sum annually, directly and indirectly, adding from 10 to 20 per cent. to the working costs. A 60-stamp mill and corresponding mine equipment would cost for transportation 100 miles beyond a railroad terminus about $75,000.

Fuel is now secured from the abundant bush over the fields, and consists of varieties of eucalypti and mulga. The latter is the better quality and has a steam efficiency about equal to oak. This wood costs from $5 to $7·50 per cord, varying with the locality. It has been largely cut off around the mining centers, and it is only through railway extension that the supply can be maintained. It is an open question whether coal is not more economical at Kalgoorlie at present.

There is a tariff duty on some mining requisites, but not a high one; the cost of transportation, however, brings up the price of supplies to a high figure. The cost of ordinary supplies at Kalgoorlie in June, 1898, was as follows: Cement, bbl., $5·64; galvanized iron, sq. ft., $0·09; candles, 25-lb. box, $2·74; dynamite, case, $16·80; blasting gelatin case, $20·40; detonators, box, $0·96; fuse, coil, $0·17; shovels, each, $0·68; hammers, lb., $0·04; drill steel, lb., $0·05; rails, ton, $57·60; quicksilver, flask, $41·40.

The ground below the line of oxidation (from 100 to 200 ft.) is usually very hard, and the cost of driving in it varies from $10 to $30 per ft. by hand work. Shaft sinking costs $30 to $60 per ft. including timber. Owing to the width of the lodes (6 to 30 ft.) the tonnage developed per foot of opening is very great. The high cost of timber, however, is a great obstacle to the cheap extraction of the ore. The average cost of mining is about $2·70 per ton. Few of the leading mines are proved below 400 ft. and none below 500. The ore reserves already opened in many mines are large and would insure a life of several years at least, even if no more ore were to be opened. At present free milling ore averaging 0·5 oz. per ton, if in large quantity, can be worked profitably.

Bolivia.—Revised statistics of the silver production in this country for the years 1894-97, both inclusive, are presented in the subjoined table, which is based on special reports to us from the producers and our own correspondents in the republic. It will be observed that the totals differ widely from any previously published statistics of silver production in Bolivia in recent years.

Companies.	1894.	1895.	1896.	1897.	1898.
	Oz.	Oz.	Oz.	Oz.	Oz.
Cia. de Huanchaca	7,972,140	4,440,354	3,035,076	4,727,685	4,631,334
Cia. Unificada Colquechaca	1,333,318	(*b*)	679,975	961,450	(*b*)
Cia. Gallofa, Colquechaca	(*a*)	(*b*)	205,724	120,452	(*b*)
Cia. Minera de Oruro	1,518,658	(*b*)	1,633,870	1,763,480	(*b*)
Cia. San Jose, Oruro	(*a*)	(*b*)	1,070,678	1,477,042	(*b*)
Cia. Guadaloupe	652,010	(*b*)	316,092	349,458	(*b*)
Royal silver mines, Potosi	365,549	(*b*)	456,850	601,715	(*b*)
Various companies	(*a*)	(*b*)	97,212	722,508	(*b*)
Five districts (licitados)	2,181,034	(*b*)	832,090	(*a*)	(*b*)
Totals	14,022,709	*c*10,500,000	8,327,567	10,723,790	*c*11,000,000

(*a*) None reported. (*b*) Statistics not available. (*c*) Estimated.

PRODUCTION OF THE COMPANIA HUANCHACA DE BOLIVIA.

Years.	Production of Bar Silver. Kilograms.	Ores Exported. Metric Tons.	Silver Contained In Ores Exported. Kilograms.	Total Silver. Production. Kilograms.
1884	101,982·790	101·200	4,320·757	106,303·547
1885	89,583·619	744·289	23,608·289	113,189·908
1886	82,555·588	4,097·001	83,241·703	165,797·241
1887	63,973·982	4,350·772	67,330·108	131,304·090
1888	64,421·979	4,510·633	65,740·925	130,162·904
1889	54,825·976	7,245·000	81,483·664	136,309·640
1890	49,810·697	14,382·291	119,570·284	169,380·981
1891	58,461·888	15,818·020	124,115·456	182,577·344
1892	66,075·205	21,413·920	141,743·151	207,818·356
1893	144,075·518	18,222·548	136,931·406	281,006·924
1894	227,748·340	3,568·956	28,555·916	256,304·256
1895	142,757·406	....	....	142,757·406
1896	96,030·841	253·000	1,556·870	97,587·711
1897	75,804·525	13,168·200	76,190·568	151,995·093
1898	92,549·662	....	51,499·781	144,049·443
	1,410,658·016	....	1,005,888·878	2,416,544·844

PROFITS OF THE COMPANIA HUANCHACA DE BOLIVIA.

(In Bolivianos. *c*)

Years.	Gross Production.	Profits and Losses.	Dividends Paid.	Years.	Gross Production.	Profits and Losses.	Dividends Paid.
1877	1,565,784·49	470,059·42	360,000·00	1889	6,977,005·23	3,050,076·93	3,770,181·70
1878	2,189,749·39	989,814·42	840,000·00	1890	7,698,326·44	3,791,829·54	3,329,853·80
1879	2,215,882·03	742,190·84	348,570·00	1891	7,440,085·17	3,434,340·04	1,742,714·26
1880	3,489,143·42	851,972·47	755,190·34	1892	7,453,935·06	3,388,226·50	3,341,269·39
1881	3,191,630·18	742,439·93	824,906·78	1893	10,750,008·93	4,542,021·68	3,083,082·88
1882	6,034,282·49	3,146,789·46	1,981,980·00	1894	12,512,063·73	5,094,378·85	3,173,647·56
1883	5,136,788·08	2,564,521·77	2,640,000·00	1895	7,891,099·78	929,205·38	1.482,418·60
1884	4,934,439·51	2,174,677·00	1,560,000·00	1896	4,387,321·38	*a* 833,432·35	....
1885	4,858,989·32	1,343,090·76	2,160,000·00	1897	4,724,527·27	*a* 1,281,818·79	....
1886	6,258,320·63	3,168,803·36	2,160,000·00				
1887	5,183,341·26	1,563,403·63	1,140,000·00				
1888	5,549,398·16	2,028,140·36	....		119,442,121·95	*b* 41,036,019·77	34,693,815·31

(*a*) Loss. (*b*) Including also a deduction of 864,651·43 lost in the construction of the Oruro Railway and the Antofagasta Canal. (*c*) The value of the boliviano fluctuates according to the price of silver. In 1895 the average value in United States currency was 46·7c., 49·275c. in 1896, 44·925c. in 1897, and 42·175c. in 1898

The Huanchaca mine, which is the chief producer of silver in Bolivia, is situated 14,000 ft. above sea level and 400 miles from Antofagasta, with which it is connected by railway. The mine is operated upon thoroughly modern principles, but its production has been reduced lately on account of the heavy influx of water in the lower levels. Its output is beneficiated chiefly at the Playa Blanca works, owned by the company, at Antofagasta, in Chile. According to a recent statement by the company the condition of the mines is improving, the water being lowered gradually by pumping, although it is believed that a complete unwatering will not be possible until the proposed drainage tunnel has been driven. Exploration work on the west side of the mine, where rich veins have been discovered, is now being actively prosecuted. It is believed that the production may be greatly increased by working the "pacos," *i.e.*, the ores which run high in lead. A dressing works for the treatment of low-grade ores will soon be completed. Improvements are being made also at the smelting works at Playa Blanca, where eight furnaces are in operation and eight more under construction. The statistics of the company from January 1, 1884, are given in the preceding tables:

Borneo.—There is a small production of gold both in the British and Dutch divisions of this island. Lately it has been proposed to work some of the gravel beds in British North Borneo by dredging. The Borneo Mining Co. has been for three years exploiting the lode near Sarawak, treating the ore by the cyanide process. The alluvial deposits in Dutch West Borneo are believed to be rich.

Brazil.—The St. John del Rey Mining Co. had 100 stamps dropping in 1898, and during the six months ending August 31 crushed 45,761 tons of ore, which yielded £137,574 gold, or 7·69 oitavas per ton of ore.* This was a considerable increase over the production of the previous six months. During the year ended February 18, 1898, the mill crushed 82,761 tons of ore, yielding 51,101 oz. of gold, valued at £170,806. The second process plant, which was only experimentally in operation in 1897, yielded £8,711. The profit in Brazil was £68,000, less £10,726 representing the taxes of the Brazilian government. Deducting expense of management the net profit was £53,238.

The Ouro Preto Gold Mines of Brazil, Ltd., during the fiscal year ending June 30, 1898, raised 68,606 tons of ore, which yielded $351,263, at a profit of $11,946, against 53,462 tons, yielding $351,138, at a profit of $25,811, in the previous year. Thus the grade of the ore fell off from $6·568 to $5·12 per ton.

The Carrapato Gold Mines, Ltd., was organized in London to reopen an ancient gold mine in the vicinity of the St. John del Rey. The S. Bento and the Faria are also new companies. Besides these enterprises there are numerous small private concerns in Minas Geraes, each working from 10 to 15 wooden stamps. When the new companies get into full operation it is estimated that the present annual production will be almost doubled.† At present the gold mining industry of this State is hampered by the State export duty of 5% and the high cost of transportation on the Central Railway, 2% of the value of the gold being charged. The state export duty realized 132,526 mil-

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* 8 oitavas=1 troy ounce. † British Consular Report, Miscellaneous Series, No. 494, Feb. 1899.

reis in 1896 and 324,339 in 1897, the milreis being nominally equivalent to 54·6c. United States currency, though its actual value varies with the rate of exchange.

Canada.—The great increase in the gold production of Canada in 1898 was due especially to the Klondike, although there was a substantial gain in the output of British Columbia and Nova Scotia. In the other gold producing districts of the Dominion mining was conducted without special new features. The new districts in Ontario have not yet proved to amount to much.

Nova Scotia.—This province in 1898 produced 31,105 oz. of gold from 86,331 tons of ore milled, this being the largest production in its history. The system of mining and milling is undergoing a radical change. Formerly only the narrow rich veins were worked, while now wide belts of low-grade gold-bearing rock are broken and put through the mill. New mills are being erected which will stamp from 3 to 4 tons of ore per head per day where formerly less than one ton was put through. As a result of these changes $2 rock is being worked at a profit. Many of the companies operating in the province realized large profits in 1898.[*]

Yukon.—According to the statistics of the Canadian Geological Survey the gold production of the Yukon district in 1898 amounted to $10,000,000, against $2,500,000 in 1897. J. B. Tyrell, who visited the district in behalf of the Survey, reported favorably upon the richness of the alluvial deposits, which he considers to be very extensive. So far only two creeks have been really worked. These are El Dorado and Bonanza. Hunker and Dominion creeks have only been scratched, although they are known to be rich. The total number of claims worked on these creeks is about 200. The rocks underlying the proved gold-bearing area are micaceous schists and quartzites cut by many small and some large veins of quartz. As yet few traces of gold have been found in these rocks. The gold is concentrated in the recent sandy gravels on the banks of the rivers and in the beds of the smaller streams, as well as in the stream terraces and moraines on the sides of the valleys. In the bottoms of the valleys there is a deposit of well-rounded gravel 6 to 20 ft. thick lying on a very much decomposed bed rock. The gold is found in this gravel in particles from flour size up to a medium-size potato, and extends usually 1 or 2 ft. into the bed rock, which is mined and washed with the richer part of the overlying gravel. On the sides of some valleys benches of well-rounded gravel are found extending 50 or 60 ft. up the hill and in favorable localities are rich in gold. On Bonanza and El Dorado creeks, especially on their western sides, some very rich claims have been worked in a lateral moraine which extends along the hillsides at a height of about 200 ft. above the bottom of the valleys. As the small and very rich deposits are gradually washed out there will remain a large amount of gold to be recovered by grouping the claims and washing the gravel by cheaper methods.[†]

Chile.—The most important silver-mining district is Caracoles, 150 miles in-

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[*] George W. Stuart, on "Nova Scotia Gold Mines," *Engineering and Mining Journal*, March 11, 1899.

[†] J. B. Tyrrell, "The Geology of the Klondike Region," a paper read before the American Geological Society, December, 1898; abstract printed in *Engineering and Mining Journal*, Jan. 28, 1899.

land from Antofagasta and 10,000 ft. above the sea level. A few years ago the production fell off heavily, but lately it has been increasing again. There are several groups of mines, of which the most important are the Placilla and Isla. Few of the mines use steam power. They are generally operated under lease, the lessees paying the owner a royalty of 15 to 30% upon the net value after deducting mining expenses. In the north of Chile, inland from Iquique, there are rich silver mines, the chief ones being the San Juan, Santa Rosa and Huantajaya.

China.—According to a recent British consular report the plans for opening gold mines in the Chien Chang are still in progress. Considerable sums of money were subscribed by officials in Chengtu in support of the undertaking. Mr. T'ang, the Cantonese in charge of the works, recently stated that the mines were beginning to pay, and that much more may be expected when all the machinery arrives. The operations at present are confined to the Ma-Ha hills between Yueh-Hsi and Mien-Ning. According to the *North China Herald* of September 19, 1898, the motives influencing the recent Russian demands in Manchuria have been largely the movement of pioneers whose expeditions through the province of Kuldja and elsewhere have revealed that from the beginning of the western end of the Celestial Mountains, trending S. W. to the Altai Mountains, dividing Kashgaria from the Russian frontiers of Ferghana, the whole steppe is rich in gold. Chinese and Russian officers are said to have already marked out concessions in this region, and steps are being taken toward the exploitation of the deposits.

German East Africa.—Gold has been found south of the Victoria Nyanza. The discoveries are regarded as being of great importance, and a commission has been dispatched to investigate them. There are good reasons to believe in the existence of gold-bearing strata along nearly the whole of the western boundary of the Protectorate.*

Guiana.—Of the three Guianas, British and Dutch experienced a falling off in gold production in 1898; French showed a small increase. The decrease was especially large in British Guiana, where according to the official report for the fiscal year 1897-98 quartz mining was almost at a standstill, although in one or two mines development work was carried on. The decrease in production was ascribed partly to the exceptionally bad weather, which interfered with operations, especially in the Potaro district, and partly to the exhaustion of the alluvial workings in the Barima district, while difficulty in obtaining credit, due to a lack of confidence which has been engendered by unchecked dishonesty on the part of individuals intrusted with exploring expeditions, has prevented the opening of fresh ground and the extension of the district generally. With respect to the statistics of production it should be mentioned that the greater part of the gold obtained by individual prospectors in French Guiana escapes taxation and the official statistics are said by the best-informed authority not to represent more than 33% and perhaps only 25% of the actual production of the Colony.

India.—The gold production of British India showed a small increase in

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* British Consular Report, No. 474, Miscellaneous Series, 1898.

1898. There were nine producers, the most important being, as heretofore, the Mysore, Champion Reef, Ooregum and Nundydroog. The increase in production in 1898 would have been larger had it not been for the flooding of the Nundydroog mine. As it was the Mysore and Champion Reef mines were mainly responsible for the increase. The district is now equipped with 641 stamps. The statistics are given in the subjoined table:

Companies.	1885.	1886.	1887.	1888.	1889.	1890.	1891.	1892.
	£	£	£	£	£	£	£	£
Mysore	23,987	62,520	50,826	73,972	198,152	229,961	260,069	249,393
Ooregum				15,246	62,508	104,041	132,974	202,887
Nundydroog			5,737	26,487	22,947	58,906	89,845	118,026
Mysore Reefs					542	812	780	
Nine Reefs			551	3,071	5,273	868		
Kempinkote			1,682	4,189	5,714	5,242		
Mysore West								
Mysore Wynaad								
Balaghat Mysore			682	10,538	13,782	13,821	20,282	27,129
Goldfields of Mysore								
Yerraconda				380	233			
Champion Reefs								25,711
Coromandel								
Oriental								
Road Block								
Totals	23,987	62,520	59,478	133,833	304,146	413,651	508,400	623,146

Companies.	1893.	1894.	1895.	1896.	1897.	1898.	Total.	Quartz Crushed.
	£	£	£	£	£	£	£	Tons.
Mysore	254,205	202,302	247,589	440,817	497,887	622,274	3,413,395	595,743
Ooregum	280,970	256,785	264,934	247,329	209,299	197,221	1,974,189	399,759
Nundydroog	103,558	110,784	144,161	168,613	212,030	156,109	1,216,703	273,321
Mysore Reefs		1,027	4,119	7,370	2,419	11,010	28,029	15,855
Nine Reefs			4,582	241	869	9,453	24,858	23,594
Kempinkote							16,777	14,283
Mysore West		1,039	15,400	24,038	39,899	20,053	100,429	52,838
Mysore Wynaad								
Balaghat Mysore	27,830	21,907	1,122	10,215	636	1,605	149,549	22,447
Goldfields of Mysore					6,972	9,131	16,103	
Yerraconda			436				1,049	737
Champion Reefs	119,662	204,441	272,472	328,096	473,114	530,330	1,954,726	366,868
Coromandel			844	24,119	48,462	30,526	103,951	42,110
Oriental								
Road Block								
Totals	786,225	798,285	955,619	1,251,738	1,491,587	1,587,712	8,999,758	1,807,055

All the principal gold mines of India are situated in the Colar district of Mysore. The boundary of the Madras Presidency is, however, only a mile or two distant, and as the gold-bearing formation extends in that direction the most southerly mine of the group is actually situated in Madras. The mines are connected with the Madras-Bengalore Railway by a branch line and can be reached from Bombay in 40 hours and from Madras in 12 hours. The country has an elevation of about 2,800 ft. above the sea level. The climate is fairly good, but epidemics of cholera and smallpox are continually breaking out among the natives, so that Europeans run considerable risk from these diseases, while mild forms of fever and ague are common. At present there are 16 mining companies operating in the district, with 510 stamps at work.

The formation in which the gold-bearing lodes are contained is a layer of schistose rock, which in the central section of the field is about 3 miles broad.

This zone of schist runs north and south for about 15 miles, tapering gradually out to nothing at each end. It is surrounded on all sides by granite. It is difficult to describe accurately the different reefs because little is yet known of their relation to one another. In the central section of the field the main reef, known as the Champion, is worked exclusively. In places this reef is 20 ft. thick, but its average may be put down at 4 ft. It is an exceedingly well-defined lode of great regularity. At a certain point north of the central section, however, it splits into several branches, and although each of these carries gold and is worked in one mine or another beyond the split they appear to be neither so rich nor so regular as the main section. There is a series of reefs 3,000 ft. west of the Champion known as the West Reefs, several of which are worked, but although they are well defined and regular they have not yet been found payable over any large area. The ore of the Champion reef occurs in shoots which dip to the north. Some of these shoots have given out in depth, but others have increased both in width and richness.

Of the 16 mining companies operating in this district 13 are managed by the firm of John Taylor & Sons, of London. In general the mining and metallurgical methods are good, but the practice of the district is open to considerable improvement by a centralization of the hoisting and milling works. There is already a tendency to bring the mills together. Underground the work has been generally of a sound nature and the reserves of ore in most of the mines is a particularly satisfactory feature. During 1898 four of the important companies were sinking new main vertical shafts to cut the lode at depths of from 900 to 2,000 ft.

Java.—Nearly 20 new companies were organized in 1897 for mining and prospecting in the Celebes and in Borneo and Sumatra. The Netherlands India Mining Co. shipped 600 tons of ore in 1898, averaging 4·75 oz. gold.

Korea.—The chief producing districts in this kingdom are found in the northern half. Some alluvial deposits are worked, but the quartz veins, which traverse granites, gneiss and crystalline schists, are more important. The annual production of gold in Korea varies between 2,000,000 and 3,000,000 yen. The reported exports amount to only 500,000 to 1,000,000 yen, the remainder of the production being carried away by Chinese travelers, although a little is retained by rich men in the country.* According to British Consular Report No. 2,132, Annual Series, June, 1898, the export of gold dust from Korea was £150,628 in 1896 and £205,527 in 1897. The increase in the production in 1897 was due especially to the discovery of a rich mine at Ho-Yang on the plateau between Söul and Wönsan. At one time there were reported to be 40,000 persons at the new field, but the number was probably largely exaggerated, and miners were doubtless withdrawn to a certain extent from the old fields in the north, where the production was temporarily arrested. The exportation of gold from Korea is chiefly to China and Japan.

Madagascar.—The hopes of establishing a successful gold-mining industry in this island have been given up by the French government, which formulated

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* K. Nishiwada, formerly Geologist of the Mining Bureau of Korea, in *Useful Minerals of Korea*

a formidable series of mining regulations and otherwise blocked the development of resources, which there is reason to believe are of considerable value.

Mexico.—There was a substantial increase in the gold production of Mexico in 1898. Our statistics include no allowance for exports, which is made by some authorities. There are doubtless certain portions of Mexico, such as the territory of Lower California, where it is difficult to control the output on account of the large extent of seacoast, but the production of gold in these districts is not large, and consequently it does not seem proper to make an important allowance for surreptitious exports. Although our statistics for Mexico are probably somewhat under the mark, this is the case with all statistics that are based on actual returns.

Peru.—The government of this republic, which is of importance chiefly as a silver producer, does not collect any statistics as to output, and formerly it has been our practice to estimate this on the statistics of exportation and the deposits at the mint; but in 1897 this method ceased to be trustworthy, owing to the remission of the export duty and the closing of the mint to the free coinage of silver. In our statistics for 1897 we used the estimate of the Casa Nacional de Moneda for the production of Peru, namely, 58,368 kg. A writer in *El Economista Mexicano* reports: Bullion—Cerro de Pasco, 1,800,000 soles; Empresa de Casapalca, 1,202,000; Castro Virreina, 360,000; Empresa de Ticapampa, 350,000; Empresa de Caylloma, 150,000; Trujillo and others, 120,000 —total, 3,980,000 soles; silver sulphides, 3,500,000; silver ores, 2,250,000; grand total, 9,730,000 soles, or $4,728,780 (245,996 kg.). This estimate, although vastly larger than that of the Lima mint, is less than that given to the United States Minister to Peru by Don Alejandro Garland, a Peruvian economist of high reputation, namely, 304,400 kg. If either of these figures is correct it is obvious that the silver production of Peru has been greatly underestimated, but in the absence of more specific information we are not ready to adopt them. The exportation of bar silver from Peru in 1897 amounted to 6,448,567 soles and coin to 1,927,454 soles. The exports in 1896 were 4,576,511 and 1,485,433 soles respectively.

According to British Foreign Office Report No. 2,117, Annual Series, the late discoveries of quartz and placer gold mines in the districts of Sandia and Carabaya have induced many persons to prospect there, but as a rule they have been disappointed in the results, chiefly on account of the many difficulties they have had to contend with outside of the natural ones arising from lack of roads; and the action of the local authorities, who do not seem to encourage the development of that part of the country. However, there is no doubt that gold in large quantities exists in these regions, and when the difficulties have been overcome the mines will begin to be productive.

Philippine Islands.—According to George F. Becker gold is found at numerous localities in the archipelago, from northern Luzon to central Mindanao. In most cases the gold is found either in existing watercourses or in ancient stream deposits. In Mindanao some of the gravels are in an elevated position, and adapted to hydraulic mining. In the province of Abra, at the northern end of Luzon, there are placers, and the gravel of the river Abra is auriferous. In Lepanto there are quartz veins as well as gravels. In Benguet the gravels of the river Agno carry gold. There is also gold in the Province of Bontoc and in Nueva Ecija. The most important of the auriferous provinces is Camarines Norte, where the townships of Mambulao, Paracale and Labo are especially well known as gold-producing localities. At Paracale there are parallel quartz veins in granite, one of which is 20 ft. in width. Besides the localities mentioned, many others in this province have been worked by the natives.

The islands of Mindoro, Catanduanes, Sibuyan, Samar, Panay, Cebu and Bohol are reported to contain gold, but no exact data are accessible. At the south end of the island of Panaon, which is just to the south of Leyte, there are quartz veins, one of which has been worked to some extent. It is 6 ft. in thickness, and has yielded from $6 to $7 per ton. In the island of Mindanao there are two known gold-bearing districts. One of these is in the Province of Surigao, where Placer and other townships show gravels and veins. The second district is in the Province of Misamis. Near the settlement of Imponan, on the Gulf of Macajalar, there are said to be many kilometers of gravels carrying large quantities of gold, with which is associated platinum.

According to F. W. Voit little is known of the geological structure of the islands beyond the fact that they are of volcanic origin. The deposition of sedimentary material is constantly in progress, as also is volcanic activity. The gold occurs chiefly in quartz veins, the chief minerals associated with it being tellurides, sulphides, hornblende, feldspar and magnetite. The reason that the Philippines are so little known as a gold field is due to the absence of large alluvial deposits and of free gold.*

Rhodesia became a gold producer in 1898, the division of Mashonaland having made an output of 6,252·5 oz. crude, and Matabeleland 19,002 oz., according to the statistics of the Rhodesia Chamber of Mines. The Rhodesian gold is worth $17·17 per oz. The annual report of the Rhodesia Chamber of Mines for the year ended June 30, 1898, affords satisfactory evidence of the development of the gold fields of Mashonaland and Matabeleland. Dr. H. Sauer, the president, stated that more work was done in the last year than during the previous three years in which the country has been occupied. At four of the principal mines—the Bonsor, the Dunraven, the Geelong and the Tebekwe—the total sinking and driving completed in the year was 20,350 ft., the ore in sight was reported to amount to 165,000 tons, the aggregate expenditure amounted to £310,000, and 120 stamps were under erection or in transit. He stated furthermore that the veins give promise of depth and permanence, in which opinion he was supported by John Hays Hammond and other geologists, and says there is absolutely no doubt, in the opinion of those controlling several of the mines just about to start reduction operations, that the value per ton is sufficient to show that a profit will be obtained. Arrangements have been made whereby the cost of dynamite will be in future 48s. 3d. per case and that of blasting gelatin 63s. 6d. per case, delivered in Bulawayo, as compared with 75s. and 97s. 6d., respectively, the prices ruling in Johannesburg at the present time. Moreover, the cost of native labor in the first half of the current

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* *Berg- und Hüttenmannische Zeitung*, 1898, pp. 251-254.

year averaged under 40s. per month, including the expenditure for food, etc., while in the Rand district native labor is said to cost 75s. per head per month. These reductions are due largely to the completion of the railway to Bulawayo in November, 1897. Foodstuffs are carried over this railway at a rate of a halfpenny per ton mile and without any customs charge.

According to an official report of the mining industry of Zululand by J. Jervis Garrard, but little attention has yet been paid to gold mining, the principal work having been at the Watkins mine in the Melmoth district by the Metropolitan Exploration Syndicate. Numerous locations have been made on the banket beds at the Upper Insuzi, owing to the discovery of the extension of what is known as Dickson's lead, which had not previously been traced. The Insuzi banket beds have hitherto been prospected only on one or two lines, several miles apart, but there is reason to believe that by further prospecting across the strike of the beds some series richer than anything hitherto met with will be discovered. The Insuzi banket formation crosses the Buffalo River into Natal in the district between Umsinga and Helpmakaar, where it underlies the superimposed horizontal coal measures. A rich strike of quartz, assaying 4 to 5 oz. to the ton, was made on the Umfuli Mission farm near Melmoth, but little has been done yet toward opening up the reef.

Russia.—The production of gold in 1898 in Russia and Siberia was $25,136,994, against $21,538,490 in 1897. The statistics for 1898 are based on receipts of gold at the Imperial Mint. From the total receipts of the year the amount contained in the first convoy of January was deducted, this having been counted previously in the production of 1897. The remainder was increased by 10% to cover gold disposed of clandestinely, as explained in our previous reports. According to the Russian minister of finance there are at present 10 joint stock companies engaged in gold mining in Siberia, of which six are Russian and four foreign. The most important of the former is the Lena Gold Mining Co., which owns the largest placers in the country, exceeding in extent those of the Imperial Cabinet. Lately the company has not been paying dividends. A new company not included in this enumeration is the Oural Gold Fields of Western Siberia, recently organized in London to operate the Michaelovski and Ouspenski mines in the Kotchkar district of the Ourals.

The placers of West Siberia, of which the exploitation was begun in 1829, are said to be now as good as exhausted, the rich deposits having been worked out entirely, and although some of the poorer may still be worked by improved machinery the output expected from this source is small. In the place of the large proprietors the industry is now in the hands of small traders who use it as a means of disposing of their goods at double prices to their workmen. However, an extension in quartz mining is to be expected soon. Veins have been discovered on the Kuludschum River, in the Kalbinski Mountains.

According to the Russian engineer, Mon. Toravako-Pokorski, the Khanate of Bokhara, which is entirely under Russian influence, produces from 115 to 130 kg. of placer gold per annum, the entire output being exported to Afghanistan. Travelers through the Russian Pamirs report the existence of many veins of gold and silver ore and state that some day the country will be a large mineral producer. According to R. Helmhacker, gold-bearing gravel has been found in Russian Turkestan in the streams flowing from the eastern Pamirs. So far the most important deposits have been found in the upper tributaries of the Sir-Daria and in the small streams falling into the steppe lakes Kara-Kul and Biul-Kul. Very little prospecting has been done so far, and no regular workings have been carried on.

Servia.—Steps have been taken to exploit the gold of the Timok Valley.

Siam.—There are two gold mining companies at work in Siam, one at Kabin and the other at Wattana. The production of the former is about 2,400 oz. per annum, but this output is likely to be largely increased with the installation of new crushing machinery, which has been ordered. There are two or three other gold mining concessions, and considerable prospecting is going on.[*]

South African Republic.—The output of the Witwatersrand in 1898 was 4,295,609 oz. (£15,141,376); that of the outside districts was 259,413, (£902,759). This was an immense increase in the production of the Witwatersrand and a small falling off in the output of the outside districts.

[†]Of the increase in the Witwatersrand production the mills contributed 63·5%, the concentrates and by-products 0·6%, the sands treatment 29·4%, the slimes 6·2%, and the returns from banks 0·3%. The mills crushed 37·7% more ore than in 1897, while the value of the gold recovered showed an increase of 40·6%, indicating a higher grade of ore milled. The cyanide works treated 39·2% more sand, the value of their recovery increasing by 44·8%. The slimes recovery was reported separately for the first time in 1898. Many slimes plants were in operation during the year and a good many more are in course of construction.

During 1898, 77 companies in the Witwatersrand reported yields of gold against 69 in 1897, the average number of companies at work per month having been 63·5, with an average total of 4,765 stamps, against an average of 51 companies, milling with 3,567 stamps, in 1897. The duty per stamp per day increased from 4·53 tons to 4·68 tons. The tonnage mined was 8,965,960, of which 1,634,514, or approximately 18·2%, was rejected as waste. These sorting operations increased largely, the percentage of barren rock thrown out having been as high as 40 in certain cases. This has been necessitated by the introduction of air drills to compensate for the shortness of native labor, whereby a large amount of country rock is unavoidably broken down with the vein matter, which has subsequently to be sorted out by hand, it having been proved to be cheaper to do this than to put everything through the mill. The tonnage milled in 1898 was 7,331,446, an increase of 2,006,091 over 1897. The average grade of the ore was 11·718 dwt. (41s. 4d.) per ton in 1898, against 11·397 (39s. 9d.) in 1897. The yield was 40s. 11·1d. per ton milled, against 39s. 7·2d. in 1897. The improvement was due to better efficiency in sorting, to the falling out of low-grade mines, to the additional recovery from slimes treatment, which in 1898 yielded 78,887 oz. (£282,220), thus accounting for 9·2d. of the increase in value per ton milled, and to the coming into

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[*] British Consular Report, No. 2,190, Annual Series, October, 1898.

[†] This review of mining in the Transvaal in 1898 is largely an abstract of an elaborate review by George Rouliot, president of the Chamber of Mines at Johannesburg, at its annual meeting in January, 1899.

operation of new producers, mostly located in rich sections of the district. The main reason of the great increase in the total output of the Witwatersrand was the additional production obtained from the deep level mines, of which five more came into operation, making a total of 11, although there was only an average of 8·6 at work throughout the year, with an average of 864 stamps per month. Their mills crushed 1,403,455 tons, producing 961,194 oz. (£3,376,497), or 22·3% of the total output. Their yield amounted to 48s. 1·4d. per ton, against the general average of 41s. 4d. Since some more deep levels will come into operation during 1899 and the development of mines located on the third row is proceeding steadily the production of gold in the district is almost certain to show a further increase this year.

The total amount paid in dividends by 40 companies out of the 77 was £4,834,160, or £2,120,590 more than in 1897, and was 31·93% of the total production, against 25·64% in 1897. The deep levels paid £955,000. The outside districts of the Transvaal paid about £152,345, which is not included in the above total. Reckoning the cost of production as being the difference between the dividends paid and the total amount realized, the average in 1898 was 28s. 1·4d., against 29s. 6·7d. in 1897. As a matter of fact the reduction in cost on the part of the old mines was really greater, since the average is raised by the costs of the deep levels, which, taken by themselves, showed an average cost of 34s. 6·1d. in 1898.

In mine management there was generally an improvement throughout the district, but the industry is now threatened with further imposts on the part of the government. Concessions have been made in the railway rates and in the price of dynamite in consequence of the recommendations of the Industrial Commission, but in 1899 the gross yield of gold from mynpachts will be subject to a tax of 2·5% and the net profits on gold from ground other than mynpachts to a tax of 5%. If these taxes had been levied in 1898 they would have yielded about £300,000, equal to 6·5% on the total dividends. In some respects the industry was worse off in 1898, especially on account of increased drunkenness among the natives and difficulties connected with the administration of the pass and liquor laws.

COMMERCIAL STATISTICS OF GOLD AND SILVER.

The average prices of gold and silver in New York and London in 1896–98, as computed by the *Engineering and Mining Journal*, are shown in the following table:

Month.	1896.		1897.		1898.		Month.	1896.		1897.		1898.	
	London. Pence	New York. Cents	London. Pence	New York. Cents	London. Pence	New York. Cents		London. Pence	New York. Cents	London. Pence	New York. Cents	London. Pence	New York. Cents
January........	30·69	67·13	29·74	64·79	26·29	56·77	August.........	30·93	67·34	24·93	54·19	27·48	59·54
February.......	31·01	67·67	29·68	64·67	25·89	56·07	September......	30·19	65·68	25·66	55·24	28·05	60·68
March..........	31·34	68·40	28·96	63·06	25·47	54·90	October........	29·68	65·05	26·77	57·57	27·90	60·42
April..........	31·10	67·92	28·36	61·85	25·95	56·02	November.......	29·46	64·98	26·87	57·93	27·93	60·60
May............	31·08	67·88	27·86	60·42	26·31	56·98	December.......	29·70	65·24	26·83	58·01	27·45	59·42
June...........	31·46	68·69	27·58	60·10	27·09	58·61							
July...........	31·45	68·75	27·36	59·61	27·32	59·06	Year..........	30·67	67·06	27·55	59·79	26·76	58·26

COINAGE OF THE MINTS OF THE UNITED STATES.

Year.	Gold.	Silver.	Year.	Gold.	Silver.	Year.	Gold.	Silver.
1893........	$30,038,140	$12,560,935	1895........	$59,596,357	$5,698,010	1897........	$76,028,485	$18,846,701
1894........	99,474,913	6,024,898	1896........	47,053,060	23,089,899	1898........	77,985,757	13,084,633

UNITED STATES: IMPORTS AND EXPORTS OF GOLD AND SILVER.

	1897.			1898.		
	Exports.	Imports.	Difference.	Exports.	Imports.	Difference.
Gold:						
Coin and bullion....	$34,174,182	$29,081,760	Exp. $5,092,422	$16,113,258	$153,145,259	Imp. $137,032,001
In ores................	102,219	4,941,052	Imp. 4,838,833	81,696	5,006,593	Imp. 4,924,897
Totals............	$34,276,401	$34,022,812	Exp. $253,589	$16,194,954	$158,151,852	Imp. $141,956,898
Silver:						
Coin and bullion....	$58,352,274	$12,146,750	Exp. $46,205,524	$53,673,605	$9,571,810	Exp. $44,101,795
In ores................	309,018	20,935,552	Imp. 20,626,534	123,499	19,554,446	Imp. 19,430,947
Totals............	$58,661,292	$33,082,302	Exp. $25,578,990	$53,797,104	$29,126,256	Exp. $24,670,848

GREAT BRITAIN: IMPORTS AND EXPORTS.

Year.	Gold.			Silver.		
	Imports.	Exports.	Difference.	Imports.	Exports.	Difference.
1895............	£36,005,999	£21,369,323	Imp. £14,636,676	£10,669,662	£10,357,436	Imp. £312,226
1896............	24,468,337	30,123,925	Exp. 5,655,588	14,329,116	15,048,134	Exp. 719,018
1897............	30,808,853	30,808,571	Imp. 282	18,032,091	18,780,988	Exp. 748,897
1898............	43,721,460	36,590,050	Imp. 7,131,410	14,677,799	15,623,651	Exp. 945,852

FRANCE: IMPORTS AND EXPORTS.

Year.	Gold.			Silver.		
	Imports.	Exports.	Difference.	Imports.	Exports.	Difference.
	Francs.	Francs.	Francs.	Francs.	Francs.	Francs.
1895............	253,875,195	244,381,198	Imp. 9,493,997	141,152,976	78,459,257	Imp. 62,693,719
1896............	298,839,299	310,900,667	Exp. 12,061,368	155,750,771	97,260,425	Imp. 58,490,346
1897............	293,456,750	131,858,875	Imp. 161,597,875	178,424,150	192,755,700	Exp. 14,331,550
1898............	198,715,221	312,858,950	Exp. 114,143,729	195,418,221	189,935,299	Imp. 5,482,922

SHIPMENTS OF SILVER FROM LONDON TO THE EAST.

Year.	India.	China.	The Straits.	Japan.	Totals.
1896..................	£4,838,778	£740,285	£824,536	£1,029,300	£7,432,800
1897..................	5,805,006	559,136	739,203	....	7,103,345
1898..................	4,312,057	764,750	405,105	....	5,481,912

GOLD HOLDINGS IN 1898. (*a*)

Bank.	January 1.	July 1.	December 31.
Associated New York..........	106,588,500	186,070,200	168,965,700
England........................	156,936,340	185,515,210	146,689,155
France.........................	388,110,000	375,148,100	364,626,800
Imperial German..............	222,085,000	223,870,000	128,150,000
Austro-Hungary...............	151,980,000	174,745,000	179,700,000
Netherlands....................	13,145,000	48,170,000	55,310,000
Belgian National..............	21,180,000	21,850,000	15,475,000
Spain...........................	47,155,000	14,500,000	21,575,000
Italy............................	60,055,000	75,500,000	74,630,000
Russia..........................	579,665,000	559,345,000	495,715,000

(*a*) The holdings of the Imperial Bank of Germany and the Belgian National Bank includes silver also.

THE GOLD FIELDS OF FRENCH GUIANA.*

BY E. D. LEVAT.

PUBLIC attention has been attracted in a particular way toward the auriferous deposits of the Guianas since the discovery of the rich beds situated in the territory disputed by Brazil and France, which extends from the southern border of French Guiana, near 4° N. lat., as far as the mouths of the Amazon

GEOLOGY.—It was believed for a long time that the auriferous formation of the Guianas was composed of two parallel zones side by side, in which the ore-bearing strata were found segregated. Maps have been published showing indications of this kind. A closer examination of the facts shows that this is not the case.

The Auriferous Formations.—It is first of all easy to recognize that the placers, while spreading out and following lines parallel to the general folding of the country, that is to say approximately east and west, depend on the auriferous strata at the contact of the gneiss or schistose rocks with the underlying granite. This feature correlates the auriferous formations of the Guianas and also those of Siberia and New Zealand. The placers of these three countries belong to the class of shallow placers, and present (disregarding climatic conditions) a striking similarity.

The auriferous beds are situated on the zones of contact between the fundamental granite and stratified rock; and at every granitic alignment there are two corresponding auriferous zones, disposed symmetrically with respect to the granite. We can thus substitute for the crude notion of merely parallel auriferous zones the more exact conception of two zones of contact between granite and gneiss, which usually follow the direction of the general strike of the granite, but which may present more diverse forms in following the changes of direction of the lines of contact.

Enrichment of Auriferous Zones.—These general indications assist in limiting the areas in which successful search for placers can be made, and therefore facilitate it; but one can go still further and determine in these zones the parts which present the greatest probabilities of maximum richness.

Eruptive Rocks Accompanying Gold Deposits.—In all countries presenting the general characteristics of enrichment by zones of contact between granite and stratified rocks it is a general observation that the gold-bearing veins or beds are usually accompanied by eruptive rocks, of which the character varies from one country to another, but which have in common the fact that they are found in conjunction with gold in its primitive or secondary deposits. In other words, it may be stated that there can always be found in the neighborhood of placers, or auriferous veins or beds, an eruptive rock differing from granite, though possibly derived from the latter by crystalline or chemical modifications.

Diorites or Greenstones.—In Guiana this special rôle has been played by diorites and diabases, which occur in numberless dikes and form the heights dominating the country. Without entering here into the details of the petrography

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* This paper is a resumé of a Report to the French Minister for the Colonies, which was published in full in *Annales des Mines*, Vol. I., 1898.

of these rocks it may be stated that the diorites of Guiana, generally well crystallized, with large crystals of labradorite, are ordinarily dark green and contain almost always an abundant proportion of pyrite, certain specimens containing as much as 5%. Amphibole is generally predominant, and in certain places, as in the town of Cayenne, where it forms, under the name of "grison," the semaphore signal hill, the rock graduates into a true amphibolite.

These dioritic rocks cover a considerable area, not only in French Guiana and in Dutch Guiana, but also in British Guiana and a part of Venezuela. They are also seen in the auriferous formation of the disputed Franco-Brazilian territory. In all this vast expanse the diorites are constantly associated with the auriferous rocks. It may be understood, therefore, that the maximum richness will be obtained where the zones of contact between granite and the stratified rocks are associated with the diorites or the diabases. These conditions are fulfilled by such placers as St. Elie, Dieu Merci, Eliseé, Pas-trop-tot, the group of Awa, etc., which have been the subject of test workings sufficient to afford evidence of their geological origin.

Radial Formation of Placers.—It may be well to remark here, in order to comprehend the placers just cited (and especially the group of St. Elie in the basin of the Sinnamary), the phenomena of radial formation which I have often had occasion to verify in other parts of the world, and which it is important to keep in mind. After having stated the exact correlation which exists between the auriferous formation and an eruptive rock, as shown in a placer country, an important occurrence of the eruptive rock having given birth to the famous placers, we conclude that these placers, far from being situated parallel on the opposite slopes of the mountains, show on the contrary a radial formation around a definite point of emission of the eruptive auriferous rocks.

In general, prospecting for placers in the Guianas can be conducted on the three following rules: (1) The placers subdivide on lines of contact between the granite and schistose, micaceous, or gneissic rocks. These zones follow along the quieter parts of the rivers. (2) On the lines of contact, the maximum enrichment is met in the vicinity of the diorite or the diabase outcrops. (3) When a paying placer has been found, not only should the river on which it has been discovered be prospected up stream, but also all other streams which come from the same mountain rock mass—applying, in a word, the rule of the radial formation of placers.

"*Roche à Ravets.*"*—The diorite rocks, owing to the ease with which they decompose, have given birth to a vast deposit of ferruginous rocks having the aspect of ordinary limonite, which overlie in an almost continuous layer the gneisses and mica schists in undulatory stratification.

The richness of the diorites in iron, in the oxidized state and as pyrite, of which they often contain large quantities, has produced by slow disintegration

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* In lieu of a better English equivalent this term might be freely translated, "ferruginous cement." As explained further on, however, it differs from the cement (or conglomerate) of the California deep gravels not only in point of origin, but also in its greater softness and friability, high percentage of iron, and absence of rounded pebbles or bowlders. It could also be called a limonite, though not being a true iron ore that designation might be misleading and hardly more satisfactory than the other. To avoid confusion, therefore, the French expression is retained.—EDITOR.

of these rocks, due to humidity, an accumulation of ferruginous deposits at the foot of the eruptive outcrops—in a word, the formation of the "roche à ravets." The characteristic aspect of this rock and its extreme frequency in all countries have caused it to be described by all writers on the geology of Guiana, including Le Blond; but hitherto it has occurred to no one to study its real mode of formation or determine its exact composition, particularly its tenor in the precious metals. It is evident that if the origin of the gold is found to be in the diorite rocks, the secondary rocks derived from them should contain, at least in part, the precious metals originally in them.

The erosion which produced the Guiana placers having been posterior to the formation of these limonites, as can be proved at once by a simple inspection of any placer in Guiana, it was natural to suppose that the disintegration of the "roche à ravets," of which pieces are frequently found in the auriferous alluvium itself, constitutes, at least in part, the source from which the placer gold has been extracted by the agencies of erosion in their secular action.

Quantity of Gold in Diorite.—This does not depend solely on the presence of iron pyrite. Specimens of diorite may be found with no traces of decomposition, containing grains of gold visible to the unaided eye and giving, by microscopic examination of thin sections, very clear characteristics of this rock, and in which, therefore, the presence of free gold cannot be considered to be the result of oxidation replacing the iron pyrite. Nevertheless, as a general rule, the specimens of diorite that I have assayed have generally given a proportion of gold and silver very nearly equal to that in the pyrite.

TENOR OF GUIANA DIORITES.

Locality.	Contents.	Grams per 1,000 Kilos.	
	FeS_2.	Gold.	Silver.
	%		
Diorite from Roche Creek	5·2	2·0	6·0
Diorite from Pichevin Creek	4·1	1·5	2·0
Auriferous diorite from Maripa (visible gold)	0·5	24·0	4·0
Grison, telegraph hill, Cayenne	1·4	0·24	0·02

Study of the "Roche à Ravets."—The true character of this secondary rock d its relation to the diorite from which it has been derived have not been early recognized by my predecessors. Many of them have confounded it ith the ferruginous conglomerates which are frequently found in the country, d with which a complete similarity of color makes confusion possible, on a erely superficial examination. It is sufficient to examine casually one of ese ferruginous conglomerates to prove the presence of numerous rounded bbles of quartz, which are, on the contrary, entirely absent in the true roche à ravets." The fracture of this rock presents the characteristic aspect the iron ore deposited from solution (bog iron ore), that is to say, a spongy ss of hydrated iron oxide, of which the interstices are filled with ocher nerally of light color, without any admixture of rounded pebbles. But aside m these lithological characteristics, the modes of formation of the "roches à ravets" and of the conglomerates are so different that confusion is absolutely impossible to an experienced eye.

The ferruginous conglomerates form the regular horizontal beds on the side or in the bottom of the true valleys, while "roche à ravets," which constitutes for the most part the superficial covering of the hills and the mountains in which the valleys are formed, does not form horizontal beds, and appears at different altitudes, which indicates clearly that its formation is not due to a purely mechanical alluvial process, by deposition in a current of water.

ANALYSES OF SPECIMENS OF "ROCHES A RAVETS."

Locality.	Fe_2O_3.	Al_2O_3.	SiO_2.	CaO.	P_2O_5.	MgO	H_2O (combined.)
	%	%	%	%	%	%	%
Maripa, No. 1.	54·70	12·10	8·55	5·50	0·67	6·10	7·80
Maripa, No. 2.	59·40	14·80	9·50	6·80	0·02	5·40	9·10
Awa, Roche Creek.	60·35	14·10	7·40	....	....	....	14·10
Awa, Pichevin Creek.	59·40	14·50	6·50	4·80	1·02	3·10	10·10
Awa, Central Settlement.	58·25	13·15	8·10	....	....	....	....

Let us compare these with the analyses made on the diorites from respective beds where the specimens of these "roches à ravets" were taken:

ANALYSES OF GUIANA DIORITES.

Components.	Maripa, No. 1.		Pichevin Creek.		Components.	Maripa, No. 1.		Pichevin Creek.	
	Decomposed Diorite.	"Roche à Ravets."	Unalter'd Diorite.	"Roche à Ravets."		Decomposed. Diorite.	"Roche à Ravets."	Unalter'd Diorite.	"Roche à Ravets."
	%	%	%	%		%	%	%	%
FeO	17·60	....	29·16	....	P_2O_5	....	0·67	....	1·02
Fe_2O_3	....	54·70	....	59·40	FeS_2	5·10	....	1·14	....
Al_2O_3	10·07	12·10	8·1	14·50	MgO	1·10	4·10	0·85	3·10
SiO_2	51·77	8·55	56·3	6·50	H_2O (comb.)	4·35	14·40	2·10	10·10
CaO	6·37	5·50	2·6	4·80	Alkali	3·40	....	pas dosé.	....

Two very distinct features are proved: (1) The increase of the iron contents in the "roche à ravets" over that of the diorite; (2) the presence of phosphorus in the "roche à ravets," while this element is not present in the diorite.

These facts clearly explain the mode of formation, by deposits, of the limonite iron, because there is no other name by which to characterize a rock having a composition that contradicts the analysis given for the "roche à ravets."

The diminution of the silica and the disappearance of the pyrite (transformed into hydrated peroxide of iron) also show clearly the mode of formation of the "roche à ravets" as a secondary product arising from the decomposition of the diorites and of the surrounding schists; these last explain the large proportion of alumina and magnesia contained in the "roche à ravets."

Gold Contents of the "Roche à Ravets."—In the "roche à ravets" the gold is usually found in the metallic state. The characteristic of its occurrence is its association with a considerable quantity of silver. At times this metal is found containing only traces of gold.

CONTENTS IN PRECIOUS METALS OF DIFFERENT "ROCHES A RAVETS" OF GUIANA.

No.	Locality.	Per 1,000 Kilograms.		Fineness of Gold.
		Gold.	Silver.	
		Grams.	Grams.	Thousandths.
1	Maripa, No. 1		20	
2	Maripa, No. 2		20	
3	Maripa, Tortue Creek	4	16	200
4	Maripa, average specimen, from a boring 8 meters deep	5		825
5	Maripa, Jean Creek workings	5		800
6	Awa, Pichevin workings	5		850
7	Awa, Central Settlement	2		850
8	Maripa workings, average specimen	18½		866
9	Maripa, Roche à Ravets, pisolitic	3		700
10	Maripa, specimen 800 meters from Central	73		850
11	Specimen from Beiman Creek	7		850

All these specimens were taken from the surface, or from the softer parts of the rock, except No. 4, from which they were loosened simply with a pick.

Ascertaining the Rich Part of a Bed of "Roches à Ravets."—It will be seen that because of the large variations in tenor, as shown by the above table, it is necessary, in order to ascertain whether the "roche à ravets" in any region may or may not be worked profitably, to make a complete test of it by means of borings and numerous analyses. This is an easy task, since the rock is soft and formed in extensive deposits at the surface, and can consequently be bored by very simple means. The diversity of composition of this rock is not surprising if we consider its mode of formation, that is, by the weathering of the diorite rocks, which have very variable contents in the precious metals. It is shown, from what has preceded and from what is shown in the detailed results below, that there exists in Guiana the "roche à ravets" presenting at the very surface of the soil a content in gold sufficient to allow profitable working.

Conclusion.—It follows from this study that the enrichment in precious metals of the "roche à ravets" is due solely to the concentration in this secondary formation of the precious metals primarily contained in the eruptive rock which has given birth to it, and its variations in composition are explained by the corresponding variations in the primitive rock. It is furthermore important to recollect that the content in precious metal of this secondary rock, far from being uniform, occurs only at certain points. Consequently it is necessary, before judging as to the profitable working of a deposit of this class, to make some preliminary tests.

AURIFEROUS VEINS.—At every step in the old formations we meet schists and mica schists as well as veins of quartz which form a series of waterfalls on a large number of the rivers, often permitting the prediction, by a glance at the map where these falls are indicated, the place where we shall find the quartz veins. It is characteristic of these veins to be apparently intercalated in the stratified rocks containing them, and consequently to follow the general direction of these strata, which is approximately east and west. Their dip varies with that of the inclosing rocks; it approaches the vertical in the vicinity of the eruptive rocks which have affected the contour of the country.

The thickness of these beds varies from a few centimeters to several meters, and in the latter case the quartz, generally white, which forms the vein, having better resisted erosive agencies, forms in the rivers pyramids or sugarloaves, of which several are famous and have been quoted by the most ancient authors.

The Quartz.—The quartz of which these veins are composed is generally milky white, changing sometimes into a light chamois or grayish blue. This last shade is usually a sign of richness. Iron pyrite is frequently found in it, associated with mispickel, but with a very noticeable predominance of pyrite.

TENOR IN PRECIOUS METALS OF SEVERAL GUIANA QUARTZ VEINS.

No.	Locality.	Per 1,000 Kilograms.		Fineness of Gold.
		Gold.	Silver.	
		Grams.	Grams.	Thousandths.
1	Bed at mouth of Sparwine Creek		24	
2	Sparwine Creek, 3 km. from No. 1	25		700
3	Vein at Village of Apatou (Ste. Hermina)			700
4	Vein crossing the Awa at Gaillot placer	20		850
5	Aponci-Kondé vein	36		800
6	Same formation—bluish quartz	22		800
7	Beiman Creek	60		850
8	Maripa vein, average sample	4		866
9	Maripa sample containing iron pyrite	60		820
10	St. Elie	64		820
11	Adieu Vat (10 tons sold at Paris in 1897)	229	4	820

The greater part of these quartz beds is unprcductive, only the quartz found in the neighborhood of the diorite outcrops showing a concentration in gold running as high as 300 g. and upward to the ton of ore.

As to the continuity of this value and regularity of the tenor in course and depth, it is impossible in the absence of any developed workings to give an opinion on this question founded upon fact. It is undeniable that such high assays as shown in sample No. 11 of the table, and which have been obtained, not from a single specimen only, but an average taken from a lot which has been sold and paid for according to that value, is one of the most interesting facts. But on the other hand, as in all the deposits which belong to the type of stratified beds, we must expect a kind of chain-like deposits, with frequent variations of size, which demand preparatory development to allow a regular production of ore for milling.

The only point where development work is being done upon a gold quartz vein is at Adieu Vat, a placer belonging to the St. Elie Co. This placer is near the Sinnamary River, navigable for steamboats just opposite the mine.

All the Guiana placers belong to the class of superficial or shallow placers. Their number is unlimited, so to speak, and we can say that generally all rivers and watercourses which are below the auriferous zones, above mentioned, contain gold. In opposition to the opinion generally adopted by the Guiana prospectors, gold in grains and nuggets, the only kind they care for, has not been carried a long distance from their primitive bed by erosive agencies. On the contrary, they are very near, and we may even assume, especially as far as the nuggets are concerned, that these have hardly changed places since the time of their liberation from the original rock. Medium-size gold, and especially fine gold, are, on the contrary, easily carried to a longer distance, and the notably auriferous rivers, like the Appronague, Sinnamary, and Mana, carry some very fine gold in suspension.

Alluvial Beds of Quartz.—A general and very characteristic feature, which has been recognized without exception by all observers who have studied the country, is the presence, almost without exception, in all the regions of Guiana covered with either Quaternary or recent alluvium, of a bed of white quartz, of varying thickness, which is found not only in the valleys, but even on points which at first sight are situated out of reach of the sedimentary waters.

The depth of the layer underneath the actual soil varies from a few centimeters to several meters along its entire level. It is sometimes covered with pure white clay, or again with a gray or yellow clay; sometimes with clay mixed with pebbles, but the latter are of a different nature from those which are found in the auriferous alluvium, and may be remnants of schist, granite, gneiss, etc., while that which in Guiana is called the "bed" is composed of white quartz without admixture.

Distribution of the Quartz Bed.—For example, the bed is found in the low and intermediate lands of Guiana, principally in all the regions of the Savanes, where it is sometimes transformed to a white sand as it nears the sea. On the contrary, as we approach the first rise, the strata converge; finally, in the placer region, they are limited strictly to the width of the actual valley.

The co-existence of the auriferous bed, quartz, and white sand, which form the substrata of the low grounds, is then plainly established, and brings us back to the beginning of the Quaternary period, when the alluvial lands of Guiana were submerged; a period when the active erosion to which most of the country has been subjected destroyed all the rocks except quartz, which, owing to its hardness, has been preserved from complete disappearance, while the other materials have contributed to the formation of large estuaries and muddy shores. The deposit exists in the beds of the present rivers. Consequently the rivers of Guiana, without exception, pass over the bed, although the peculiarity is sometimes hidden by sandy banks or recent gravels which have been carried down by the rivers.

Another very important characteristic of the bed is that it is found both in the auriferous and in the barren regions, upon a layer of clay of a particular color and aspect, under which it does not continue. All the prospectors of Guiana are well acquainted with this peculiarity, and in their test-pits, as soon as they reach this clay, or according to their picturesque expression, as soon as they have "touché," they stop probing immediately and carefully gather the last few centimeters of the layer in immediate contact with the clay, and also a thickness of from 5 to 10 cm. of the latter; these two parts constitute the richest portion of the alluvium. They never penetrate underneath, and numerous probings made in this layer of clay prove that they were perfectly right in following this rule. In fact, the clay constitutes the actual bed-rock, being nothing but the old bed-rock decomposed. The gold found in nuggets or in grains in the clay was introduced there when the bed-rock was still rock, and has remained there after the transformation of the rock into clay.

PROGRESS IN THE METALLURGY OF GOLD AND SILVER.*

BY WALTER RENTON INGALLS.

IN continuing the review begun in the last volume of this series the same line of division is preserved between the metallurgy of gold and silver and the metallurgy of silver-lead, silver-copper, etc., as was established in the first paper, *i.e.*, limiting the subject to the recovery of gold by amalgamation, chlorination, and cyanide lixiviation and the recovery of silver by pan amalgamation and the various hydrometallurgical processes. The importance of these special processes of gold extraction is increasing; that of the silver processes continues to decrease.

In the metallurgy of gold the cyanide process continues to attract the most attention, the excellent process of barrel chlorination, which for many ores might be superior to the cyanide process, being relegated to a secondary position. The greater interest in the cyanide process is no doubt due largely to the fact that it recovers a portion of the silver contained in many ores, while no recovery of silver is possible with chlorination. The percentage of silver recovered by means of the cyanide process varies within wide limits, this being accounted for by the fact that certain silver minerals are readily soluble in a solution of potassium cyanide, while others are more or less difficultly soluble or quite insoluble. Lately the cyanide process has been applied at a mine in Utah for the treatment of a silver ore which carries only a small amount of gold, the silver occurring in the ore in the form of chloride.

In the treatment of gold ores by the cyanide process a very high degree of efficiency has now been reached, and further improvement is to be looked for in the line of the mechanical apparatus rather than in chemical principles. The latter may be considered well settled, although it is only within a year or two that the necessity for oxygen in the process has become generally recognized, and a determination of the free oxygen in the solution has been a regular part of the work of the analytical staff. In the United States attention has not yet been given strongly to this matter, except perhaps in a few works which are run on more scientific principles than the average. However, in the matter of technical investigations it must be recognized that the South African metallurgists are taking the lead in so far as the cyanide process is concerned. The bromo-cyanide process and other so-called improved cyanide processes, in which an oxidizing agent other than the atmosphere is introduced into the solution, have not yet turned out to be of any practical importance, and it is not expected by unprejudiced metallurgists than they will do so.

With respect to silver milling by pan amalgamation and hyposulphite lixiviation I estimated that the total production by these processes in the United States in 1896 did not exceed 3,300,673 oz. The production in 1898 was still less, owing to the idleness of the large mills at Park City, Utah. The silver mills of the Tintic district, Utah, produced 1,468,500 oz. by pan amalgama-

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* Reference should be made also to the papers on "Progress in Gold Milling," by Robert H. Richards, and the "Hydro-Metallurgy of Silver," by Ottokar Hofmann, elsewhere in this volume; also to the papers on "Recent Improvements in Lead Smelting," by H. O. Hofman, and that on "Present Practice in Silver-Lead Smelting in Denver and Pueblo," by L. S. Austin.

tion, and in Arizona, Montana, and Nevada there was a small production by this method, but I think it doubtful if the total for the United States exceeded 2,500,000 oz.

AMALGAMATION OF GOLD ORES.

Cost of Gold Milling.—The following data are taken from the latest official reports of certain important mining companies:

Alaska-Mexican Mining Co., Douglas Island, Alaska: The quantity of ore mined in 1897 was 158,005 tons, and the same amount was milled, with an average crushing of 3·79 tons per stamp per 24 hours. The cost of milling was 32·17c. per ton. The mill ran 346·5 days, using water power for 217 days and steam for the remainder of the time. There were 2,863 tons of pyrites saved, in addition to which there were 667 tons on hand at the beginning of the year. Of this amount 3,473 tons were treated at the Alaska-Treadwell chlorination works at a charge of $10 per 2,000 lb. The recovery was 93·7%. The total gold obtained was $335,629, of which $226,322 was obtained by amalgamation and the remainder by chlorination. The average yield per ton of ore was $1·4324 by amalgamation and $0·6918 by chlorination, a total of $2·1242. The average tenor of the tailings was 19c. per ton. The cost of mining was 95·32c. per ton; milling and concentrating, 32·17c.; chlorination, 21·06c.; bullion charges, freight, refining, etc., 2·36c.; new construction, 1·27c.—total, including general expenses at Douglas Island, San Francisco, London, and Paris, $1·5729. The mill has 120 stamps, which weigh 1,020 lb. and drop 8 to 8·5 in. 96 times per minute. Miners are paid $2·50 per day with board, white laborers $2 with board, Indian laborers $2 without board, millmen $2·50 to $3 with board, engine drivers and smiths $4 with board.

Alaska-Treadwell Gold Mining Co., Douglas Island, Alaska: According to the report of the superintendent for the year ended May 15, 1898, the reserve of ore in sight above the 220-ft. level on that date amounted to 4,477,500 tons of 2,000 lb. On May 15, 1897, the estimate was 2,960,014 tons. There were mined during the year 254,329 tons of ore at a cost of 59·49c. per ton. This includes the cost of mining, hoisting, and tramming to the dump 7,324 tons of waste rock. The 240-stamp mill crushed 254,329 tons of ore at a cost of 37·08c. per ton. The cost in 1896 was 38·43c. per ton. The mill ran 342 days in 1897. There were used up 500 shoes and 415 dies, while 156 new stems, 16 boss heads, 12 tappets, 30 Blanton cams, and 3 cam shafts were required. The chlorination works treated 4,446 tons of concentrates at a cost of $6·2325 per ton, or 10·89c. per ton of ore milled. In the previous year the averages were $6·3911 and 12·27c. The construction charges, which amounted to 27·45c. per ton against 32·62c. in the previous year, and the bullion charges, which came to 2·47c. per ton, together with the general expenses in San Francisco, London, Paris, etc., brought the total operating expenses up to $1.4776 per ton against $1·5698 in the previous year. The bullion yield ($2·3165) and store profits amounted to $2·4341 per ton. The dividend payments for the year were $300,000, *i.e.*, the same as in 1896-97. The yield of the ore by amalgamation was $1·53 per ton, against $1·87 per ton in the previous year, while 79c. against 93c. per ton were obtained from sulphurets. The total yield per ton in 1897-98 was $2·32, against $2·80 in 1896-97. Up to date this mine has yielded a total of $7,888,868 ($3·11 per ton of ore) at a profit of $4,167,155, the operating expenses having averaged $1·47 per ton. An examination of the yearly figures reported by the company shows that the grade of the Treadwell ore is decreasing steadily. The cost of mining amounted to 38.21c. per ton for labor and 21·29c. for supplies. In the previous year the figures were 35·86c. and 22·82c. respectively. There were used 212,996 lb. of dynamite, 574,370 ft. of fuse, and 108,728 caps. The corresponding figures in 1896-97 were 219,698, 424,918, and 99,373. Thus it will be observed that explosives cost 12·5c. per ton of ore in 1897-98 and 11·43c. per ton in 1896-97, *i.e.*, upward of 50% of the cost of supplies, and about 20% of the total cost of mining in each year. There were used 34,641 lb. of drill steel, which cost 0·95c. per ton of ore, and drill-fittings to the amount of 0·69c. The consumption of candles was 7,600 lb., and of lubricants 3,119 gal. Of kerosene 2,162 gal. were used. The cost of steam power was 3·51c. per ton of ore, against 3·54c. in the previous year. In milling, labor amounted to 15·4c. per ton and supplies to 21.68c., the corresponding figures for the previous year having been 15·4c. and 23·39c. There were consumed 500 battery shoes (80,198 lb.) and 415 dies (58,675 lb.), which cost 3·71c. per ton of ore. In the previous year 517 shoes and 496 dies were used, and the cost was 4·49c. per ton of ore. Quicksilver came to 0·61c. per ton of ore in 1897-98, 39 flasks being used. Steam power cost 8·02c. per ton. Of screens, 376 were used. The chlorination works treated 4,446·6 tons of concentrates for the Treadwell mine and 3,261·5 for the Alaska-Mexican Mining Co. The cost of labor in chlorination was $3·9741, supplies costing $3·8525. The corresponding figures in the previous year were $4·0128 and $3·5919. There were consumed 2,201 cords of wood, at a cost of $1·2136 per ton of concentrates. The consumption of sulphuric acid amounted to 444,000 lb., of oxide of manganese 133,718 lb., of salt 583 short tons, and of scrap iron 63,596 lb. The corresponding figures for the previous year, when 7,000 tons of concentrates were chlorinated, were 359,200, 78,986, 519 and 20,564.

Gold Coin Mines Co., Black Hawk, Colo.: The output of ore in 1897 was 31,797 tons, of which 1,238 tons, averaging $71·81, was shipped directly to smelters, and 30,559 tons, averaging $6·51, was milled. The cost of mining was $5·55 per ton; milling and other charges, $1·36; mint and smelter deductions, $0·90—total, $7·81, leaving a profit of $1·24 per ton. The cost of milling at the Kansas mill (17,704 tons producing 1,485 tons of concentrates) was $0·99. At the Hidden Treasure mill (7,707 tons producing 518 tons of concentrates) it was $1·11. At outside mills (5,148 tons producing 346 tons of concentrates) it was $0·88.

Homestake Mining Co., Deadwood, S. Dak.: During the year ended May 31, 1898, 548,390 tons of ore were treated in the three mills, the yield being $2,467,998 in gold and $15,377 in silver. Concentrates saved from the mill tailings brought $34,651, bringing the general average up to $4·56 per ton. The expenses came to $3·38, of which $2·17 was for mining, $0·77 for milling,

$0·08 for general repairs, $0·14 for general expenses, and $0·22 for purchase of new property. On May 1, 1898, the mine began furnishing ore to 140 additional stamps, making 540 in all.

Montana Mining Co., Ltd., Marysville, Mont.: The report for the half-year ended December 31, 1897, showed a production of 13,004 oz. of gold and 65,763 oz. of silver from 37,290 short tons of ore milled, and 4,255 oz. gold and 29,884 oz. silver from 46,980 tons of tailings treated by the cyanide process. The cost of mining and milling was $6·74 per ton, and the treatment of tailings by the cyanide process, including transportation and amortization of the works, was $1·47.

Water Required for Gold Milling.—A writer in the *Engineering and Mining Journal*, April 16, 1898, states that the result of investigations as to the minimum amount of water required for stamp mills showed that with a very soft ore, which had been ground to pass a 60-mesh screen, 96 gal. of water per ton of ore was sufficient, while with a harder ore only 66 gal. was required.

Weight of Stamps.—There are some mill men who advocate very heavy stamps, and some of 1,400 lb. weight were installed in the Big Cañon mill, in California, not long ago; but after a short trial they were reduced in weight by removing the heavy iron bosses on the upper end of the stems, as it had been found that the jar of such heavy stamps caused frequent breakage of cam shafts. Stamps weighing 1,200 lb., designed to drop 6 in. 100 times per minute, were placed in La Fortuna mill, in Arizona, in 1896. They worked on a hard ore, crushing through a No. 00 screen, and under the existing conditions gave satisfactory results, although the ordinary plates could not take care of the pulp. While there are some mill men on the Pacific Coast who believe in the use of very heavy stamps, there are others who go to the other extreme, advocating very light ones, say 750 lb., dropping 105 times per minute. The most experienced California mill men, however, favor stamps weighing from 750 to 1,000 lb.

Treatment of Amalgam in the Transvaal.—F. L. Carter described the practice at the Crown Deep works, where 8,000 oz. of amalgam are retorted at a time. The inside of the retorts is whitewashed, and semicircular pieces of sheet iron are used to divide the amalgam into convenient blocks, so that the gold can be easily handled. A little niter is mixed with the amalgam, which oxidizes some of the base metal and makes the bullion finer. Copper is the principal impurity. The amalgam contains 33 to 36% gold; in retorting, about 90 to 100 oz. of mercury are lost. In melting the bullion a current of air through a pipe from a compressor is allowed to play on the surface of the molten metal, oxidizing base metals, especially lead, very effectively. In casting the bars some melters skim the gold as much as possible, until the surface looks clear and bright, and then pour it into the mold, throwing on handfuls of sugar as soon as the gold is completely poured, the sugar making a very clean surface. Others add a good deal of salt, and pour with a salt cover. At the Crown Deep the gold is melted with a flux, consisting of 5 parts borax, 2 parts soda, 0·5 part sand, and 1 part niter. The slag, which is about 1 in. thick, is poured with the bullion; by applying a little water it comes off clean from the bar, leaving the gold very bright. After the bar has cooled somewhat it is plunged into nitric acid water, which makes the metal glisten. The Crown Deep bullion assays 850 parts gold, 90 parts silver, 40 parts copper, and 15 parts iron.*

THE CYANIDE PROCESS AND ITS MODIFICATIONS.

Oxygen in the Cyanide Process.—Dr. Loevy, president of the Chemical and Metallurgical Society of South Africa, in his inaugural address, August 22, 1898, remarked that it is now recognized that oxygen is an important element in the solution of gold by cyanide of potassium, and also that the available oxidizing agents, not excepting cyanogen bromide, cannot be applied successfully on a large scale, chiefly for economical reasons. Experience has shown that the only available source of oxygen is the atmospheric air, and that the success of the process depends on the manner in which it is supplied. Bodlaender is of the opinion that Elsner's formula does not actually illustrate the reaction which takes place in the solution of gold by cyanide. He maintains that in the first place peroxide of hydrogen is formed according to the following equation:

$$2Au+4KCy+2H_2O+2O=2AuCyKCy+2KOH+H_2O_2$$

and that the peroxide of hydrogen thus formed acts upon another part of the gold not yet dissolved in the first reaction, according to the equation

$$2Au+4KCy+H_2O_2=2AuCyKCy+2KHO.$$

In Dr. Loevy's opinion this theory is quite rational.

Application of Oxygen.—H. T. Durant advocates the use of a centrifugal pump for handling slimes, especially designed so that the quantity of air drawn in can be regulated in a more satisfactory manner than by trusting to the air sucked in through loose glands and defective packing.† The usual method of supplying the necessary oxygen in South Africa at the present time is to transfer the charge from one vat to another, during which it is aerated.

Estimation of Oxygen.—A. F. Crosse employs a modification of Thresh's method described in Sutton's *Volumetric Analysis.*‡ All cyanides and absorbents of iodine must first be removed, which is done by precipitating them with zinc sulphate, the amount required being determined by titrating a portion of the solution under examination with a standard zinc sulphate solution, using phenolphthalein as an indicator, the zinc sulphate solution being run into the cyanide solution until the magenta color of the latter is just destroyed. The proper amount of zinc sulphate having been added to the cyanide solution under examination, and the zinc cyanide having been allowed to settle, the clear liquor is siphoned off without undue access of air, the end of the immersed limb of the siphon being covered with a small bag of lint to filter off any floating particles of precipitate. Two or three pipettes (300 c.c.) full of the siphoned solution are retained. A preliminary test of the iodine-absorbing power of the solution due to unprecipitated double cyanide is made

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* *Engineering and Mining Journal*, Nov. 12, 1898.

† *Journal Chemical and Metallurgical Society of South Africa*, 1898, I., pp. 59-62.

‡ *Idem*, I., pp. 107-112, pp. 125-127; *Journal Society Chemical Industry*, Nov. 30 and Dec. 31, 1898.

by adding to a quantity equal to that used in the test 0·9 c.c. of dilute sulphuric acid (1:1) and a few drops of potassium iodide and starch. Dilute bromine water (1 Br : $2H_2O$) is added until a blue color is obtained. Another pipetteful of the liquid is then taken, 0·9 c.c. of sulphuric acid, and the required amount of bromine water found by the preliminary experiment are added, the stopper is put into the wide-mouthed bottle used in Thresh's test, and the pipette is turned over several times; 1 c.c. of the potassium iodide and sodium nitrite solution is then added, and the free iodine, freed in proportion to the oxygen in the solution, is determined by means of standard sodium thiosulphate. The following reactions occur in the pipette, and after the addition of the thiosulphate and starch solutions:

I. $H_2SO_4+KNO_2=KHSO_4+HNO_2$. *II.* $2HNO_2+2HI=2H_2O+I_2+N_2O_2$.
III. $N_2O_2+H_2O+O=2HNO_2$. *IV.* $HCy+I_2=ICy+HI$.
V. $2Na_2S_2O_3+I_2=2NaI+Na_2S_4O_6$.
VI. $HI+ICy+2Na_2S_2O_3=HCy+2NaI+Na_2S_4O_6$.

A correction has to be made for the small quantities of nitrites in the cyanide solution. To make this correction pour into it, in a very strong 350-c.c. flask, a quantity of solution equal to that used in the experiment (say 293 c.c.), add a few drops of potassium hydroxide, and close the flask with a rubber stopper having one perforation, through which is passed a glass tube with a glass stopcock. Boil the solution for a few minutes and close the stopcock. Cool the flask, and when cold pour the liquid into the pipette, and add 1 c.c. of iodide and nitrite solution and 1 c.c. of sulphuric acid (1:1). Then let it stand for 10 minutes, and, in the presence of coal gas, run it into the wide-mouth bottle employed as above, add starch, and titrate with thiosulphate. The quantity required gives the correction for nitrites and for the reagents, as the same amount of acid and of iodide and nitrite solution is used in each case.

A mill solution pumped to a leaching vat showed 6·3 mg. oxygen per liter; the same solution run from the vat 30 hours later had only 0·6 mg., and at the end of the zinc boxes showed only 0·3 mg.

Influence of Temperature in Cyanide Lixiviation.—Dr. Loevy expresses the opinion that the temperature of the solution plays an important part in the cyanide process and that the subject deserves more attention than has hitherto been given to it. He refers to an experiment in the treatment of a pyritic ore assaying about 4 oz. gold per ton with solution at 40° C., when the extraction after four hours leaching was 14% higher than was obtained by leaching the same ore with a solution of the same strength for 50 hours in the cold. He states it as a fact that a temperature of 35° to 40° C. accelerates and increases the solution of gold to the extent that an additional extraction of $1 per ton may be obtained from tailings assaying $6.

It is by no means unknown to cyanide metallurgists that the dissolving power of the solution decreases with the temperature, and in cyanide works designed for operation in the winter proper measures have to be taken to maintain the temperature of the tank room and the solutions, not only to preserve the maximum dissolving power of the latter, but also to guard against the danger of freezing in the pipes, etc. Recent experiments by H. G. Heffron at the Sacramento mill at Mercur, Utah, showed that the gold of the ore treated there was most readily dissolved at 95° F., while below 80° F. there was considerable falling off in the solvent power of the solution; above 95° F. there was no gain.

Conveyors for Charging Leaching Vats.—The Jeffrey Co. has designed a conveyor for charging pulp into leaching vats whereby it is claimed a considerable saving of labor can be effected. The system, which is a modification of the trough conveyor, is used in the Arequa mill at Elkton, Colo. However, the opportunity for saving in this direction is not generally large. If the distance be not too great two men with a car holding 1 ton of pulp and a properly laid track can charge 10 tons per hour into a 20-ft. vat, including weighing and distributing the pulp. At the wages customary in the West this amounts to only 5c. per ton.

Direct Treatment of Witwatersrand Ore.—Franklin White described the practice at the Luipaards Vlei Estate Co.'s mine, where the ore was crushed dry with rolls, the results showing that dry crushing and direct treatment may in some cases give better results than a wet stamp mill. The ore was the ordinary Witwatersrand pyritic banket. In crushing it from 1 to 2·5 lb. lime per ton were mixed in. The leaching was done with a 0·25% strong solution followed by two 0·15% solutions. The solutions were not allowed to stand, but were drained off as soon as the tank was filled, the time of leaching being 66 to 70 hours. Precipitation was effected by zinc in the ordinary manner. In the treatment of 25,123 tons (2,000 lb.) of ore assaying 8·28 dwt. per ton there was an actual extraction of about 82%. The cost of crushing was 4s. 1·6d. per ton, and cyaniding 3s. 5·86d., the consumption of cyanide being 0·40 lb. per ton, which cost 7·7d.*

The Direct Filling Process for Tailings.—F. O. Pengilly states that the treatment of tailings by the direct filling process is now practiced successfully at two works on the Witwatersrand. The tailings from the battery plates are lifted by a wheel into a launder, by which they are conveyed to the slime separators, of which there are two. The slimes from the first separator pass into the second, thus saving fine sand. Fifteen per cent. of the slime is thereby eliminated, and 10% more is washed out when running the remaining pulp into the treatment vats, thus producing tailings fairly free from slimes. These tailings are then drained, treated with an alkali solution to neutralize the free acid from pyritic ore, and subjected to 0·6 lb. of cyanide per ton of tailings in solutions of different strengths, this treatment lasting over a period of six days. The gold is precipitated by zinc, preferably as zinc-lead couples containing 1 to 2% of lead, to overcome the action of foreign metals in retarding the precipitation of the gold, which couples are made by dipping the shavings into weak acetate of lead solution. The precipitate, containing 4% of gold when dry, is agitated with a 10% solution of sulphuric acid, which dissolves the zinc and increases the gold contents from 33 to 45%. The bullion

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* *Transactions Institution of Mining and Metallurgy*, Jan. 18, 1899.

obtained yields an average fineness of ·847 gold and 60 silver. The chief advantages of the direct filling process are the smaller plant necessary and the saving in working cost.

Coagulation of Slimes.—W. A. Caldecott states that perfectly clear lime water will instantly cause flocculation of slime pulp, but in the case of weathered ores yielding ferric hydrate with the lime the rate of settling is less rapid than in slimes from new ores.*

Slimes Treatment in Montana.—M. W. Alderson describes† the practice at Schmidt & John's mill, at Silver Star, where the slimy material is shoveled into an agitator 16 ft. in diameter and 8 ft. deep, with a vertical shaft bearing two blades, driven at 18 r. p. m. The usual charge of ordinary tailings is about 20 tons; of very compact slimy material about 12 tons. The percentage of solution to ore varies from 1 : 1 to 2·5 : 1, depending on the character of the material and the proportion of solution which can be successfully decanted. The solution is used of such strength as to test 1 lb. cyanide per ton during agitation. Agitation is continued for 12 hours. The mud is then allowed to settle, after which the clear liquor is drawn off. A second solution is then run in, and after this has been drawn off a third lot is run in. In drawing off the last it is run to a separate vat to be used as second solution for a new charge. Precipitation is done successfully with any solution carrying as much as 0.5 lb. cyanide per ton.

Slimes Treatment in Victoria.—According to W. B. Gray‡ the treatment of slimes is accomplished successfully at the South German mill at Maldon, Victoria, by means of Deeble's patent agitator. The agitation vats, which are 18 ft. in diameter and 4 ft. deep, are first filled with 2 ft. of sump solution, and the agitator then set in motion, after which slimes are charged to the amount of 15 tons, which brings the level of the liquor to within 3 in. of the top of the vat. A charge of 80 lb. of lime is then added, partly as an alkaline agent, but chiefly to help in the subsequent settlement of the slimes. A proper quantity of cyanide is then put into the vats to bring the solution up to the strength of about 0·15% and the agitation is continued for at least 30 hours, at the end of which time the agitator is stopped and raised out of the body of the slime, which is allowed to settle. This takes from 8 to 16 hours, according to fineness. As the settling proceeds, the clear supernatant liquor is drawn off gradually, by depressing the outlet gate, and run into a filter vat where any fine slimes which may have escaped are arrested. The drawing off of the first liquor having been completed, the gate is raised, the agitator set in motion, and the first sump solution wash is run on. Agitation is continued 30 minutes and the first wash is then drawn off. A similar process is followed with a second sump solution and a final water wash, the quantity of the latter being gauged so as to keep the total quantity of solution in circulation constant. Finally the slimes are sluiced out.

Filter Press Treatment of Slimes.—This process has been developed in Western Australia largely through the scarcity of water, which excluded the adoption

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* *Journal Chemical and Metallurgical Society of South Africa*, 1898, I., p. 96.

† *Engineering and Mining Journal*, 1898.

‡ *Transactions Australasian Institute of Mining Engineers*, Vol. V.

of the South African practice of coagulation with lime and washing by decantation, since by the latter method the discharged slimes still contain 50% of their weight of weak cyanide solution, besides which there is the increased loss of water by evaporation from the larger quantity in circulation. At Hannan's Brownhill mine, at Boulder, West Australia, the ore is crushed dry and divided by a pneumatic separator into sands and slimes. The former are leached in vats in the ordinary way; the latter are agitated with a 0·3% cyanide solution for six hours in a vat, after which the thin mud is forced into a filter press and squeezed, whereby the gold-bearing solution is separated, the cakes being washed by subsequently forcing water through the press. The whole operation of filling the press, leaching, and emptying occupies from 1·5 to 2 hours. At the Lake View Consols mill the wet tailings from the battery are partially settled, and the slimy liquor is treated in a set of 10 filter presses, which give an 85% extraction with $10 slimes.*

At Hannan's Brownhill mill the mud from the agitators is forced into the filters at a pressure of 60 to 80 lb. per sq. in. by means of compressed air. The filtrate flows in a constant stream through the zinc precipitation boxes to the sumps. The cakes having been washed, the press is opened and the cakes are allowed to drop into a truck standing under the press. The filter presses are of the distance frame, flat-plate and four-eyed type, each press having 20 chambers, which form cakes 28 in. square by 3 in. thick. The pressed cakes carry about 20% moisture and weigh about 130 lb. per cu. ft., from which figures it appears that each press treats about 1·25 tons of dry slimes.

Experiments carried out on 3-in. cakes show that the washing is very complete and that uneven percolation does not take place. Filling the press from the receiver (in which the mud is accumulated from the agitators) occupies 15 minutes, washing the cakes in the press requires 19 minutes, and discharging and closing the press 16 minutes, a total of 50 minutes. The amount of solution required in the agitators to reduce the pulp to such consistency that it can be readily forced into the filter presses is about 1·5 times the weight of the slimes. The amount of wash water required is very small, but no figures as to this are given in the paper from which this account is taken.† In the treatment of slimes by the decantation process in South Africa from 8 to 10 tons of liquid are required for one ton of slimes. According to H. C. Hoover the slimes treatment by filter press can be done at $1·92 to $2·40 per ton. The report of the Lake View Consols Co. for 1898 showed that its filter press slimes plant had treated 7,423 tons of slimes, yielding 2,794 oz. gold at an average cost of 10s. 8·4d. per ton.

Recovery of Gold from Cyanide Solutions.—Sherard Cowper-Coles proposes to substitute aluminum for lead as cathode material, advantage being taken of the fact that a loose film of oxide is very rapidly formed on aluminum. The gold is obtainable as pure metal, and daily returns can be made of the amount precipitated, while there is also economy in labor. The deposition of gold on an aluminum plate from cyanide solution proceeds in a uniform manner, in

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* Edward S. Simpson, *Transactions American Institute of Mining Engineers*, February, 1898.

† William McNeil, "Filter Press Treatment of Gold Ore Slimes at Hannan's, W. A.," *Transactions Institution of Mining and Metallurgy*, Vol. VI., 1897-98, p. 247, *et seq.*

such a way that a metallic sheet is formed which is easily detachable almost as soon as formed. Aluminum cathodes can be made in the form of revolving cylinders from which the deposited gold can be removed automatically and continuously. Gold has been successfully extracted in this way from solutions containing only 0·01% potassium cyanide and 2·5 dwt. gold per ton, the best results being obtained at a temperature of about 100° F. It is also advantageous to use a greater current density with the necessary increased voltage when the aluminum plates have been freshly placed in the solution, so as to insure their being covered with a film of gold as quickly as possible, since otherwise there is a tendency to the formation of aluminum hydrate. In about 10 hours an extraction of 95% was obtained from a solution of the above-mentioned strength, with a rate of flow of about 15 gal. per 100 hours for every cubic foot of electrolyzing cell, or 3 sq. ft. of cathode surface.*

E. Andreoli does not have a favorable opinion of the Cowper-Coles process. His judgment may be influenced, however, by the fact that he also has a process of his own for the precipitation of gold from cyanide solution. Andreoli thinks that the use of iron plates at the positive pole will contaminate the gold-potassium cyanide solution, while the ease with which the gold can be stripped from the aluminum electrodes is a disadvantage, owing to the facility with which the product may be stolen. The iron anodes used in the Siemens-Halske process become dissolved and contaminate the electrolyte with ferric oxide and prussian blue. With the lead peroxide anodes used in the Andreoli process the solution is always clear.†

The Lead-Zinc Process of Precipitation.—T. L. Carter describes‡ the process introduced by himself and W. K. Betty at the Crown Deep slimes plant at Johannesburg. This plant was equipped originally with the Siemens-Halske system, which has now been displaced by the new process. The latter makes use of a zinc-box 40 ft. long, 6 ft. broad, and 5 ft. deep (of which one-half is claimed to do the work of six S.-H. boxes), divided into nine compartments. The gold-bearing solution is permitted to flow through this box at the rate of 15 tons per hour, but before entering it must be clarified as much as possible from all matter in suspension. The box is filled with zinc shavings prepared by immersing them in a solution of lead acetate of about 10% strength, in which they are thoroughly washed and stirred until they acquire a dark color. With this preliminary preparation the zinc must be placed in the box and covered with the gold-bearing solution as quickly as possible, since exposure to the air appears to have a bad effect. An essential part of the process is the addition of fresh cyanide at the head of the box, where in beginning the precipitation a 2·5% solution is allowed to run freely into the gold-bearing solution for a period of about four hours, raising the strength of the solution passing through from 0·007 to 0·025%. When 20 lb. of cyanide have been added in this manner 10 lb. more are run into the box, taking 6 hours, and raising the solution about 0·007% higher. Between 12 and 14 hours after starting a slow drip is allowed to fall into the solution as it enters the box, raising the

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* *Transactions Institution of Mining and Metallurgy*, 1898.

† *Engineering and Mining Journal*, June 18, 1898, p. 733.

‡ *Journal Chemical and Metallurgical Society of South Africa*, October, 1898.

strength of that going through from 0·007 to 0·008%. This drip is continued through the run of the box. During the first four or five days precipitation is very active, but gradually the zinc in the upper compartments loses its power, and after 10 or 11 days becomes dead and rises to the surface, precipitation going on actively in the meanwhile in the subsequent compartments. Cleaning up the box run in this manner causes considerable trouble when the dead zinc has to be dissolved. A way out of this difficulty has been found, however, in running the zinc by the new system for two or three days and then transferring it to the sand plant, where it is serviceable with the stronger solutions employed in the regular manner, while its place in the slimes zinc-box is filled with fresh shavings prepared in the manner described. Owing to the lead which is introduced in the form of lead acetate, the auriferous zinc slimes contain a high percentage of this metal, one sample assayed by Mr. Carter showing 23% lead; but by the use of a flux consisting of 60% borax, 19% niter, 11·5% sand, and 7% soda, and the employment of clay liners, he succeeded in producing a bullion of ·876 fineness.

The use of zinc coated with lead by immersion in a solution of lead acetate is by no means new. It has been patented by J. S. MacArthur,* and his invention may have been anticipated by others. W. Bettel in 1894 made experiments on this line which, according to him, was pointed out by Dr. Gladstone and Mr. Tribe. According to James Macintyre a similar process has been in successful use for five years by the Lisbon-Berlyn Co. in the Lydenburg district, where it was introduced by J. S. MacArthur. Betty's claim is for the combination of free cyanide, lead, and zinc.

According to Mr. Carter† the cost of treatment of 10,317 tons of slimes by the Siemens-Halske process in April, May and June, 1898, was 6·96d. per ton for cyanide, 3·84d. for lead foil (less litharge), 0·95d. for power and 1·18d. for royalty, making a total of 1s. 0·95d. per ton. The cost of treating 9,295 tons by the new process in October and November was 4·05d. per ton for cyanide, 0·66d. for zinc (including cutting), 0·61d. for sulphuric acid, 0·31d. for power and 0·22 for lead acetate, making a total of 5·85d. per ton. The outflow from the zinc-box assays about 2 grains gold per ton. Mr. James Macintyre confirmed Mr. Carter's statements as to the completeness of the precipitation, from his own experience at the Lisbon-Berlyn mill.

Precipitation by Charcoal.—W. B. Gray describes a process practiced in Australia. The first solution and all washes are passed through charcoal filters, for which the charcoal is prepared by a Dunn's charcoal crusher and afterward cleansed by washing with water. There are 40 filters, which number is required for the treatment of about 400 tons of tailings per week. They are arranged in eight rows, five in a row, the flow of liquor being distributed equally per row. Each filter has a capacity of about 560 lb. of charcoal. Experiments have shown that the first filter precipitates 45% of the gold, the second 25%, the third 15%, the fourth 9%, and the fifth 5%. Thus 99% of the gold is recovered, while the

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* United States Patent No. 592,153, Oct. 19, 1897; THE MINERAL INDUSTRY, Vol. VI., p. 342. His first patent on this process was issued in 1894.

† *Journal Chemical and Metallurgical Society of South Africa*, December, 1898.

average assay of the outflow is only 2 grains per ton. These results are with a flow of about 200 gal. per hour. Mr. Gray considers that precipitation by charcoal is cheaper and better than by zinc. The cost of charcoal is about 2s. per filter, or 2·75d. per oz. of gold saved. Under the most favorable circumstances 1 lb. of zinc at 6d. per lb. will be consumed for each ounce of gold, while other expenses of recovery are greater than with charcoal. The gold from charcoal precipitation is of greater fineness than from zinc. Charcoal as a precipitant also has the advantages that it will throw down the gold from the weakest solution of cyanide and that it does not contaminate the solution.*

This is an application of the Johnson process.† It is analogous in all respects to the carbon precipitation of gold from solution in chlorine water, which has been employed on a large scale at several works, but has generally been abandoned in favor of precipitation with hydrogen sulphide. The carbon carrying gold constitutes a very bulky mass, troublesome to handle in recovering the gold, which is done by burning off the carbon in cast iron pans wherein there is considerable danger of loss.

Efficiency of the Siemens-Halske Process.—According to S. J. Jennings the Crown Reef slimes plant, Witwatersrand, from July, 1896, to October, 1898, treated 82,542 tons of slimes which according to assay contained 18,408.217 oz. of gold, while the residues contained 4,675.735 oz., giving a theoretical extraction of 74·6%. The fine gold actually recovered from the lead bullion was 7,503·447 oz., and from the by-products 2,508,679 oz., equivalent to an actual extraction of 54·4%. In order to explain the discrepancy, the furnace in which the lead was melted was torn down and ground up in a ball mill, the sides of the precipitation boxes were planed and the shavings burned, the anode sacking was removed and burned, but still the difference of 20% could not be explained. Every precaution, supported by ample means, was taken to check the tonnage, which is usually one of the main sources of error. Mr. Jennings came to the conclusion that there are too many ways in which gold can escape in this process to make it a permanently satisfactory one. The value received for the above product was £40,319 4s. 1d. Up to March 31, 1897, 3% was paid as royalty, and after that date 2%. If the 10,012 oz. had been recovered in the form of zinc bullion the company would have received £2,644 more for it, or 932 oz. less gold might have been recovered and the company would still have derived the same amount of money, £40,319. The amount of gold derived from by-products was 25% of the total recovery.

At the Ferreira, according to Mr. Hall, the slime plant (S.-H. process) gave a theoretical extraction of 80·18% and an actual extraction of 76·09%, the gold realizing 84s. 6d. per oz. There are four boxes, each 30×12 ft. and holding one ton of lead. The consumption of lead is 1 lb. per ton of slime. The flow through the boxes is from 15 to 20 tons an hour, the solution entering at 1 dwt. 5 gr. and leaving at 4 gr. per ton. By-products are treated in a pan furnace and cupellation furnace at a cost of about 2·5d. per oz. of gold. According to P. S. Tavener the recovery by the S.-H. process at the Bonanza plant for

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* "Treatment of Slimes," *Transactions Australasian Institute of Mining Engineers*, Vol. V., pp. 138-142.

† THE MINERAL INDUSTRY, Vol. IV., p. 342.

the 12 months ending April 30, 1898, was 82·72% theoretically and 81·15% actually; in the seven months ending November 30, 1898, the theoretical recovery was 85·9 and the actual 84·59%.

There appears to be a big discrepancy as to the results of the Siemens-Halske process. This is probably to be looked for in errors in the method of weighing or computing weights and sampling, perhaps in assaying, and probably to some extent in each particular. W. A. Caldecott has found that in assaying slimes containing soluble gold the full amount of gold is not obtained, one test of a sample containing 25·5 gr. per ton yielding only 15 gr. This discrepancy is accounted for by the hypothesis that the soluble gold in the dry slimes exists in such a minute state of division that even the lead globules from the litharge fail to collect it.

Cyanide Lixiviation and Pan Amalgamation.—S. Newhouse, A. J. Bettles and T. Weir patented a process wherein the ore is treated with weak cyanide solution in a pan, the gold precipitated from the solution, and the precipitate amalgamated together with any gold which is too coarse to be dissolved. It is claimed that the cost of the process is about the same as in pan amalgamation, and that it is especially advantageous for the treatment of clayey ores.[*]

Alfred James describes a process devised by Messrs. Gilmour and Young and employed for the treatment of the ore of the Santa Francisca Gold Mining Co. in Nicaragua.[†] The ore contains a large percentage of clay, which makes leaching very difficult. It is crushed to pass a 30-mesh screen by means of a ball mill, and the slime separated from the sand by pneumatic sorting. The slime is then charged into a 5-ft. Boss pan in 2-ton lots with 100 gal. of water, so as to form a very thick pulp. Mercury is then added until the globules can be seen circulating in the pulp, and then the required amount of potassium cyanide. After two hours' running about 10 lb. of mixed zinc and copper amalgam are added and the pan run four hours longer. This precipitates the gold very completely, only 10 grains ordinarily remaining in solution, and this has even been reduced to 1 grain per ton of ore. The contents of the pan are then discharged into a 7·5-ft. settler and the mercury recovered in the usual manner. About one hour is sufficient for the solution of the gold in the slimes, while the sand requires four days' percolation. The slimes show an extraction of over 90% and the sands without retreatment upward of 75%, or an actual bullion recovery of 84% on the whole ore. The slimes assay 1·5 oz. gold and 4·5 oz. silver per ton. About 70% of the silver is recovered. In the treatment of 4,129 tons of ore assaying 1·275 oz. gold and 2·379 oz. silver per ton, during the 12 months ending with June, 1898 an extraction of 83·6% of the gold and 63·9% of the silver was effected. The zinc amalgam, which is made by pouring mercury into molten zinc, should be kept under water. The consumption of cyanide is less than 0·5 lb. per ton of ore. With rich ores the consumption of chemicals per ton of ore is as follows: cyanide 1·67 lb., caustic soda 0·63, copper sulphate 1·16, cast iron turnings 0·25, zinc 0·39, mercury 0·40. In

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[*] United States Patent No. 601,201, March 22, 1898.

[†] "Notes on a Process for Treating Slimes without Filtration or Decantation," *Transactions of Institution of Mining and Metallurgy*, Nov. 16, 1898.

Nicaragua these come to 3s. 0·8d.; labor comes to 3s. per ton. For 50 tons of ore per day 24 h. p. are required. The works have four pans and two settlers.

The Pelatan-Clerici Process in Washington.—According to Patrick Clark, president of the Republic Mining and Milling Co., of Republic, Stevens County, Wash., the Pelatan-Clerici 10-ton plant was so successful that a 35-ton plant has been installed. Considerable trouble was experienced at first in pulverizing the ore to the proper fineness, but this was overcome by the barrel pulverizer, manufactured by Fraser & Chalmers. The ore is crushed to pass a 120-mesh screen, this being necessary on account of the fine dissemination of the gold. Crushing and pulverizing cost $1.50 a ton. The total cost of treatment is $6.50 a ton. Of the gold and silver 90% is saved. The ore averages $66 gold and $1.50 silver per ton. Amalgamation yielded only 40% of the value of this ore. The pulp, which is mixed with the cyanide solution in the ratio 1:1, is allowed to remain with it 20 hours.

*Cyanide Practice at Mercur, Utah.**—In some cases the ore gives satisfactory results in leaching at 0·75 in. size, but generally 0·25 to 0·5 in. gives the best results. The gold is so fine that it goes into solution very quickly, the time of leaching being from 48 to 72 hours. The strength of the cyanide solution varies from 0·1% to 0·2%. Solutions of 0·1% have given good extraction, but the precipitation was imperfect until the solution was restrengthened. The consumption of cyanide is as low as 0·75 lb. per ton; of zinc from 0·25 to 0·375 lb. per ton of ore is used. The consumption of both zinc and cyanide is governed to some extent by the judicious use of lime before leaching. With $4 ore an extraction of 75 to 80% is realized; with $6 ore 80 to 86%. Mining costs from $1 to $1·50 per ton; milling from 75c. to $1. Several of the companies are mining and milling for less than $2 per ton.

While the surface or oxidized ores of Mercur presented few difficulties, the unoxidized ores are less docile, containing native sulphur, realgar, orpiment and ferrous arsenide, all of which affect the solution. The alkaline cyanide solution acting on the sulphur and sulphides of arsenic produces a soluble alkaline sulphide, in the presence of which no gold can be dissolved, the alkaline sulphide using up the oxygen which is necessary for the solution of the gold. Some of the Mercur ore contains a carbonaceous shale, which is objectionable, its carbon being a precipitant of gold. The idea that the arsenic in this ore is the objectionable element is erroneous, the sulphur in combination with the arsenic and the carbon in the shale being the real causes of trouble. Ore with over 3·5% of arsenic, existing in the minerals scorodite and haidingerite, has been treated successfully. In fact, arsenic is always present in the cyanide solution in the Mercur district, and is precipitated to a certain extent with the gold on the zinc. The sulphide in solution accounts for the inefficiency of the solution when applied to a thoroughly oxidized ore even if fresh cyanide has been added. In order to overcome this difficulty a preliminary roasting of the ore is required. Certain oxidized ores in the Mercur district are difficult to treat on acount of their talcose and argillaceous character. They contain hydrous silicate of magnesium, hydrous silicate of

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* William Orr, *Proceedings International Mining Congress*, Salt Lake City, Utah, July 6-9, 1898, pp. 113-117.

aluminum, etc. After dehydration by roasting they leach beautifully. It is only necessary to heat sufficiently to expel the combined water.

Montana.—At Schmidt & Johns' mill at Silver Star, according to M. W. Alderson, the strong solution contains only 1 lb. cyanide per ton, *i.e.*, 0·05%. At the large tailings plant at Marysville the standard is 4 lb. per ton, but at most of the other mills in Montana from 5 to 10 lb. per ton is used. At the Silver Star mill solutions before entering the zinc boxes show only 0·7 to 0·9 lb. cyanide, but the gold is precipitated successfully.

India.—Laurence Pitblado stated* that in the Kolar field there are at present six cyanide works in operation. The ores of the field are very simple, consisting mainly of pure quartz, with only a small percentage of pyrites. The material treated is tailings from the stamp mills. Those first worked in the Mysore plant averaged 4·5 dwt. gold and yielded 65%, with an average consumption of 1 lb. of cyanide per ton. In 1897 a month's test with 40-mesh screens in the batteries gave the following result: 90·65% amalgamated in batteries and on plates; 74% of the value in the tailings recovered by cyanide lixiviation, making a total extraction of 97% of the ore as delivered to the mill. The cost of cyaniding in the present small plant of the Mysore company is 2s. 10·5d. per ton, exclusive of royalty and depreciation, but in the 4,000-ton plant which is being erected alongside the heap of accumulated tailings that is to be worked, it is believed that the cost will not greatly exceed 2s. At the Champion Reefs mill, where 20-mesh screens are used in the batteries, the extraction from the tailings is about 56%, with a consumption of 1 lb. cyanide.

In refining the precipitate in this district it is first passed through a 30-mesh screen, drained, dried, roasted with or without a small percentage of niter, and fused directly in plumbago crucibles. At the Mysore works the precipitate is retorted before roasting, yielding about 100 lb. of mercury per month. The presence of mercury in the zinc boxes generally leads to the production of much floured and brittle zinc. In fluxing and smelting directly the retorted slimes the bullion assayed 56·4% gold, 3% silver, 2·4% lead, 19·6% copper, 18% zinc, and 0·1% nickel. The slag contained a good many shots of metal. In roasting with niter a slag freer from prillions was produced, and a bullion assaying 49·5% gold, 4·6% silver, 4·29% lead, 38·21% copper, 2·10% zinc, and 0·22% nickel. These results being unsatisfactory, the following practice was adopted at the Mysore works: The retorted and dried slimes are mixed with 10% niter and roasted at a bright red heat. When cold they are boiled with dilute sulphuric acid (1:2), which dissolves the copper. The dried and washed product is fluxed with about 35% borax, 15% soda, and 10% sand, giving a slag free from shots of metal and a bullion assaying 81·3% gold, 6·9% silver, 2·71% lead, 6·78% copper, 0·4% zinc, and 0·12% nickel.

Western Australia.—According to H. C. Hoover the cost of roasting telluride ores on a large scale does not exceed $2·40 per ton.† The average cost of cyanide treatment of sand at four mines is $1·26 per ton. The report of the Lake

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* *Journal Society of Chemical Industry*, Feb. 28, 1898.

† *Engineering and Mining Journal*, Dec. 17, 1898.

View Consols Co. for 1898 showed that 36,182 tons of tailings, yielding 29,207 oz. of gold, had been treated at an average cost of 6s. 7·6d. per ton.

New Zealand.—The Waihi Gold Mining Co., of the Upper Thames district, has a hard white quartz ore, which has to be crushed to pass a 40-mesh sieve in order to give a good extraction, but with proper treatment from 90 to 92% of the gold is obtained. Roasting in kilns with wood costs 51·6c. per 2,240 lb., crushing in rock breakers 18·4c., stamping 75·3c., cyanide and zinc 65.4c., royalty and depreciation 11·6c., power, labor, and sundry expenses in lixiviation 23·2c., assaying and melting 18·2c.

The government regulations in connection with the use of the cyanide process, the rights of which were recently purchased for £10,000 from the Cassel Co. by the government, prescribe a royalty of 1% when the value of the ore does not exceed £2 per ton; 1·5% when the value exceeds £2 but does not exceed £3; 2% when the value exceeds £3 and does not exceed £4; and 2·5% when the value exceeds £4 per ton.

Golden Gate Cyanide Works, Mercur, Utah.—These works, which were constructed in 1898, are built on a side hill with eight levels. In order to get the ore to the top of the works it has to be hoisted on an incline 800 ft. long. The mill is 294 ft. wide, and 420 ft. in length up and down the slope. The difference in elevation from top to bottom is 145 ft. The retaining walls, which are 2 ft. wide at the top and have a batter of 1 ft. in 12, required over 5,000 cu. yards of rubble masonry. The mill is driven by power transmitted electrically a distance of 35 miles at a tension of 40,000 volts. The various floors were constructed by blasting out the side hill. The broken stone thus obtained was used for the retaining walls and filling behind them. The loss of energy in transmission is said to be only 5%. At the works the 40,000-volt 3-phase current is transformed to one of 220 volts of 2-phase. The current is delivered at a contract price of $60 per h. p. per year. The first section of the mill contains the coarse crushers, and in the second are the dryers. In the third section is the fine crushing machinery, which consists of four sets of 26-in. rolls and three sets of 36-in. Berthelot apparatus is used for sizing. There are six elevators, with a lift of 60 ft. The fourth, fifth, and sixth sections contain the roasting furnaces, which are of Brown's straight-line design, four in number. Those intended for arsenical ores are estimated to have a daily capacity of 75 tons, while those for talcose ores are rated at 150 tons. The ore is stirred by the rabbles once each minute. One man attends to two furnaces. The gases are carried from the furnaces through 6×8 ft. flues into the main dust chamber, which connects with a steel chimney 8 ft. in diameter and 85 ft. high, located on the hill above the buildings. The top of this chimney is 275 ft. above the lowest level of the building. The leaching department, which constitutes section 7, is 60×294 ft. It has two floors, the main floor supporting 10 rectangular tanks 25×50×5 ft., and three solution tanks 20 ft. in diameter and 12 ft. deep. The tanks are supported by masonry piers. They are charged by hand from cars run on bridges over the tanks. The eighth section of the mill, which is 50×70 ft. and two stories in height, is for the precipitation department. The tailings from the leaching tanks are discharged into cars, which are trammed to the waste dumps. The building is constructed of steel.

Drumlummon Cyanide Works, Marysville, Mont.—The works of the Montana Mining Co., Ltd., erected in 1897 for the treatment of an accumulation of old tailings in Silver Gulch, about six miles below Marysville, have a capacity of 400 tons per day. The tailings, brought to the works by a tramway line, are charged into the vats by a revolving chute or distributor. The vats are 38 ft. in diameter and 9 ft. deep, holding 400 tons of tailings. The filling requires about eight hours. There are four of these vats, and the treatment of each charge occupies about four days. The tailings are discharged by sluicing with two 2·5-in. hose, the water being under a 60-ft. head, through four side-discharge gates and one bottom-discharge opening in the center of the vat. The discharging of a tankful of tailings occupies three hours or less, and costs less than 2c. per ton. There are four settling tanks, 22 ft. in diameter, and 14·75 ft. deep, two storage tanks 38 × 9 ft., and two water tanks 22 × 14·75 ft. All the tanks are made of California redwood. The works are not covered, and consequently are operated only in summer.

Henry Nourse Cyanide Works, Witwatersrand.—These works are the largest and latest in the Transvaal. The tailings wheel is 60 ft. in diameter. Having been elevated by the wheel, the tailings are classified by spitzluten, the concentrates and coarse sands going to vats reserved for them, while the remaining sands are separated from the slimes by spitzkasten, the slimes going at once to the settling pond and the sand being distributed through 3·5-in. hose to the vats. There are 12 vats, arranged on the double-treatment system, each having a capacity of 400 tons. The six upper vats are 40 ft. in diameter and 8 ft. deep; the six lower, 37 × 10 ft. They are made of 0·25-in. tank steel, and are supported by steel girders and tubular steel pillars, the latter instead of the usual masonry piers. An asphalt floor is laid under the vats and around the tailings wheel to collect leakages. Each vat has seven bottom-discharge doors. It is expected that the spitzluten will yield 25% of the tailings as concentrates with $15 gold per ton, and four vats are intended for the treatment of this product. The remainder of the tailings, minus a large percentage of the slimes, will be subjected to six days' treatment in the upper tanks, after which the charges will be dropped into the lower tanks and treated for six days with stronger solution. The old cyanide plant is being adapted for slimes treatment.*

Analysis of Cyanide Mill Solutions.—W. J. Sharwood reported the analyses of various cyanide mill solutions.† The methods employed in making the determinations were as follows: Free cyanide was estimated by silver nitrate, using a few drops of 5% ferrocyanide solution as indicator. Total cyanogen was obtained by continuing the titration with silver after addition of caustic soda and a little ammonia and potassium iodide; this, however, does not include cyanogen in double cyanides of copper, silver, gold or mercury. Calcium was estimated by direct precipitation of 100 c.c. of the solution with ammonium oxalate, after addition of ammonium chloride and some excess of ammonia, the washed

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* *Engineering and Mining Journal*, Feb. 25, 1899.

† *Ibid.*, Aug. 20, 1898, p. 216.

precipitate being dissolved in hot dilute sulphuric acid and titrated with permanganate. For iron, copper and zinc 100 c.c. of the solution was twice evaporated with nitric acid and redissolved in dilute sulphuric. The iron was precipitated by ammonia in excess and at once redissolved in hydrochloric acid and estimated colorimetrically as thiocyanate, unless the quantity sufficed to allow of reduction by zinc and titration by permanganate. Copper was approximately estimated by the color of the ammoniacal filtrate from the iron. It was then removed by acidulating with sulphuric acid and heating with a strip of aluminum; the metal was then washed, redissolved in nitric acid, and determined by the iodide-thiosulphate method. The filtrate, after removal of iron and copper, was neutralized by sodium carbonate, acidulated with a fixed amount of hydrochloric acid, diluted to 200 c.c., heated and zinc estimated in it by ferrocyanide with uranium indicator. Thiocyanate was estimated by acidulating 10 or 20 c.c. with hydrochloric acid, adding ferric chloride, and comparing the color with standard thiocyanate under the same conditions; in some cases ferrocyanides precipitated and required to be filtered off. Ferrocyanide was calculated from the iron found above. The methods for estimation of ferrocyanides and thiocyanates based upon oxidation by permanganate were found to be totally unreliable when tested experimentally upon solutions containing known quantities in presence of the substances accompanying them in cyanide solutions. The colorimetric methods give fairly approximate results. Sulphate was weighed as barium sulphate, precipitated by adding barium chloride to 100 c.c. of solution, after first adding an excess of hydrochloric acid, heating till odor disappeared, and filtering off any zinc and copper ferrocyanides, prussian blue, or silver chloride that fell out. The solid residue was obtained by evaporating 20 to 50 c.c. in a nickel or platinum dish; the former appears to be the less attacked by cyanide solutions and fused residues. Alkalinity toward methyl orange was determined (*a*) by direct titration of 25 or 50 c.c. with decinormal acid, (*b*) by adding the standard acid in considerable excess, heating till all odor disappeared, and titrating back with standard alkali; the results were rendered somewhat uncertain by the precipitation of zinc compounds and ferrocyanides.

Antidotes for Cyanide Poisoning.—Hydrogen peroxide is a powerful antidote for cyanide poisoning. It is applied as a 2·5% solution in subcutaneous injections, which are performed every four minutes at different parts of the body. At the same time the stomach is washed out with a 2% solution of the peroxide. Hydrogen peroxide forms with hydrocyanic acid a harmless compound, oxamide, 2 $CONH_2$. This remedy is used regularly at the Crown Deep cyanide works at Johannesburg, S. A. R. A good many accidents in treating zinc-gold slimes with acid which are attributed to cyanide poisoning are really due to the evolution of arseniuretted hydrogen from the decomposition of arsenic contained in the zinc.

BARREL AND VAT CHLORINATION AND BROMINATION.

Chlorination at Mount Morgan.—Two classes of ore are treated, oxidized and sulphide. The former is dried in revolving cylinders, crushed in Krupp ball mills and calcined in revolving cylindrical furnaces, 57·5 ft. long and 7·5 ft. in diameter exclusive of the fire-brick lining. The ore is 20 minutes in passing through this furnace. Since there is only a trace of sulphides in the ore the calcination is chiefly a process of dehydration to improve the leaching. Three hundred pounds of wood, costing 2s. 6d., are required to calcine one ton of ore. The ore discharged from the furnaces is cooled by passing through cylinders 30 in. in diameter, with buckets bolted to the outside, revolving in a water trough so that the buckets raise the water and discharge it over the sides of the cylinder, whence it drips back into the trough.

The chlorination vats are pits in the ground lined with concrete, each 55×12·5×4 ft. inside and holding about 100 tons of ore. The filter bed is gravel with a top layer of sand. An automatic scoop, traveling on rails on the dividing walls between the vats, discharges the exhausted ore. The ore is leached with chlorine water, which is made from gas generated from salt, sulphuric acid and pyrolusite in stills heated by steam, the gas being collected in a gasometer, whence it is drawn for absorption in water, in scrubbing towers filled with bricks, old crucibles, etc. One hundred and ninety-four gallons of this chlorine water, costing 13d., are required per ton of oxidized ore, the extraction of the gold being 96 to 98%. The gold is precipitated from the solution by means of charcoal, as described further on in this review.

The sulphide ore is roasted in hand reverberatories, a Ropp straight-line furnace and a kind of multiple hearth furnace designed by Capt. Richards. The cost of reducing the sulphur to 0·09% or less is 23s. per ton with the Ropp furnace, 20s. with the reverberatories, and 10s. with the Richards furnaces, wood costing 17s. per ton. The Richards furnaces are 30 ft. long by 12 ft. wide with 11 superimposed hearths. The ore fed in at the top is moved from hearth to hearth by an intermittent air blast through 0·5 in. tuyères inserted at regular intervals in the sides of the furnace. At each puff of the blast a shower of ore is shot to the hearth next below. The dust is caught in a chamber in the flue, whence it flows back automatically into the furnace. The loss by dusting is said to be moderate. In the lixiviation of the roasted sulphide ore about four times the amount of chlorine is required as for the oxidized ore, the cost being 4s. 6d. per ton The extraction averages 94%.[*]

According to Thomas I. Dyson[†] the precipitation is effected by charcoal, which is prepared as follows: The charcoal is crushed and sifted; all that passes 20-mesh and remains on 30-mesh is called "coarse;" that which passes 30 and remains on 40 is called "fine;" all that passes 40 is thrown out; the "fine" and "coarse" are washed with water to free them from dust. The filters can be any size desired, but it is well not to have the charcoal column less than 18 in. in depth. The small filters at Mt. Morgan contain 12 cu. ft. of charcoal, measured loose, *i.e.*, 4 cu. ft. of "coarse" and 8 cu. ft. of "fine," 1 cu. ft. of "coarse" being placed on the bottom, and the 8 cu. ft. of "fine" above this; on top is put the remainder of the "coarse," with a heavy perforated sheet of lead to keep it down. The charcoal is stamped in hard with the foot, and espe-

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[*] J. D. Smith. *New Zealand Mining Journal*, Nov. 1898.

[†] Private communication to Dr. W. B. Phillips, to whose courtesy I am indebted for a copy.

cially around the circumference of the filter, to prevent the gold liquor finding its way between the charcoal and the walls.

For seven months the liquor running away from these filters assayed in one works 0·00316 oz. (0·76 grain) per ton and in the other 0·00787 (1·89 grain) per ton. Before passing through the charcoal the liquor must be warm. After passing the charcoal all liquors are run through a concrete tank with a bed of sawdust a foot deep and covered with lead plates like the filters. Every six months this is burned and from 15 to 20 cwt. of ferric oxide ash is obtained, assaying 200 to 300 oz. gold per ton, which is sold to smelters.

Formerly, when barrel chlorination was practiced, it was the custom to amalgamate the ash from the charcoal filters, which ran only 5 to 10% in gold; but after the introduction of vat lixiviation the ash became rich, assaying 30% Au. In amalgamation this formed rich slimes, assaying as much as 1,200 oz. Au per ton, which were sold to smelters; but the loss by this method was high, and moreover it is undesirable to put so much value into the form of a by-product. Smelting the ash in crucibles was then adopted, but the amount to be smelted was about 30 cwt., or 60 cwt. with fluxes, per month; so smelting in crucibles was very laborious, while the slags were always full of gold beads, which had to be amalgamated. A small reverberatory furnace was then designed, with a cast-iron pan to hold the brick hearth. The pan was supported on four screw standards, which could be moved up and down, and the whole hearth withdrawn from the furnace when necessary. Smelting in a sand hearth was tried, but was given up in favor of the simple fire-brick hearth. The gold dust bearing ash (2 parts), soda ash (1 part), and borax (1 part) were mixed dry on a concrete floor, and then moistened with water. On charging into the furnace this at once formed a hard cake from which no dust could be taken by the draft. After six months' work, during which time 75,000 oz. of gold were smelted, 25 ft. of the flue was torn down and crushed in a battery, and yielded only 35 oz. of gold, most of which came from the first 8 ft. The fire-brick hearth allowed gold to soak in, the joints being soon eaten out by the alkaline fluxes, and after three or four months the hearth was lowered, the bricks were torn out of the cast-iron pan, the large pieces of gold were picked out, and the rest was sent to a 5-stamp battery. A clean-up like this would generally produce 2,000 oz. of gold. The work was easy, and even if necessary to put in a new hearth every month, was cheaper than smelting in graphite crucibles. When the charge melted there was formed a layer of 6 in. of slag above the gold, and it took a long time to get this hot enough to melt the gold, but this proved to be unnecessary, the gold becoming semi-molten after four or five hours and settling out of the slag in irregular lumps; then the slag was tapped into a large mold (capable of holding 9,000 oz. of gold), whence it ran into the slag pots. This slag assayed only about 20 to 30 oz. per ton and contained no amalgamable gold. The furnace is then plugged and fired heavily for an hour and a half, and when the fusion is complete the charge is tapped into molds, as much as 9,000 oz. having been smelted in this manner in one operation.

The new works completed in 1898 crush, dry, and chlorinate 10,000 tons of low-grade oxidized ore per month. A plant of equal capacity for the treatment of the pyritous ore is now in course of construction. The cost of treating the oxidized ore per ton is as follows: Mining and transportation 2s., crushing (including drying) 2s. 5d., roasting 3s., chlorinating and discharging vats 3s. 6d., general 1s. 7d.—total, 12s. 6d. To this may be added 2s. 6d. for head office expenses, dividend tax (5%), and government royalty (6d. per oz.).*

Chlorination at Cripple Creek, Colo.—John E. Rothwell describes a new mill of 25 to 30 tons per 24 hours capacity. Five men are required for the day shift and three for the night. The ore is crushed with slow-speed rolls, roasted, cooled automatically by means of a device which occupies a floor space 5·5 ft. sq. and a total height of 11 ft. and takes the ore from the furnace at bright red heat and delivers it at a temperature so low that it can be held comfortably in the hand. In the chlorination department one man on a shift attends to the two barrels and the solution and precipitation, and has time to spare. The passage of the ore from beginning to end is automatic. The mill is so free from dust that the door between the engine room and crushing department can be left open, and there are no partitions between the different departments.†

Chlorination at the Delano Mill, Boulder, Colo.—John Roger describes‡ the practice at this mill, which was erected in 1897. The ore is crushed by two sets of 14×30-in. rolls, running at 45 r. p. m., to pass a 20-mesh screen made of No. 24 wire. These screens have about 70 sq. ft. of surface. The ore, which is of telluride character, is roasted in a Pearce furnace with hearth 8 ft. wide. The ore discharged from the furnace passes into an automatic cooling device which consists of a series of 120 tubes 3 in. in diameter and 4 ft. long, surrounded by a current of cold water. The ore is drawn from the lower end of the tubes, in which it remains about 20 minutes. The tubes are always full of ore. The chlorinating department contains three barrels, each designed to hold a charge of about 5 tons. They are 5 ft. in diameter and 8 ft. long inside, the shell being made of 0·5-in. steel and the ends of cast iron, with 12-in. trunnions. These barrels are lined with 18-lb. lead on the sides and 24-lb. on the ends. Each has a 12×16 in. charging hole and four 2-in. connections for filtering pipes. The barrels are provided with the usual filters. They are revolved at a speed of 5 r. p. m. There are eight solution tanks, each 6·5 ft. deep and lined with 6-lb. lead, in which the sediment is allowed to settle and the gold is precipitated from the clarified solution. The precipitation is done by hydrogen sulphide.

The chlorination of a charge occupies about 2·5 hours, the average quantity of chemicals required being 10 lb. of bleaching powder and 20 lb. of sulphuric acid per ton of ore. The ore treated assays 4·5% iron, 89·5% silica, 1·5% lime, 0·25% magnesia, 2·2% sulphur, 0·10% copper, and 0·65 oz. gold and 1·25 oz. silver per ton. In roasting, the percentage of sulphur is reduced to 0·4–0·04%, according to the quantity put through the furnace. The 40-ft. Pearce cannot desulphurize properly 50 tons of this ore per day, at least not with the inferior lignite which is used for fuel. The percentage of gold recovered is about 92. The amount of labor required per day shift is as follows: Sampling,

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* *Economist*, Nov. 5, 1898. † *Engineering and Mining Journal* Feb. 25, 1899. ‡ *Engineering*, July 22, 1898.

three men at $1·75; crushing, one man at $2·50 and one man at $1·75; roasting, one man at $2·50 and one at $2; chlorinating, one man at $2·50 and one at $2; precipitating, one man at $2. The night shift requires the same number of men in the crushing, roasting, and chlorinating departments. There are two engine men at $2·50, one laborer at $1·75, one carpenter at $2·25, and one millwright at $3. There are consequently 21 men, whose daily wages come to $45·75. Chloride of lime cost $2·50 and sulphuric acid $1·30 per 100 lb. at the mill. The cost of chlorinating is computed as follows: Labor $1 per ton, chemicals 55c., fuel 50c., other expenses 49c., total $2·54, not including general expenses or management.

The Colorado City Chlorination Works.—These works, owned by the Colorado-Philadelphia Reduction Co., were described by H. V. Crowell.* The ore is comminuted by Gates crushers and Reliance belted rolls, of which there are two sets of 14×36 in. in the sampling department and six sets of 14×30 in. in the crushing department proper; these crush the ore to pass a 12-mesh screen with No. 19 wire; ordinarily 5 sets of rolls crush 230 to 250 tons per 24 hours to this size. There is the usual system of belt elevators and revolving trommels, though there are some new devices in connection therewith. A novel arrangement is the position of the main line shaft, which is placed on the ground floor directly under the rolls. The roasting department is equipped with two 100×14 ft. Ropp furnaces and one 8 ft. Pearce, having a combined capacity of 200 to 225 tons. The ore, containing 2·6 to 3·5% S, is reduced to 0·02% S after deducting soluble sulphates. The Ropp furnaces do 90 to 94 tons per day and the Pearce 55 to 62. Oil residuum is used for fuel, and the results are said to be excellent, the temperature of the furnace being absolutely under control. The roasting furnaces and the drying furnaces connect with the dust chamber. The roasted ore is cooled in a Durant automatic cooler, after which it is conveyed mechanically to the chlorinating department. The latter is equipped with 10 barrels, each 6 ft. in diameter and 12 ft. long, arranged to turn at 4 r. p. m., and holding 19,000 lb. of ore at a charge. They are piped so that the pressure can be used either from the main water supply tank or from the steam pump. The filters in the barrels are constructed of a sheet of 4-lb. lead, with fine perforations, supported by a sheet of lead ⅜-in. thick, with ⅜-in. perforations, and held in place between two gratings of hard wood. The last are fixed by cross bars keyed under side bars bolted to the shell of the barrel. The heads of the barrel are made of cast iron heavily ribbed, and are designed to withstand a pressure of 100 lb. per sq. in. The shell is made of ⅝-in. steel. The barrel is lined throughout with 24-lb. sheet lead. The chemicals are carried from the store room to the barrels in buckets suspended from two wheeled crawls running on overhead tracks. In this way the time of charging is made very short. In the operation of the barrels the usual practice is followed. They are rotated from one to three hours. In filtering, a pressure of 20 to 40 lb. per sq. in. is maintained. The time of filtration varies from 45 minutes to four hours. The settling tanks have a capacity of 80,000 gal. They are lined with 6-lb. lead. The precipitation tanks have a capacity of 60,000 gal. Hydrogen sulphide is employed as precipitant. The sulphides

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**Engineering and Mining Journal*, Oct. 8, 1898, p. 425.

after roasting contain 72% Au, producing bullion ·940 fine or over; the slag from the melting contains about 0·5% of the total value handled. There is the usual system of pressure tanks, filter presses, melting furnaces, etc. The works treat from 4,500 to 6,500 tons per month. There is very little manual handling of the ore and in design and construction the works are thoroughly modern.*

Gold Chlorination at the Plymouth Rock Mine, Cal.—According to T. K. Code the ore is a mixed sulphide of zinc, copper and lead, assaying 1·5 oz. gold per ton, together with some silver. Cyanide lixiviation of gold followed by chloridizing roasting for extraction of the silver was unsuccessful. Chlorination of the gold followed by lixiviation of the residues with sodium hyposulphite has given an extraction of 90 to 96% of the gold and 86 to 91% of the silver. The gold was precipitated by ferrous sulphate. After the gold had been thrown down, copper was recovered from the solution by electrolysis, 60% of the copper in the ore being won. When the percentage of copper in the solution was too low for this method, it was precipitated with scrap iron. The total cost was $6·25 per ton. About 5,000 tons of ore were treated by this method.†

The Black-Etard Permanganate Process.—This consists in the lixiviation of gold ore in vats, after the same manner as in the cyanide process, with a solution of potassium permanganate and chlorhydric acid or potassium permanganate, sulphuric acid, and sodium chloride, the idea being to develop chlorine, which will attack the gold and make it soluble as chloride. Two solutions are made up as follows: (A1) 100 gal. water with 40 to 60 lb. of chlorhydric acid; and (A2) 100 gal. water with 12 to 16 oz. potassium permanganate; or (B1) 100 gal. water and 20 to 40 lb. strong sulphuric acid, and (B2) 100 gal. water with 20 to 40 lb. of sodium chloride and 10 to 20 oz. of potassium permanganate. The solutions are mixed before application to the ore, but after mixture they must be used within 24 hours to retain their strength. The ore if a sulphide or telluride must be thoroughly roasted before the solution is applied. Tanks should be lined with enamelled iron, glazed earthenware, or coated with paraffine paint or some substance which will not affect the permanganate solution. The lixiviation requires 12 to 72 hours, according to the coarseness of the gold. If the reddish-violet color of the solution becomes faint or disappears during the leaching, more should be added, but not otherwise. To precipitate the gold from the solution, sulphurous acid or ferrous sulphate is use, preferably the former. It is claimed that free chlorine is not generated by this process, but that the chemicals concur to effect the solution of the gold so long as the latter is present, herein differing from the ordinary chlorination process. Precisely what is meant by this is not clear, and the process may be classed as a modified chlorination process, which is probably more expensive, wasteful, and inefficient than ordinary barrel chlorination.

The Cassel-Hinman Bromination Process.—C. Wilson Moore, at the annual

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* Much of the information in this account is taken from a private letter from C. M. MacNeill, Esq., general manager of the works.

† T. K. Code private communication, Jan. 3, 1899.

meeting of the Sheba Queen Gold and Exploration Co., London, October 11, 1898, stated that he had treated a lot of 18 cwt. of ore assaying 5·5 oz. gold per ton by this process with recovery of 95% of the gold and loss of bromine of only 0·4 lb. The cost of performing the process was estimated to be about the same as the cyanide process. It is doubtful if so favorable results will be obtained in practice.

SMELTING GOLD AND SILVER-BEARING PYRITES.

In Reverberatory Furnaces.—At the Selby works in California two furnaces are used in this work, one of them 35×16 ft. in the clear and the other 35×18 ft. Crude petroleum is used as fuel. The ore is fed directly into these furnaces from Ropp roasting furnaces. The 35×16 ft. furnace averages 60 to 65 tons per day, while the other handles 70 to 75 tons. The concentration ranges from 10 : 1 to 10 : 0·75. The furnaces are lined either with raw chrome iron ore or chrome bricks. The lining lasts as long as six to seven weeks without a pound of fettling.

Pyritic Smelting in Cupola Furnaces.—The Bi-Metallic Smelting Co., of Leadville, in 1898 erected a new furnace, 36×215 in. at the tuyères which is believed to be the largest pyritic smelting furnace that has ever been built. Pyritic smelters are in operation in Colorado at Ward, Ouray, Silverton and Buena Vista. The Buena Vista works were built by persons connected with the Mary Murphy mine. At first they had difficulty in obtaining satisfactory mixtures of ore, but after some experimenting this was overcome. The Kokomo furnace has been out of blast for the last five years.

PAN AMALGAMATION OF SILVER ORES.

Cost of Silver Milling in Montana.—The total receipts from the Elkhorn mines in 1897 were £65,851; expenses were £61,718. The production included 715 tons of ore sold to smelters, which contained 109 oz. gold, 96,872 oz. silver and 90 tons of lead. In the pan amalgamation mill 10,917 tons of ore were treated, to which 2,609 tons of salt were added. The yield was 420 oz. fine gold and 489,960 oz. silver. The yield was 92·16%. Average work per stamp per day, 1·91 tons. The cost of mining was $15·2140; milling, $7·9722. These costs were divided as follows: Mining superintendence, $0·7163; mine labor, $9·1486; ore house labor, $0·6212; fuel, $3·0343; explosives, $0·6404; timber, $0·0992; illuminants, $0·0788; repairs and supplies, $0.4525; surface expenditures, $0·1495; taxes, insurance, etc. $0·2732. Milling superintendence, $0·4298; labor, $2 6475; salt, $2·2187; fuel, $1·1171; quicksilver, $0·4479; chemicals, $0·0623; repairs and supplies, $0·6323; surface expenditures, $0·1567; taxes, insurance, etc., $0·2599. The total cost of mining and milling was therefore $23·18 per ton.

Silver Milling at Tintic, Utah.—The mill of the Eureka Hill Mining Co. has 100 stamps, of 950 lb., which drop 7·5 in. 100 times per minute. The ore is crushed wet through a 35-mesh wire screen and concentrated on Frue vanners. The tailings from the vanners run into 5-ft. standard amalgamating pans, of which there are 32, accompanied by 16 8-ft settlers and the usual clean-up pans, etc. The ore averages about 20 oz. silver and $2 gold per ton and 4%

lead. About half the value is taken out by the vanners as a concentrate averaging 75 oz. silver, $6 to $7 gold and 22% lead. The tailings from the vanners pass into settling tanks, whence they are drawn into the pans in charges of about 3,000 lb., sufficient slimes being added to bring the pulp to the proper consistency. About 5% salt, 3 lb. bluestone, 2 lb. sulphuric acid, 2 lb. iron borings, and 2 oz. concentrated lye are charged with the ore. Steam is turned on and the temperature raised to about 180° F., which takes two hours, after which 200 lb. of quicksilver are added. The pan is run six hours more before discharging into the settler. The average extraction is about 80%.[*]

Silver Milling at Potosi, Bolivia.—The production of tho Royal Silver Mines of Potosi, Bolivia, Ltd., in 1894, 1895 and 1896 was respectively 304,268, 235,048 and 456,794 oz. The loss in treatment was 16·8, 23·97 and 12·95%. The loss of quicksilver per oz. of silver produced was 0·336, 0·17, 0·24 oz. The high extraction in 1896 was due to improvement in percentage of chlorination, which now exceeds 90% of the silver contained in the ore.

The Velez Patio Process.—The report of the directors of the Huautla Santa Ana Mining Co. for the fiscal year ended July 3, 1897, showed a production of $169,481.91, at an expense of $80,757.54. The report of the general manager, Willard S. Morse, gives results of the metallurgical practice at Huautla (State of Morelos). During previous administrations it had proved impossible to treat the ores successfully by pan amalgamation. After a series of careful experiments it was found that they could be treated economically by a combination of mechanical dressing with the new patio process invented by J. M. Velez, of Guanajuato. This process has yielded an extraction of 90 to 95% of the total value of the mineral, at a cost of $10 per metric ton.

Lixiviation of Silver Ores with Potassium Cyanide.—At the Chloride Point mine, Utah, the ore is crushed to 0·25 in. size and leached in steel vats 20 ft. in diameter and 8 ft. deep, with capacity of 100 tons of ore. There are ten of these vats. Tailings are discharged from the bottom by the Butters' patent discharge into cars which are trammed to the dump. The ore is of cherty character and almost free from clay, which permits it to be charged in the vats to a depth of 7 ft., while the existence of the value in seams and fractures of the ore instead of being finely disseminated through it makes fine crushing unnecessary. The silver of the ore occurs in the form of chloride, which is readily soluble in potassium cyanide solution. The strength of the solution employed is 0·5%. The first solution stands 10 hours, and when drawn off carries 18 to 20 oz. silver per ton of solution. The next liquor is of the same strength. It is allowed to stand longer than the first. This procedure is continued for five days or more, when a solution of 0·25% is put on, and finally a wash of clear water. The total treatment lasts from seven to ten days. The consumption of cyanide is 2·5 to 3 lb. per ton of ore. The crude ore assays 18 to 28 oz. silver per ton and 50c. to $1 in gold. The extraction is between 65 and 77%. The silver and gold are precipitated by means of zinc in the usual manner. The bullion assays ·800 to ·945. This is the product obtained from the zinc boxes by simply drying and screening through a 10-mesh screen.

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[*] *Transactions Institution of Mining and Metallurgy*, 1898.

So far as I know this is the only place in the world where the cyanide process has been applied to the treatment of a silver ore, although it has been done previously on a small scale in treating amalgamating mill tailings in Nevada.

MISCELLANEOUS REFERENCES.

Electrolytic Gold Refining.—The process worked out by Dr. Wohlwill at the Norddeutsche refinery at Hamburg depends upon the use of impure gold bullion as anode in a hot solution of gold chloride containing an excess of chlorhydric acid or of an alkali metal chloride. The cathodes are thin sheets of pure gold hung 3 cm. from the anode. The current density varies from 95 to 190 ampères per sq. ft. At the lower density an electromotive force of one volt is sufficient. The silver is recovered as chloride in the anode slimes. Platinum and palladium pass into solution, whence they are recovered at long intervals by chemical means. The process has been in operation for some time at the works of the Frankfurter Gold und Silver Scheide Anstalt, where a plant covering 64 sq. ft. of ground suffices to produce 75 kg. of fine gold per 24 hours.

Melting Points of Gold and Silver.—D. Berthelot has made careful redeterminations of the melting points of gold and silver, and sets down 1064° C. as the correct figure for the former and 962° C. for the latter. These points are important, since they are often employed as the basis for the standardization of pyrometers.

Estimation of Gold and Silver by Formaldehyde.—L. Vanino states that gold can be estimated by adding to a chloride solution ordinary formaline (an aqueous solution of formaldehyde) followed by a few drops of caustic soda and warming for a little while on the water bath. The precipitated gold is filtered off, well washed with water and alcohol and dried at 180° C., or heated to redness in a crucible. Silver in solution as nitrate can be determined accurately by the same method, either in the cold or on the water bath, the only precaution necessary being to wash well with absolute alcohol to remove adhering water. Silver chloride is completely and rapidly reduced by the same reagent, and the process is available as a means of preparing pure silver from silver residues.*

Literature.—Following are references to certain technical papers on the metallurgy of gold and silver, which have not been quoted in the preceding paragraphs: "Notes on Dry and Wet Crushing with Cyanide Treatment, in New Zealand," by John McConnell, *Transactions Institution of Mining and Metallurgy*, November 16, 1898. "Notes on Dry and Wet Crushing," by Frank Merricks, *idem*. "On the Successful Treatment of Tailings by the Direct Filling Process on the Witwatersrand," by F. Cardell Pengilly, *Transactions Institution of Mining and Metallurgy*, Vol. VI., 1897-98, pp. 113-117. "Notes on the Action of Cyanogen on Gold," by James Park, *idem*, pp. 120-126. "Some Notes on the Recovery of Gold from Cyanide Solutions," by Sherard Cowper-Coles, *idem*, pp. 219-224. "Some Notes on Gold Milling," by W. H. Vale, Vol. V., *Transactions Australasian Institute of Mining Engineers.*

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* *Berichte der Deutschen Chemischen Gesellschaft*, xxxi. (11), pp. 1763-1765.

"Notes on the Milling of Gold Ores," by John E. Hardman, *Journal of th* *Federated Canadian Mining Institute*, 1897, Vol. II. "On the Mining of Lo Grade Gold Ores in Nova Scotia," by C. F. Andrews, *idem*. "The Direc Treatment of Auriferous Mispickel Ore by the Bromo-Cyanide Process at De loro, Ontario, Canada," by Hugh K. Picard, *Transactions Institution of Minin* *Engineers*, August, 1898, p. 417. "Hydraulic Dredging, or the Working o Deep Alluvial Deposits by Elevator with Centrifugal Pumps," by A. S. Ken yon, Vol. V., *Transactions Australasian Institute of Mining Engineers*, Vol. "Treatment of Slimes in South Africa," by Charles Butters, *Journal Chemic* *and Metallurgical Society of South Africa*, Vol. I., No. 1, February, 189 "Charcoal as a Precipitant of Gold from Cyanide Solution," by N. D. Powe *Australian Mining Standard*, October 14, 1897.

PROGRESS IN GOLD MILLING.

BY ROBERT H. RICHARDS.

*Milling Gold Ores on the Witwatersrand.**—In this district most of the min would not pay satisfactorily by amalgamation alone, but do yield satisfacto returns by combining amalgamation with cyanide treatment. For this reaso and because it pays best to work the stamps for high capacity, it is n attempted to make amalgamation as complete as possible, though this metho of extraction is not neglected. The State Mining Engineer's report for 18 gives the number of stamps for that year as 4,291, with an average duty p stamp of 4·392 tons in 24 hours. The extraction was $6·41 per ton amalgamation, and $3·07 by cyanide—total, $9·48. Large mills are the ru few having less than 60, while most have 100 or more stamps. The new Simm & Jack mill has 280 stamps.

Level mill sites have to be used. The ore is generally raised to the bins inclines, though occasionally by elevators. There is generally both inside a outside amalgamation, and occasionally concentration. The concentrates a chlorinated, the tails cyanided. The mills are almost invariably built near t mines instead of near water supply. Buildings are framed with timber a covered with corrugated iron.

The ore bins generally have sloping bottoms (40° to 45°), but a few have f bottoms. Bottoms and sides are generally made of double thickness of 2 or in. plank with sheet-iron lining. Mortar blocks are 12 to 15 ft. high, made pine timbers 14½ in. square, firmly bolted together. They rest on 1½ to 2 ft. concrete. The pit is excavated to hard rock where possible, and is walled with masonry or concrete. The space around the mortar block is filled wi tailings or concrete. A sheet of thick rubber, or occasionally felt, is plac under the mortar.

The mortar boxes resemble the Homestake pattern, but are larger than mort boxes generally, being from 11½ to 18 in. wide at the discharge. They gen ally have cast-iron or steel liners and false bottoms. The screens are of ir or steel wire, 22 to 30 mesh, 26 being very common. Diameter of holes rang from 0·063 to 0·084 in. (1·6 to 2·1 mm.). Screens last between two and thr

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* H. H. Webb and Pope Yeatman, *Engineering Magazine*, June, 1898.

days on an average. The height of discharge is regulated by chuck blocks, or sometimes by using a set of screens with different widths of frame.

As large crushing capacity is of prime importance, unusually heavy stamps are used, the recently built mills going up to 1,100 to 1,250 lb. According to the State Mining Engineer's report for 1896 the average was 973 lb. The shoes and dies are 8½ to 9 in. in diameter and chrome, manganese, cast and forged steel are used. Heads and tappets are of cast steel. The stems and cam shafts are made of faggoted iron. There is generally one cam shaft to a battery; in some cases one shaft to two batteries. The cams are made of cast steel. The Blanton cam is popular.

The drop is 7 to 9 in., averaging about 8 in.; speed 92 to 104, average about 96; order, 1, 3, 5, 2, 4, most commonly. The water in battery amounts to 8 or 9 tons per ton of ore. The water is repumped, the loss being estimated at about 25%. Amalgamation is done both inside and outside the batteries. Apron plates are 10 to 12 ft. long and 4½ to 5 ft. wide, sloping 1 to 1¾ in. per ft. Sand wheels are generally used to elevate tailings. They require but little power or attention. For lifts of more than 40 ft. plunger pumps are used. The latter have given good service when clear water has been supplied to the plungers to prevent scouring.

The details of mill designs, stamp batteries, power arrangements, etc., are given from the Simmer & Jack and Rand Mines Co.'s mills. The following figures are given showing stamp mill data from 15 mills:

At the Windsor mine a 750-lb. Sandycroft stamp does 3·3 tons per 24 hours; at the New Heriot a 1,050-lb. stamp does 4·4 tons; at the May Consolidated one of the same weight does 4·5 tons; at the George Goch Amalgamated an 1,140-lb. Krupp stamp does nearly 5 tons; at the Crown Reef an 1,175-lb. stamp does 4·9 tons; at the Glen Cairn a 1,250-lb. Sandycroft does 5·09 tons; at the De Koop one of the same make and weight does 5 tons, stamping through a screen with 1,000 to 1,200 meshes per sq. in., or 31·6 to 34·6 meshes per linear inch; at the Village Main Reef a 1,250-lb. Fraser & Chalmers stamp, making 92 drops per minute, does 5 tons; at the Lancaster battery a 1,250-lb. Krupp stamp does about 5 tons; at the Simmer & Jack a 1,250-lb. stamp does 4·8 tons; at the Nigel Deep 1,325-lb. Sandycroft stamps are used; at the Robinson, Witwatersrand and Treasury the duty per stamp is respectively 5·5, 5·1, and 5·11 tons per 24 hours. It should be remarked that at some if not all of these mills the nominal ton is really less than 2,000 lb., and where this is the case the figures are too high.*

Reason for Coarse Stamping in South Africa.†—It is found that, owing to the gold being deposited on the outside of the pebbles (the ore is conglomerate), it is not necessary to reduce the material as finely as in American practice.

Ore Sorting at the South African Gold Mines.‡—The principle of picking out the country rock, which, owing to the thinness of the reef, has to be broken in some mines, is being increasingly adopted on the Rand. The working costs

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* These figures are from various issues of the *South African Mining Journal* for 1898.

† Edward D. Chester, THE MINERAL INDUSTRY, Vol. VI., p. 383.

‡ Webb and Yeatman, *Engineering Magazine*, April, 1898; A. C. Key, *Cassier's Magazine*, May, 1898; *South African Mining Journal*, March 19, 1898, p. 550; Aug. 13, 1898, p. 935.

depend in great measure upon this sorting, because the barren rock can be picked out for very much less than the milling cost, especially as the cheap native labor (50c. per day) easily learns to do the work satisfactorily.

The surface work is done either at the shafts or at a central station. Belts 30 to 50 ft. long, circular tables about 30 ft. diameter, or floors, are used. With rotary tables it is customary to have pickers inside the picking surface as well as outside. For this purpose at the Crown Deep, for example, a table 25 ft. outer diameter has a rail bolted to its under side, and this rail travels on supporting wheels. In previously designed plants the rail was stationary and the wheels attached to the table, but it was found that lumps of rock fell on the rail and caused trouble. The table is driven by bevelled gears.

In general the ore is dumped over grizzlies with 1½ in. or larger spaces. The oversize is wet from a hose before picking. Native laborers pick out the waste, and the good ore then goes to Blake or Gates type of breakers, set for a product 3 in. in diameter or less. At some mines where the reef is narrow double sorting is used. In this case the grizzlies make three sizes, the finest going direct to the mill, the next being picked and then sent through a breaker to the mill, while the coarse, after a preliminary picking and crushing, is again picked and crushed.

In the single operation the amount picked out probably averages from 12 to 16%, though at the Ferreira it has sometimes risen to 40% on floors alone. At the Jumpers Deep the mine ore is dumped over a grizzly with about 1¾ in. spaces. The undersize goes to the mill, the oversize to a trommel, on which a large amount of water is used. The undersize and oversize of the trommel are picked on separate revolving tables, with 4 ft. picking width and outer diameter of 26 ft. The good ore remaining from the undersize is broken in a Comet crusher for the mill; that from the oversize is broken in a Blake, and passes over a grizzly, the oversize of which is again picked. All of this work is done in a rock house at the mine. The proportion of the mine ore hand picked has been as follows: March, 1898, 30·23%; April, 32·79%; May, 35·01%; June, 34·70%; July, 34·87%.

The following figures represent practice in 1897: Ferreira, 61,596 tons picked out, 125,326 tons milled; Rose Deep, 47,750 tons mined, 8,678 tons picked out, or 18·2%; Crown Deep, 81,188 tons mined, 9,605 tons picked out, or 11·8%; Geldenhuis Deep, 231,066 tons mined, 22,515 tons picked out, or 9·7%;* at the Robinson the average for 1897 was 11·4% picked out, while with the new equipment 18·9% is picked out;† at the Simmer & Jack 15% is picked out.‡ At the Geldenhuis Deep the waste assayed only 0·056 oz. per ton and at the Ferreira only 0·025 oz.

The cost of picking and crushing, with tables or belts, varies from 5½d. (11c.) to 1s. (24·3c.) or more per ton milled, according to the amount sorted out. At the Meyer & Charlton, with belts, it has been 5·54d. (11·08c.) per ton milled. At the Geldenhuis Deep, with round tables, it has averaged 5·58d. (11·16c.) per ton milled. With sorting floors, where as much as 35·5% of waste

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* *South African Mining Journal*, March 19, 1898. † *Idem.*, March 26, 1898. ‡ *Engineering and Minin Journal*, June 4, 1898.

has been picked out, the cost of sorting alone has amounted to 10d. (20c.) per ton milled. At the Ferreira the cost per ton of waste was between 44 and 45c. in 1897.

Cyaniding of Concentrates Compared with Cyaniding without Concentration.—The George Goch* mine, with ore assaying 8 to 9 dwt., using 60 Krupp stamps, 1,140 lb. each, and stamping through 22½ mesh nearly 5 tons per 24 hours, formerly treated the pulp from the amalgamated plates directly by cyanide, and the final tailings assayed 1½ dwt. per ton. Now a pointed box is placed at the foot of each amalgamated plate. The spigots of these are treated on 24 vanners, and the overflows on 12 more. The pyrite concentrates are treated by cyanide, while the tailings are too poor to pay at present. The saving of amalgam in the catch boxes on the vanner is sufficient to pay for the cost of the vanner treatment, and the final average tailings are reduced to an assay of 10 grains per ton. Mr. J. R. Williams makes the statement† that the combined processes of the mill, the cyanide works and the slimes plant on the Rand extract from 90 to 95% of the total gold contents of the ore at a cost not exceeding 7s. per ton.

Stamp-mill Practice at the Utica Mine, Calaveras County, Cal.‡—W. J. Loring describes in detail the practice at the Utica stamp mills, giving data as to size and life of wearing parts, the operation and cleaning up of the mill, and the cost of milling. At this mill a spare cam shaft, fitted with its 10 cams, is kept ready in case a cam shaft breaks. By this means a broken cam shaft causes the hanging up of 10 stamps for only three hours instead of for 48 hours in case the broken shaft has to be stripped and a new one turned and fitted. The stamp shoes are cast with hollow necks, that is, a 2½-in. hole 3½ in. deep is left in the neck, having ½ in. of metal around it. This reduces the weight about 3 lb., saving at least 25c. a shoe in first cost, besides the freight from New York.

An iron apron bolted to the lip of the mortar carries a 1 × 12-in. rough board, the surface of which is flush with the lip. This board has been found to be a good amalgamator, after a month's use, amalgamating as quickly as a copper plate, but it will not stand the jar as well as a plate. It can be cleaned in an eighth of the time required for a plate.

The shoes are 8½ in. in diameter, while the dies are 8¾ in. This has been found better than to have the two diameters the same, because the stem guides are bound to wear, allowing the stamp to swing. By having the die ¼ in. larger than the shoe the full surface of the shoe is utilized. If, however, the difference in diameter is greater than ¼ in. the shoe will "cup," causing a loss of efficiency. Seven men per 24 hours, besides the superintendent, operate the 60-stamp mill. In 1897 the total cost of milling, exclusive of power, was 13·8c. per ton. As the company owns its water rights the expense for power was, of course, comparatively small.

Effect of Temperature in Amalgamation.—Thomas J. Grier,§ manager of the

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* *South African Mining Journal*, 1898, p. 504.

† *Ibid.*, 1898, p. 551.

‡ *Trans. American Institute of Mining Engineers*, October, 1898.

§ *Engineering and Mining Journal*, Jan. 29, 1898, p. 126.

Homestake mine, South Dakota, found that more complete amalgamation of gold took place when the battery water was at 50° F. than when at 60 to 70°. The most probable explanation appears to be that at the higher temperature the amalgam becomes so liquid that it will not remain on the plates. The difficulty certainly was not due to oil in the warmer water, for there was no chance for oil to get into it.

A. von Dessauer states[*] that at the Twelve Apostles mill in Transylvania the summer temperature was from 80 to 90° F.; in winter 34 to 36° F. The theoretical and actual extraction in both seasons showed hardly any difference, but the amalgam from warm water had 25 to 35% of gold, while in winter it had only 7 to 10 or 12%. In the spring and autumn the stiff amalgam that formed during frosty nights would soften and run off in the daytime.

J. J. MacDowell,[†] manager of the Rose Deep mill, says that the actual temperature makes no difference, but changes cause trouble. Personally he prefers about 90° F., but has had good results at 76° and 100° F.

A mill is reported[‡] where a saving of 90% was effected with water at about 80° F., while the saving was only 80% with the water at about 60° F., the conditions being otherwise similar. The ore carried pyrite and some galena. When using the warmer water the mercury squeezed from the amalgam contained more gold than with the cooler water.

At the Virtue mine, where the only available water had a temperature of 90° F., much less mercury was fed per ounce of gold than would have been required with colder water. The amalgam both in the battery and on the plates was kept rather harder than is common, because the mercury drained off the plates when it was softer.

W. Frank Drake[§] has found that the water should be above 50° F. to give the best results. A low temperature lessens the cohesive power of the mercury, especially when sulphurets are present in considerable quantity, and the globules run off the plates and are lost. By heating the water to not less than 80° or more than 90°, the quicksilver became normal and the maximum saving of gold was attained. Less mercury is required with a higher temperature. Care has to be used that the amalgam does not become too thin.

Fine Gold Produces Hard Amalgam.[‖]—At a certain mill where the water is practically constant at 70° F., and the gold is fine and of high quality (often ·940 fine), it is found impossible to keep the amalgam soft, but the results nevertheless are good. The amalgam is so hard that it is very difficult to remove it, even with scrapers. A successful method consists in packing the lip plates in hot sand for half an hour, and in spreading the same on the apron plates, renewing it as necessary. This heating softens the amalgam so that it is easily scraped off. Arthur L. Collins, from his experience at Blackhawk, Colo., confirms[¶] Woakes' opinion that fine, high-quality gold produces hard amalgam.

Lay System of Placer Working.[**]—The Mindrinetti Co., operating placer mines

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[*] *Journal Chemical and Metallurgical Society of South Africa*, April, 1898, p. 52. [†] *Ibid.*, p. 41.

[‡] *Mining and Scientific Press*, Nov. 19, 1898, p. 505. [§] *Ibid.*, Dec. 3, 1898.

[‖] Ernest R. Woakes, *Engineering and Mining Journal*, Sept. 24, 1898, p. 370.

[¶] *Ibid.*, Oct. 8, 1898, p. 423.

[**] H. Tweddle, *Engineering and Mining Journal*, July 23, 1898, p. 100.

in Guiana, uses the Lay system, which is designed for situations where there is little head of water and not enough grade to remove the tails. It consists in hydraulicking with pump pressure (100 lb.), sluicing to a sump, lifting from the sump to a wash trommel by a centrifugal pump, running the fines from the trommel over riffles containing mercury, and elevating the tailings from the latter by a second centrifugal pump, in order to get sufficient grade to carry them away. The wash trommel is a cylindrical framework 4 ft. in diameter, 30 ft. long, stayed by longitudinal I-beams, and carried on hollow trunnions, which are supported in movable stirrups to permit variable slope. The cylinder is covered with perforated plate. A spray pipe passes through the hollow trunnions. As the trommel revolves the I-beams lift and drop the ore, thus thoroughly disintegrating the clay. The coarse stuff passes off to the tailings sump and the fine to the riffles. Working in moderately stiff ground, containing a considerable amount of plastic clay, each of the 2-in. nozzles handles an average of something over 30 cu. yards per hour.

*Laboratory Tests and Mill Tests.**—At the Ottawa Gold Milling and Mining Co.'s mill, Keewatin, Ontario, careful tests have been made to determine how nearly laboratory tests, properly conducted, would check with regular mill runs in an amalgamating gold mill. Pulp from the sampling works, crushed to the mill mesh (usually 40) is carefully sampled and a portion put through 120-mesh. On this material six to 36 assays are made (depending on the size of the lot, richness, condition of free gold, etc.). Three to twelve charges of 100 g. each are taken from the 40-mesh stuff and each put into a wide-mouthed bottle with some water and 100 g. of mercury. The bottles are then tightly covered and are agitated for 30 minutes in a frame that is operated by a vertical eccentric rod making 350 two-inch throws a minute.† The contents of each bottle are then panned separately, first to remove the mercury, then to remove the concentrates. The concentrates and tails are dried, weighed, and assayed, and the amount of gold amalgamated determined by calculation. It has been found more satisfactory to use several 100-g. charges than a single charge of 1 kg. To test the accuracy of this method several lots were milled and cleaned up separately; some of the results are given below:

Lot.	Net Weight. Tons.	Laboratory Results.			Mill Clean-up. Bullion.	Laboratory Variations.	
		Assay Value. Per Ton.	Free Gold. Per Ton.	Total Free Gold.		Free Gold.	Total Value.
A..........	18·8	$24·70	$20·50	$385·40	$390·29	−1·3%	−1·0%
B..........	22·8	24·08	20·90	476·52	468·60	+1·7	+1·5
C..........	38·0	4.12	3·51	133·38	131·61	+1·3	+0·9
D..........	49·5	2·53	1·65	81·67	84·80	−3·7	−2·5
E(a)........	195·97	14·42	11·29	2,212·50	2,226·97	−0·65	−0·5

(a) Seven lots combined.

In regard to concentrates, a factor has to be used, since closer work can be done with a pan than in the mill.

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* H. A. Guess, *Canadian Mining Review*, October, 1898, p. 270.

† Better satisfaction was had from this than by either side shake or by clamping the bottles to a revolving shaft.

The above method enables the company to treat small shipments at l rates than otherwise, because the expense of individual clean-ups is not incu

Estimating Weight of Ore by Bulk.[*]—Previous to 1897 it was the custo the Rand to call a 20 cu. ft. truck load of ore a ton of 2,000 lb.; but from eral tests at the Ferreira mine it is found that an ordinary truck load is 1,900 lb.

Economy of Water.[†]—The demand for economy of water in stamping cyaniding gold ores in South Africa has led to the use of lime as a coagu or precipitant for the slime, and spitzkasten and settling vats in which the is separated and the water clarified. By these devices the loss of water, in ing the generation of steam, is claimed to be brought down to 20%, and i case even to 10%. The quantity lost in the sands (probably 22-mesh to pe 100-mesh) after cyaniding is 12 to 14% of the weight of the sands (28·8 to gal. per ton of sand); and that lost in the slimes is taken to be about eq the weight of the slimes (240 gal. per ton of slimes).

The owner of a mine in an arid region desired to estimate the losses of in milling.[‡] He concluded that with proper care to minimize leakage evaporation the greatest loss would be the absorption by the ore. To tes quantity lost in this way he obtained samples of pulp, both of which had ground to pass a 60-mesh screen in a dry process mill, and after treatmen remained some months under cover, and were to all appearances free moisture. Of each 33⅓ lb. was thoroughly saturated with water, so that piled on a slightly inclined board a little water would drain off. One o samples, which was from a very soft ore (apparently nothing but "ta absorbed 1·6 gal. of water, or 96 gal. per ton of ore. The other sample dently from a much harder ore than the first, absorbed 1·1 gal., or 66 ga ton of ore.

Regularity in Quantity of Feed.—J. A. Wilkes[§] writes that no one kno well as the manager of a cyanide plant which is only just large enough to the average amount of ore crushed by a mill how provokingly irregular i tonnage of the average stamp mill per day and week. The losses due to must be considerable, as it necessitates either irregular treatment or ru tailings to waste. This irregularity can be prevented by (*a*) roughly re ing the number of shoes and dies replaced at one time; (*b*) putting in fals as the old ones wear; (*c*) seeing that night men do not feed faster to sav trouble of shifting tappets; (*d*) keeping the height of discharge nearly stant; (*e*) placing suitable weights on the stems as the shoes wear. Thi is the most effective but the most neglected expedient in the Rand mill has been tried and abandoned, but the reason was that old tappets were which were too long for short stems, and were so awkward and troubl that the men did not put them on. If some care had been used to des weight for this purpose the cyanide works would not be deluged one wee starved the next.

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[*] *South African Mining Journal*, March 19 and 26, 1898.

[†] *Ibid.*, 1898, pp. 490, 659, 674.

[‡] *Engineering and Mining Journal*, April 16, 1898, p. 459.

[§] *Journal of Chemical and Metallurgical Society of South Africa*, April, 1898, p. 42.

THE HYDRO-METALLURGY OF SILVER.

BY OTTOKAR HOFMANN.

In this paper I intend to refer only to the lixiviation of silver ores with sodium hyposulphite solutions, setting forth the present development of this branch of metallurgy. The operations of the process are: Chloridizing roasting, base-metal leaching, silver leaching, precipitation, and refining of the precipitate. The subject of chloridizing roasting I reserve for a future paper.

BASE METAL LEACHING.—If a complex ore be subjected to chloridizing roasting the metal sulphides are converted into chlorides, subchlorides, sulphates, and oxides. A number of these salts are soluble in water, and must be removed from the roasted ore by leaching with water before silver extraction, in order to prevent them from entering the silver solution and afterward the silver precipitate. Besides the heavy metal salts more or less sodium chloride which was not decomposed during roasting will also be dissolved by water, together with the sodium sulphate which is formed in roasting.

Solubility of Silver Chloride in Base Metal Chlorides and Salt.—The base-metal chlorides as well as sodium chloride have the property of dissolving silver chloride, their dissolving energy increasing with the temperature and concentration. It is therefore not advisable to leach with hot water or to use too deep leaching vats, even if the filtering quality of the ore permits it, because the water by passing through a thick layer of ore becomes highly charged with these salts and considerable silver chloride will be dissolved. A very dilute chloride solution does not dissolve silver chloride, and in order to prevent the base metal chlorides from dissolving silver, the washing of the ore should be so conducted that a sufficiently dilute solution will be produced.

Prevention of Loss of Silver in Base Metal Solutions.—In tank lixiviation this can only be accomplished to a certain degree (not perfect) by two methods which were introduced by the writer: (1) The water is allowed to enter the vat below the filter until it gradually rises above the ore. Thus the most concentrated portion of the solution will appear above the ore, and if then diluted with a stream of water and the course of filtration reversed a large part of the silver chloride will be precipitated and remain in the ore, while the outflowing solution will be dilute, although not sufficiently so as to be entirely free of silver. (2) A quicker but not quite so effective a method is to fill the tank partly with water and then to dump the ore into it, either dry or moist. Enough water should be taken to rise about 4 in. above the charge when complete. By this method the outflowing solution will not be so dilute as by the first, but much more so than it would be if it had been charged in an empty tank and then leached from above. However, the only method by which this problem is actually solved is trough lixiviation, also introduced by the writer. By this operation the ore can be brought at once in contact with such quantities of water as to produce a solution so dilute that not a trace of silver will be dissolved. Concerning this method of lixiviation I shall write further on.

The amount of silver dissolved during base-metal leaching varies greatly and depends on the nature of the ore, the thickness of the layer, the amount of salt used in roasting, the mode of treatment in the vat, and the temperature of the outflowing solution. In some works, usually in large ones, the ore is charged steaming hot from the cooling floor, or if for special reason it be charged dry it will enter the tank even hotter, and as a matter of course the first part of the outflowing solution will be very hot and contain considerable silver. To illustrate how the amount of silver dissolved by the base-metal solution varies under different conditions, a few examples may be given:

San Francisco del Oro, Parral, Mexico.—Heavy zinc-lead ore with iron pyrites, roasted with 4% salt; the tanks partly filled with water and the ore dumped dry and cool into it; average value of roasted ore 26·1 oz. per ton; charge, 8 tons; leaching time, 8 hours; silver dissolved, 1%.

Sombrerete, Zacatecas, Mexico.—Heavy ore containing zinc blende, galena, iron pyrites and some sulphuretted copper minerals, roasted with 6% salt; the tank partly filled with water and the ore dumped dry and warm into it; average value of roasted ore, 42·6 oz. silver per ton; charge, 52·5 tons; leaching time, 12 hours; silver dissolved, 1·23%.

Cusihuiriachic, Chihuahua, Mexico.—Not so heavy an ore, containing galena, zinc blende, some iron and copper pyrites, some silver-copper glance and also some ruby silver; roasted with 8% salt; the ore wetted on the cooling floor and charged slightly warm into an empty tank and leached from above; average value of roasted ore, 47·9 oz. silver per ton; charge, 8 tons; leaching time, 53 hours; silver dissolved, 2·5%.

Determination of Amount of Silver Dissolved.—Mr. Russell reports the amount of silver dissolved from the Cusihuiriachic ore as 11·6% if the ore is charged cold and dry and leached from above. Though Mr. Russell roasted with 10% of salt and I used only 8%, I believe his figure to be much too high. His method of ascertaining the amount of silver dissolved is not correct. He determines on a sample in the laboratory the amount of salts soluble in water contained by the roasted ore, and from this figure he calculates the amount of silver which ought to be contained in the ore after leaching with water, compares it with the actual amount found in it by assay, and then estimates the amount of silver dissolved. This is not correct, because a large charge in the works cannot be so thoroughly washed as a small sample in the laboratory, and therefore the calculated value of the washed ore will be much too high, and consequently the calculated amount of silver dissolved. Besides, as Mr. Stetefeldt correctly remarked, there exists more or less water, free from salts, under the filter of the vat, left there from the second wash water of a previous charge, which will precipitate silver chloride when the concentrated brine meets it, the said silver chloride being subsequently dissolved by the hyposulphite solution.

The correct way to ascertain the amount of silver dissolved is to collect separately in vats the whole base-metal solution of one charge and measure its volume. If it be then determined how much silver is contained in 1,000 c.c. it is an easy matter to calculate the total amount of silver dissolved. To take the sample out of the vats would not be correct, because the solution being collected in different vats will not be uniform in concentration and therefore will not be uniform as to tenor of silver. Besides, a large portion of the silver will be precipitated by the gradual dilution of the first concentrated brine, which, to a certain extent, will escape the sample, even if the contents of the vat be agitated. However, there is no difficulty in obtaining a correct sample. A small rubber tube, terminating with a glass tube drawn to a fine point, is inserted in the outlet of the vat and left there during the whole time of base-metal leaching. Thus a very fine stream of the outflowing solution is obtained, and if collected in a glass vessel of proper size, a representative sample of about 2 or 3 gal. of the whole solution is obtained. The volume of this sample is correctly measured and all the heavy metal salts contained therein precipitated with calcium or sodium sulphide. The precipitate is collected on a filter, dryed, weighed and assayed, and the amount of silver contained in it is calculated. With this figure and the total volume of solution collected in the vats the total amount of silver dissolved during base-metal leaching may be estimated. The figures of silver dissolved under different conditions quoted above were obtained by this method and therefore may be considered correct.

Effect of an Excess of Salt.—If the proper amount of salt be used in roasting, so that all, or nearly all, be decomposed, and the leaching be done by one of the improved methods, it may be assumed that the amount of silver dissolved will never exceed 3%. An excess of salt will cause a large percentage of silver to be dissolved, reaching 60 to 70% in my own experience. In this case the ore was of such a nature that it required a large excess of salt in roasting to produce a satisfactory chlorination of the silver. Of this very interesting occurrence I shall speak below.

Precipitation of Silver from Base-Metal Solutions.—There are several ways of precipitating the silver contained in the base-metal solution, including precipitation by dilution with water, to be described further on, and precipitation by one of the following reagents: (1) Milk of lime. This produces a very voluminous precipitate which is rather difficult to handle, and from which the silver cannot be extracted unless there is a smelting furnace connected with the leaching works. In such a case lime may be used to advantage, especially since it is so cheap a reagent, but care must be taken not to use the base-metal solution too concentrated, in which case it is coagulated and much trouble in further handling is caused. (2) Calcium or sodium sulphide. These precipitants are used in most of the lixiviating works, but they are rather expensive, because they precipitate base metals also. If sodium sulphide be added gradually to a tank charge of base-metal solution, it will be observed that the precipitate formed first is the richest in silver, and that it becomes poorer as precipitation goes on. A complete precipitation, however, can be produced only when all the base metals are thrown down. Since the principal portion of the silver is precipitated in the beginning, and only the smaller portion remains in solution with the main bulk of the base-metal salts, the point will be reached in precipitating, when the cost of the precipitant will exceed the value of the precipitated silver, and therefore a complete precipitation is not advisable. In most lixiviation works only a partial precipitation is performed, after which the solution with the remainder of the silver, which it did not pay to precipitate, is allowed to run to waste.

Recovery of Silver from Waste Liquor by Precipitation with Copper.—Although the waste liquor will be found very low in silver, still if its large volume be taken in consideration it will appear that the loss occurring in this way during a year is important. This loss, however, can be diminished greatly by precipitation with copper, which acts most energetically if finely divided like cement copper, and if the solution be warm. To effect a perfect desilverization it is necessary to treat the heated solution in charges and to use mechanical appliances to produce an intimate contact with the cement copper, but the base-metal solution, after a partial precipitation with calcium or sodium sulphide, is not rich enough to warrant an expensive and complicated treatment, and it will be found to be more profitable to employ a more primitive method, even if not all of the silver is recovered. The true economy in metallurgy, as in any other industry, is to save the most at the least expense. As soon as its expense exceeds the value of the recovered metal, a method ceases to be practicable, no matter how interesting it may be. That method should be adopted which is best suited to local conditions. I was criticised severely by Mr. Stetefeldt in his book on lixiviation because I clung tenaciously to the old method of treating the silver sulphides on a lead bath in a cupelling furnace and not by an improved method like that with the Roessler converter, which he highly recommended. Later Mr. Stetefeldt had an opportunity to try this method, but was not successful, while the refining of the sulphides on a lead bath is still practiced and is so far the most rational method.

The largest part of the dissolved silver can be saved by conveying the solution through a series of flat tanks in which cement copper is so divided that it offers a large surface of contact to the solution. The exhaust steam of the engine can be used to increase the temperature of the stream, the steam being made to enter the solution through pipes about 12 to 18 in. below the surface in different tanks of the system. If the base-metal solution contains copper, the pipe projecting into the solution ought to be of lead. The steam being condensed a vacuum is formed, and very little if any back pressure to the engine will be noticed. Wherever the climate allows it these tanks can be built in the yard outside the works, without roof or shelter. If the ore contains copper, sufficient cement copper will be formed by placing scrap iron in the tanks, but if copper is wanting, the cement copper has to be made from a dilute solution of blue vitriol. Ores which are treated by lixiviation are seldom entirely free from copper, and for this reason it is well to place some scrap iron with the cement copper.

Base-Metal Leaching at Sombrerete.—At Sombrerete, Mexico, I conducted the base-metal leaching in the following manner: The ore is very permeable and Stetefeldt and Russell, who built the works, erected leaching vats 15 ft. 6 in. in diameter and 7 ft. 6 in. deep, capable of receiving 55 to 58 tons of roasted ore. These tanks are decidedly too deep. The ore filters freely enough to permit a deep charge, and in silver leaching this would not interfere, but in base-metal leaching it caused much trouble. I did not wet the ore on the cooling floor to produce additional chlorination, and charged it dry. The water passing through 7 ft. of roasted ore became so saturated with salts that they crystallized and blocked the filter, the space below the filter, and the outlet pipe, which interfered greatly with the work and was very annoying. I overcame this difficulty by filling the vats to the depth of about 3 ft. with water and dumping the ore into it; frequently the ore had to be charged while quite hot in order to make room on the cooling floor, and consequently the outflowing base-metal solution was warm and dissolved more silver chloride than it would have done if more favorable conditions could have been maintained. But notwithstanding this, the amount of silver dissolved was much less than when the ore was leached from above.

If a base-metal solution containing silver chloride be diluted with water the silver chloride will precipitate as such. In the Sombrerete works there are two base-metal precipitation vats 9 ft. 9 in. in diameter and 9 ft. deep, and in order to take advantage of the above reaction I allowed the base-metal solution to run into both simultaneously, by which method I got the concentrated portion of the solution, which contains the most silver, divided evenly into the two vats. This left considerable room in them for diluting, and by allowing the less concentrated and finally the very weak solution to run evenly into both vats, I obtained a uniform solution dilute enough to cause the main portion of the dissolved silver chloride to be precipitated. Silver chloride thus precipitated being very finely divided requires a long time to settle. For the purpose of effecting quick settling and at the same time precipitating more silver, I added 5 to 10 gal. of sodium sulphide solution to each vat with agitation. In this way a part of the silver chloride was converted into sulphide, while the undecomposed part was readily collected by the flocculent precipitate of copper, lead and other base metals, and after agitation was interrupted the whole precipitate would settle quickly.

After the precipitate has settled the clear solution is decanted by a stiff rubber hose, which enters the vat close to the bottom and projects above the surface of the liquor. By lowering this a few inches below the surface the liquor flows out, and if the hose is lowered gradually as the liquor runs out it is always the clearest part of the latter which is drawn off. This method of decantation is better than a series of tubes inserted at different levels in the side of the vat, because during precipitation these tubes are filled with precipitate, and in decanting the first solution from each tube contains precipitate and consequently has to be conveyed to filters.

Outside the building I constructed a series of flat square tanks 2½ ft. deep, built of stone and mortar, well plastered and coated with asphaltum varnish, which were provided with a number of movable wooden double benches, or shelves, loaded down with scrap iron. In this way the scrap iron was well distributed throughout the tanks and offered a large surface to the solution. They were so arranged that the flowing solution would move in its whole depth and had to take a zigzag course.

The ore of Sombrerete contains about 2% copper, and the outflowing solution for a considerable time is colored green by cupric chloride, and a good deal of copper is precipitated by the scrap iron. The precipitated cement copper incrusting the iron is very loose and spongy and offers a very large surface for precipitating the silver. It ought not to be disturbed by stirring the solution, since thereby the copper falls to the bottom and does much less service. However, it is difficult to precipitate all the silver, and it requires a rather large number of tanks. Silver dissolved in a cupric chloride solution cannot be completely precipitated until all the cupric chloride is decomposed; therefore as long as the solution leaving the last tank still gives a reaction for copper it can be assumed to contain some silver also. This indicates the necessity of erecting one or more additional tanks. In Sombrerete these tanks were cleaned once a month and the cement copper obtained from them contained 500 to 600 oz. silver per ton, and 60 to 70% copper.

Cupric chloride in solution in contact with cement copper, especially at an elevated temperature, is converted into cuprous chloride, which settles as a heavy white crystalline precipitate. Cuprous chloride again in contact with metallic iron is converted into metallic copper, and the iron into ferrous chloride. It is clear that in these tanks, in which the cement copper surrounds the iron, an intimate contact between the iron and the cuprous chloride is not possible, and that consequently the cement copper will contain a large percentage of cuprous chloride. This is not a desirable associate for the cement copper, because in the subsequent treatment it always causes a large loss of copper and silver, and therefore the cuprous chloride ought to be decomposed. This should be done shortly after the cement copper is taken out of the tanks, because if left exposed to the air the cuprous chloride will change into oxichloride. The quickest and most rational way is to charge the cement copper in a revolving barrel with an addition of water, salt and some light scrap iron. The barrel is made of 3-in. staves and heads, and is bound with copper hoops. It is constructed like an amalgamation barrel. The inside is lined with hardwood lath about 1½ in. thick, so that the body of the barrel is protected from the wear and tear. Four longitudinal ribs, projecting about 2½ in., are inserted diametrically in the lining, which produce a lively mixing of the charge. The axles, or trunnions, which are made of brass, or better of bronze, are provided with a strong flange by which they are bolted to the heads of the barrel. Each is bored and provided with a stuffing-box through which a copper pipe enters. One of the pipes is turned downward close to where it enters, but must not project so far into the lower half of the barrel that the scrap iron and the cement copper will strike it. Through this pipe steam is introduced to heat the pulp. The pipe that passes through the other trunnion is turned upward high enough to just clear the ribs of the barrel. This pipe serves as outlet for the gases and steam and leads outside the building. A manhole on the side of the barrel, provided with a copper frame and lid, serves for charging and discharging, while another in the head gives entrance in case the lining is to be renewed or some other repairs are to be made.

When the barrel is charged with cement copper, water and iron, salt is added, steam turned on, and the barrel is put in revolution. A warm solution of salt (sodium chloride) dissolves readily cuprous chloride, and from this solution the iron precipitates the copper with great energy. Water should be used moderately, just enough to produce a lively movement of the pulp, say three to four times the volume of the cement copper, and to produce a strong brine without using too much salt. The reaction creates heat and the escape pipe has to be watched. When the steam commences to come out freely the steam inlet ought to be closed. The side of the barrel is provided with a plug-hole through which the operator can take a sample by means of a flask fastened to a copper wire. If the filtrate does not show a reaction for copper the process is finished. This takes about 45 minutes. Below the barrel is placed a square flat filter tank, above which is a large, strong iron screen with ½-in. holes, the frame of which rests on four car wheels on rails, so that it can be moved easily or withdrawn entirely from the tank. The contents of the barrel are discharged on this screen, and the cement copper washed into the filter tank by a stream of water, while the scrap iron remains on the screen. Then some sulphuric acid is added to remove the basic iron. This acid solution is allowed to pass through the cement copper, after which the latter is well washed. By this method I obtained cement copper containing 90 to 95% copper.

If the cement copper has to be prepared from blue vitriol it may be desirable to use it for a longer time, in which case it will become much richer in silver, but it has to be taken out occasionally and treated with dilute sulphuric acid to remove the basic salts. Thus purified the cement copper acts again with energy.

In order not to interfere with the regular work, the tanks have to be so arranged that during cleaning half of them remain in operation while the others are disconnected and cleaned. As soon as half are cleaned, the scrap iron or cement copper is put in place and the solution allowed to enter again while the other tanks are cleaned.

Precipitating the Dissolved Silver Chloride by Dilution with Water.—This method was first recommended and introduced by the writer. All alkaline and metal chlorides have the property when concentrated of dissolving silver chloride, and dropping it again as such when diluted with water. The precipitation takes place in proportion to the dilution, and if sufficient water be added all the silver will be precipitated. This method of desilverizing the base-metal solution is undoubtedly the most effective and cheapest, all that is required being a sufficient supply of water and a few more vats for base-metal precipitation than are usually found in lixiviation works. I used this method first at the Silver King mill in Arizona in 1880 to 1882 with very good results, then in various other localities where the supply of water permitted it, and also at the mill of the Hidalgo Mining Co. at Parral, Mexico, in 1894.

Experience at Parral, Mexico.—The observations I made at Parral were very interesting, and I shall give them in detail. The ore which was treated consisted principally of galena, lead carbonate and blende, and was almost free from iron pyrites. Neither galena nor blende produces in roasting with salt much chlorine, especially if the blende belongs to that variety which contains little or no iron sulphide. In this case the chlorination of the silver depended principally on the chlorine produced by the action of the quartz on the salt and by the direct action of volatilized sodium chloride. Such roasting requires a large excess of salt and a high heat. The roasting was conducted in a White-Howell furnace with 9 to 10% of salt. Experiments were made with the aim of reducing the proportion of salt, but as soon as the amount fell below 9% the extraction suffered so much that it was more rational to use a higher percentage of salt. If the ore had contained sufficient iron pyrites 3 to 4% of salt would have been sufficient, because lead-zinc ores in presence of iron pyrites require less salt for a successful roasting than any other class of complex ores; but since the iron pyrites was wanting and an excess of salt had to be used, and since zinc blende and galena act but slightly on the salt, the roasted ore contained a large amount of sodium chloride.

The lixiviation mill of the Hidalgo Mining Co. is well constructed and arranged, and reflects credit on Mr. J. T. Long, who erected it. This mill has a very large cooling floor, on which the roasted ore is allowed to cool dry for three days. Notwithstanding this long time, the ore, when charged, is still hot enough to impart to the outflowing base-metal solution for a rather long time a temperature of 140° to 200° F. The excess of salt contained in the roasted ore dissolved readily in the water of this temperature, forming a highly concentrated brine in which 60 to 70% of the silver was dissolved, and therefore the principal extraction was done during base-metal leaching. This, of course, required a careful treatment of the base-metal solution.

Sodium sulphide was used as precipitant, but a large quantity was required on account of the large amount of lead contained in the solution. If a zinc blende ore containing iron pyrites is roasted with the proper amount of salt and charged cool into the vat the base-metal solution will contain but little lead, if any, providing cold water is used; but if a concentrated hot solution of sodium chloride is formed in leaching, the result is entirely different, since this dissolves not only lead chloride, but also lead sulphate, and the solution will contain large quantities of these lead salts. If the solution be very concentrated, and allowed to cool, large crystals will be formed, while the mother liquor remains clear. If the solution be not very concentrated there will be only a crystalline precipitate which turns the solution milky. Neither the crystals nor the crystalline precipitate contains much silver. The main part of the silver chloride, lead sulphate and lead chloride remains in the solution, but if this be diluted with water it turns milky, forming a heavy white precipitate; and if sufficient water is used the white precipitate will contain all the silver and lead dissolved in the solution before diluting.

For illustration I shall record my observations made on charge No. 344: The ore when charged was rather hot. The leaching was done from above. The solution flowing out first had a temperature of 200° F. and was unusually concentrated. I collected some in a large beaker and left it to cool. A large amount of transparent crystals was formed. After pouring off the clear mother liquor I added to the latter a quantity of water, which caused a heavy white precipitate. This was separated from the solution by filtration, and the three substances were assayed for silver with the following results. (1) The crystals contained 7·6 oz. silver per ton; (2) the white precipitate had 3,386 oz. silver per ton; (3) the filtrate had none. Of this charge of ore, which contained 19·6 oz. silver per ton, 62.6% of the silver was extracted during base-metal leaching.

Working by this method I found it more convenient to do the precipitation in the trough leading from the lixiviating vats to the precipitation vats, by allowing a stream of water to enter the trough. In flowing some distance the solution and water become thoroughly mixed and the solution is perfectly desilverized before it enters the vats. The stream carries along the white precipitate, which settles in the precipitation vats. To ascertain whether the solution receives the proper quantity of water or not, a sample of the diluted solution just before it enters the vat is taken and filtered. Of the filtrate 50 c.c. are poured into a beaker and 150 c.c. of cool, clean water are added. If it remains clear after a few minutes it shows that no more water is needed; if it turns milky the stream of water has to be increased. Thus by decreasing or increasing the stream the proper dilution can be maintained. Of course the base-metal solution can also be diluted in the precipitation vats, but by this method, if the two streams simultaneously enter the vat, it is difficult to ascertain the proper proportion, and if part of the vat is filled with solution and then water added there will be a considerable loss of time. Where mechanical devices are used for agitation it is advisable to have the solution agitated while the vat is filling; where agitation is performed by hand, it ought to be done from time to time, to make the precipitate settle more quickly. The tanks are filled to about 12 to 18 in. below the rim, to leave room for more water in case the proper proportion was not maintained in the trough and a correction is necessary. When this is done a short time is allowed for the heavy part of the precipitate to settle, then one quart of sodium sulphide is added and the solution agitated again. This is done to convert the very fine particles of the precipitate, which otherwise would remain suspended for a long time, into sulphide, in which state they assume a flaky condition and settle much more quickly.

The lixiviation tanks were charged with 30 tons of roasted ore, and the base-metal leaching required about 24 hours. If the precipitation was not done by water but by sodium sulphide, the base-metal solution filled 7¼ tanks of 3,500 gal. each, and 53 gal. of the precipitant were used. The solution running to waste still continued 0·19 oz. silver per 1,000 gal., or 0·15 oz. per ton of ore leached. If the precipitation was carried so far that all the base-metals were precipitated, the solution running to waste did not contain any silver, but then the cost of the precipitant exceeded the value of the silver saved. If the precipitation was effected by dilution with water in the trough 22 precipitation charges of 3,500 gal. each were filled. The consumption of sodium sulphide amounted only to 2·75 gal., and the clear solution running to waste did not contain any silver at all. This result shows that it requires on an average about two parts of water to one part of solution. Of course, in the beginning, when the outflowing solution is very concentrated, considerably more water has to be added than in the above proportion, but the volume of water to be added decreases as leaching progresses, and toward the end it is less than the volume of solution.

On account of the large quantity of lead which the base-metal solution carried, and which is precipitated by water as well as by sodium sulphide, there was not much difference in the silver contents of the precipitate whether obtained by water or by sodium sulphide. In this particular case, however, a great difference in the silver contents of the precipitate was observed if the water for leaching was first applied below the filter or from the top. In the former instance it contained 1,200 oz. silver per ton; in the latter only 800 oz.

The precipitation with water is undoubtedly the cheapest and most effective mode of treating the base-metal solution, and it ought to be used in all works where there is an ample supply.

The Use of Cupric Chloride During Base-metal Leaching.—The chlorination of the silver, and correspondingly the extraction, can be increased greatly in ores which do not contain any or only a little copper, by use of a solution of cupric chloride while base-metal leaching is in operation. The cupric chloride can be prepared in the works by boiling one part of blue vitriol with two parts of salt in a wooden vat by direct application of steam. The cupric chloride is made in large quantities and used as required. It takes from 3 to 4 lb. of blue vitriol and 6 to 8 lb. of salt per ton of ore. To avoid the formation of too concentrated a solution the cupric chloride is not added until the leaching has proceeded one or two hours. It is better not to add the whole of it at once, but to divide it so that one-fourth is added to about 8 in. of water above the ore, and when this charge is disappearing below the surface another charge of 8 in. water and the second fourth of the cupric chloride is made, and so on until all is used. If the outflowing solution is green, it ought to be collected in a separate vat, and can be used over again either for the same charge or for the next one following, according to circumstances. It is best to lift this solution by means of a lead steam siphon or injector. If an ore contains so much copper that the base-metal solution is colored green, this reaction, which produces an additional chlorination of the silver, takes place in every charge in the regular operations of the process, and this is one of the reasons why cupriferous ores are the easiest to treat and permit the closest extraction. However, even with such ores, better results are obtained if the green base-metal solution is collected separately and allowed to pass through the charge a second time. Any defect in the roasting will be much lessened and uniformly good results will be obtained. The more inferior the roasting the higher will be the percentage of gain in chlorination. At Parral, where I treated the San Francisco del Oro ore containing 25·5% zinc, 11·56% lead and 1·02% copper, I gained on imperfectly roasted charges by the use of cupric chloride 32 to 38% in chlorination. At the Silver King, Arizona, where I treated an ore containing a large percentage of copper, I increased the extraction a little by using part of the green solution over again. With ore from the Veta Grande of Parral, which contained mostly zinc blende, galena and lead carbonate, and no copper, I experimented on a large scale with cupric chloride, and obtained much better and more uniform results.

SILVER LEACHING.—After the ore is freed from base-metal salts by leaching with water, a weak solution of sodium hyposulphite is introduced on top of the ore. It is of great importance to work with a weak solution. If the solution is strong, 1, 2, or 3%, and an attempt is made to maintain this as standard strength, it can be done only by frequent additions of sodium hyposulphite, else the volume of the stock solution will decrease rapidly and the preparation of a new stock solution will soon be required, because the ore absorbs a large volume of solution, and when, after the silver extraction is concluded, this solution is displaced by water, it is impossible to regain the same volume of the original strength. It will be found that only a comparatively small portion of the outflowing solution will have the standard strength, and if the displacement of the solution is done so as to maintain the same volume of stock solution, much water will enter the latter, while considerable hyposulphite salt will remain in the residues. Such a dilution of the stock solution takes place also in the beginning of the silver leaching. The "hypo" solution passing through the washed ore has to displace the water absorbed by the ore, and this cannot be accomplished without the first portion of the solution becoming mixed with the water, and since even a very dilute solution of sodium hyposulphite dissolves silver chloride, the outflowing solution has to be conveyed to the silver precipitation vats a long time before it reaches its standard strength.

Therefore it will clearly be seen that the standard strength can only be maintained by a continual addition of sodium hyposulphite to the stock solution. A continuous supply is furnished by the precipitant, whether that be sodium or calcium sulphide, because in preparing it some sodium or calcium hyposulphite is formed. As much as 5 to 7% of this salt may be found in the freshly prepared precipitant, without the latter being exposed to the action of the air for any considerable length of time. The constant supply from this source, however, does not suffice to replace the loss of hyposulphite salt which the stock solution suffers if the latter has to be kept at a strength of 1 or 2%, but it does suffice, and in fact in most cases exceeds the loss, if a weak solution is used. The consequence is that in the first case sodium hyposulphite has to be bought and added, which increases the treatment expense, while in the second case not a pound of this salt has to be added, even during years of operation, and on the contrary water has often to be added to reduce the strength to the standard.

Best Strength of Solution.—In my experience I have found the best strength of the solution to be 0·25 to 0·50%. Such a solution offers not only the above mentioned economical advantage, but it produces also a much cleaner and a richer in silver precipitate. Lead sulphate and cuprous chloride dissolve much more easily in a strong than in a weak solution, and therefore with a strong solution a very low grade precipitate will result and a much larger amount of the precipitant will be required. It is a wrong supposition that a strong solution shortens the leaching time and extracts the silver better. The strong solution, dissolving readily lead sulphate and cuprous chloride, becomes heavily charged with these salts and thereby loses much of its dissolving power for silver chloride, and therefore neither the leaching time will be shortened nor will the extraction be better.

Method of Leaching.—When base-metal leaching is finished and the sodium hyposulphite solution is to be applied, it is well to do this before all the water has disappeared from the surface of the ore. The solution then, in its course downward, follows closely the water and replaces the latter perfectly in all parts of the charge, and has less opportunity to become mixed with the water absorbed by the ore than if the water is allowed to drain from the ore before the solution is applied, and consequently the reaction, when the silver appears at the outlet, will be more precise. There is another advantage connected with it, inasmuch that if the charge be allowed to drain, the space before occupied by water will be filled by air, and when the hyposulphite solution is run on, it is very difficult, in fact almost impossible, to drive this air out again. The layer next to the surface will give up its air, which can be observed by the bubbles at the beginning, but further down in the charge the friction becomes too great, and the air, not being able to escape, will compress to let the solution pass, and by doing so will reduce the speed of filtration and prevent a quick and intimate contact between the ore particles and solution, thus increasing the time of leaching and decreasing the percentage of extraction. Therefore attention should be paid that during no time of lixiviation the liquid be allowed to sink below the surface of the ore. In a fresh charge the ore is loose and when water is applied on top it will, in descending, easily force downward the air, which will escape through the outlet.

Testing the Solution for Silver.—It requires close watching to ascertain the time when the silver appears at the outlet. If the liquid is tested with sodium or calcium sulphide the reaction is very uncertain, because the base metal leaching is seldom carried to such an extent that no light clouds are formed by an addition of these reagents, and therefore the reaction cannot show the first traces of silver in the outflowing stream. Even a very dilute solution of sodium hyposulphite dissolves silver chloride, and we can safely assume that, as soon as sodium hyposulphite can be detected at the outlet, the stream contains silver, and since furthermore the liquid naturally will contain more hyposulphite salt than silver, it is safer to adopt a method by which this salt can be detected in the outflowing stream. I use the following test, which is reliable and convenient:

A small strip of starch paper is dipped into iodine solution and then held in the stream. If the blue color disappears it is a sign that the liquid contains a hyposulphite salt and consequently silver, and the stream has to be turned at once into the trough leading to the precipitation tanks. The base metals have to be leached with cold water to make this test applicable, because hot water also discolors the blue paper. I advise all who practice the lixiviation process to use this test and see that when base-metal leaching is changed to silver leaching the outflowing stream is very closely watched. By doing so they will find that shortages in silver for which they could not account before will be avoided.

Effect of Lead and Copper.—If the ore contains lead and copper we shall find both metals in the hyposulphite solution, because lead sulphate and cuprous chloride which are present in the roasted ore are not soluble in water but dissolve in sodium hyposulphite. Both these metals are precipitated together with the silver, and we find them in large quantities in the precipitate, reducing materially the fineness of the latter. To remove the lead from the solution Mr. Russell precipitates it as carbonate by adding sodium carbonate to the solution previous to precipitation with sodium sulphide. By doing so he obtains a sulphide precipitate free from lead and lead carbonate as a by-product, but this complicates the manipulations without offering much practical advantage. Only where the sulphides are refined by melting them with iron and borax in crucibles is the use of this method justified, because if the precipitate is free from lead a silver bullion over ·900 fine will result. But this treatment is too expensive and inconvenient, and is not used except in small works and then only in exceptional cases. In large works the refining is done on a lead bath in the cupelling furnace, and lead in the precipitate is then of great advantage. To use Mr. Russell's method for the production of lead for the market is out of the question, because it costs more to produce it than the price realized for it.

For the sake of information I tried the Russell method in a modified way. Instead of precipitating the lead in the solution after it had left the lixiviation vat, I precipitated it inside the ore charge by adding sodium carbonate to the solution before leaching. The outflowing solution was entirely free from lead. The precipitated lead carbonate remained in the ore. No injurious effect on the extraction was noticed. The residues contained about as much silver as when leached without an addition of sodium carbonate, but the precipitate was much richer in silver.

Calcium Sulphide as Precipitant, and Action of Calcium Hyposulphite.—If calcium sulphide is used as precipitant the solution will contain after precipitation calcium hyposulphite, and it used to be generally assumed that within a short time the original sodium hyposulphite solution was replaced by calcium hyposulphite. I discovered and demonstrated that this is not the case; in fact that even if the original lixiviating solution were calcium hyposulphite, that compound could not exist in the process for any length of time, but would be converted into sodium hyposulphite. We read in standard works that Patera leached with sodium hyposulphite and precipitated with sodium sulphide, while Kiss used calcium hyposulphite as solvent and calcium sulphide as precipitant. It is mentioned that calcium hyposulphite possesses a greater dissolving energy for gold than the corresponding sodium salt, and that for this reason the Kiss process is more suitable for gold-bearing silver ores. However, Kiss was not leaching, as he thought, with calcium hyposulphite, but with sodium hyposulphite.

In roasting sulphuretted ore with salt, sodium sulphate is formed in large quantities. This salt is not easily removed from the ore by leaching with water. If leaching with water be prolonged it will be found that the outflowing liquid will react for sodium sulphate long after all the heavy metallic salts are removed. Therefore when the hyposulphite solution for the extraction of silver is applied the ore still contains sodium sulphate in considerable quantity. Calcium hyposulphite reacts with sodium sulphate, forming sodium hyposulphite and insoluble calcium sulphate. If, therefore, calcium hyposulphite is brought into the stock solution by the precipitant, or when calcium hyposulphite is used as solvent, it will be converted into sodium hyposulphite by the regular operation of the process.

Time Required for Lixiviation.—The time of lixiviation varies according to the nature of the ore, its permeability, and the size of the charge. I observed that if the main part of the silver in the ore is contained in galena, the silver extraction will be slow, even if the ore filters well, while if all or the principal part of the silver is contained in copper, zinc, arsenical or antimonial minerals the extraction is quick. At the Silver King mine, Arizona, a charge of 8 tons required only nine hours' silver leaching, though it contained on an average 200 oz. silver per ton; while at Cusihuiriachic, Mexico, the time required for a charge of 8 tons was 53 hours, the ore containing but 47 oz. silver per ton. The filtering capacity in both cases was very nearly the same. At the Silver King the silver was mostly contained in gray copper ore, antimonial fahlore and silver-copper glance, while at Cusihuiriachic it was principally contained in galena. I have made the same observation with ores of many other localities.

A free filtration is very important for a quick and thorough extraction. The ore particles should be brought rapidly in contact with fresh solution, which cannot be done if the filtration is slow. After the solution, descending through the ore, has dissolved a certain quantity of the salts present it loses much of its dissolving energy, and therefore the small stream of a bad filtering ore will not contain much more silver per liter than the large stream of a quick filtering ore. The same observation can be made if the outflowing stream of a quick-filtering ore be checked and reduced to a small stream. Likewise, if lixiviation is interrupted and the solution allowed to remain in contact with the ore for some time, say over night, the solution will not contain much more silver after leaching is resumed than the stream did before the interruption. Ores which after roasting run on the cooling floor like water always filter badly and are not suitable for tank lixiviation. Mixing such ores with sand or chopped straw does not improve their permeability. By mixing in a small percentage of galena before roasting, however, the ore loses somewhat of its dusty condition and permits a little better percolation. The only rational way, however, of treating such ores is by trough lixiviation.

Filters for Leaching Tanks.—A free filtration being of great importance for the success of the process, the selection of a proper filter is, therefore, also of great importance. Some use gravel and sand without a wooden filter bottom, as at La Baranca, Sonora, Mexico. The material of such a filter is cheap enough, but its preparation is troublesome, and therefore a filter is usually kept in use until the filtration becomes so bad that it has to be replaced by a new one. At Sombrerete straw filters were once used. Short straw was spread on a wooden filter bottom about a foot thick and then the ore was dumped on top of it. This rather coarse filter did not produce a very clear filtrate when new, but after the first charge it improved in this respect and produced a clear solution; however, the outflowing stream decreased with every charge, and when the filter had to be renewed it was found that the straw had rotted and packed tightly.

Filters like those described above, which cannot be cleaned, and are therefore kept in use as long as possible, invariably cause a decrease of the working capacity of the leaching vats and with it a decrease in production and percentage of extraction. I found burlap, the material used for making grain sacks, to be the best and most lasting filter. It is cut in pieces and spread over the wooden filter bottom so that one strip laps over the other about 3 in. The ends are rolled up and packed tightly around the circumference, or better into a groove 2 in. wide and 2 in. deep, made by nailing around the circumference and close to the filter bottom a 2 × 2-in. wooden strip and on this another strip 4 in. wide and 1 in. thick. After each charge the strips of burlap are carefully rolled up, so that no residue which adheres to them may drop below the filter bottom, are well washed and again spread on the filter bottom. If this is done regularly the filter is kept in the best possible condition. If the washing is not done after every charge a thin layer of residue will form on the filter cloth, gradually growing thicker and harder, partly by the repeated pressure of the feet of the laborers and partly by the deposition of calcium sulphate. The gypsum also incrusts the fibers of the cloth, makes it stiff and hard, and finally stops filtration entirely. Where burlap cannot be obtained coarse sheeting is a good substitute.

End of the Silver Leaching.—The extraction of the silver from a charge may be finished while the outflowing solution still gives a considerable precipitate upon addition of calcium or sodium sulphide. It is impossible to judge by the color of this precipitate whether it contains silver or not. It is, however, important to know when all soluble silver is extracted in order to avoid unnecessary consumption of the precipitant and loss of time. To ascertain when the end of the silver leaching has been reached a large beaker is filled with the solution and some calcium sulphide is added. The precipitate is allowed to settle, the clear solution decanted, and the precipitate poured on a paper filter and washed well. It is then removed from the filter and dissolved in nitric acid, filtered to remove the sulphur, and a drop or two of hydrochloric acid is added. If a white precipitate or only a cloudiness is produced, which by dilution and boiling does not disappear, there is still silver in the solution and the leaching has to be continued. If the solution remains clear the extraction is concluded.

The influx of sodium hyposulphite is then stopped and the solution is allowed to drain until it commences to disappear under the surface of the ore, when a stream of water is turned on to displace the sodium hyposulphite solution absorbed by the ore. It is only necessary to continue this second application of water for a comparatively short time; just long enough to keep the same volume of stock solution on hand. Then the charge is allowed to drain as dry as circumstances permit, after which the residues are discharged. This is done by shoveling them either into cars or, as is more convenient, into chutes, of which one is placed between each two tanks and which discharge into a large triangular trough beneath the vats, in which a current of water flows. The trough ought not to have an inclination less than 1 in. to the foot. In some works the residues are sluiced out. To make this method successful the vats must not be too deep nor of too large diameter. The stream of water has to be applied under pressure by means of a pump, or under good head from a storage tank.

PRECIPITATION.—In all modern lixiviation works the precipitation vats are provided with a mechanical contrivance to agitate the solution during and after precipitation, and only in antiquated works is this operation done by hand with a paddle. A horizontal beam about 12 in. shorter than the inside diameter of the vat is fastened to a vertical iron shaft. This horizontal beam, which moves above the surface of the solution, is provided with hardwood staves about 2 in. square and reaching down to about 1½ in. above the bottom. These staves are so arranged that when the agitator is in motion they cut the liquid with the edge. The agitator is set in motion by a friction clutch, and it should be started gradually to avoid breaking the staves. The inside of the tank is provided with four vertical wooden wings, projecting 3 in. toward the center and reaching nearly to the bottom. They break the violent current around the periphery and throw the solution toward the center, thus causing a strong whirling motion. The agitator has to make 30 revolutions per minute if the diameter of the vat is not more than 8 or 9 ft. Agitators of this construction are only durable when used in vats not deeper than 6 ft. In deeper tanks the staves will break and an agitator of stronger construction has to be used.

Another method of agitation which is very convenient is by the use of compressed air, furnished by a small compressor. From the receiver a pipe line leads along the whole row of precipitation vats. At each vat is inserted a T with a ¾-in. branch. These branch tubes are provided with valves and connected with a rubber hose 6 to 8 ft. long, the other end of which terminates with a ¾-in. gas pipe. This pipe must be long enough to reach any point of the bottom and still project about 2 ft. above the rim of the tank. If the vat is full and the air is turned on the solution is put into violent motion, as if boiling. It is well to change the position of the pipe from time to time, especially if the vat has a large diameter. This method of agitating the solution is now extensively used.

Adding the Precipitant.—The precipitant, whether sodium or calcium sulphide, is kept in a reservoir made of boiler iron, from which it is conveyed through an iron pipe to the precipitation vats. At each vat there is attached to the pipe a hose, which is closed by a pinch cock. In commencing to precipitate it is well to open the hose a little and to throw, by swinging it, some of the precipitant over the surface. By the appearance of the clouds which are formed, whether heavy or light, the precipitator can see at once if he has to precipitate a concentrated or dilute charge. Then he sets the solution in agitation and allows the precipitant to flow in. An experienced precipitator can judge by the color which is created when he splashes some of the precipitant over the surface, while the solution is agitated, the progress of precipitation, and knows when it is nearing the end.

While precipitation is going on the clouds which are formed become gradually lighter in color and toward the end almost yellow. When nearly finished the influx of the precipitant is stopped and after a few minutes the agitation also. Then the flaky precipitate is allowed to sink somewhat below the surface and some of the precipitant is splashed over the surface. According to the appearance of the clouds more or less precipitant is added, and the solution is agitated again. This operation is repeated until precipitation is complete. If by an addition of the precipitant no reaction takes place, it is well to throw some strong silver solution over the surface after the precipitate has partially settled. If the places where the silver solution fell turn reddish brown, the precipitant is in excess and more silver solution has to be added. One who is not experienced would best make this test in a beaker.

The Precipitate.—In precipitating the base-metal solution we have seen that not all the metals present are equally affected by the sulphur of the precipitant, and that the silver especially is more readily precipitated than the other metals. This is also the case if these metals are dissolved in the sodium hyposulphite solution, and therefore the precipitate which is obtained in the earliest stage of precipitation contains far more silver than that obtained later. Thus the operator has it in his power to make different grades of precipitate. This, however, does not offer such advantages as in base-metal leaching, and is of no direct practical value, because in order to maintain the dissolving energy of the sodium hyposulphite solution it is absolutely necessary to precipitate as perfectly as possible all the metals dissolved in it, but it explains why the black layer of sulphides, which we frequently find deposited on the surface of the ore charge, is so much poorer in silver than the precipitate itself. If the precipitation was done well, and ample time was given to the precipitate to settle, and the decantation of the solution was always performed properly and without mishap, we will not find any black deposit on the top of the ore charge; but such exact work is not always done, especially in large works, which seldom have a sufficient number of vats to give the precipitate ample time to settle.

The different sulphides settle according to their respective specific gravities. Lead, silver and copper go down first, while antimony, zinc, iron and free sulphur follow. While this separation is not theoretically perfect, it takes place to such a degree that the particles which settle last may contain but 30 to 100 oz. silver per ton, while the total precipitate may contain 5,000 to 15,000 oz. per ton.

This black layer of sulphides deposited on top of the ore charge after an extended lixiviation, being so much poorer than the precipitate, does not involve any notable loss of silver if it is carefully scraped off before the residues are discharged and the scrapings are mixed with the ore and roasted.

Accumulation of Sodium Sulphate in the Solution.—I stated above that when the leaching with water is stopped and the charge is ready for silver leaching the ore still contains some sodium sulphate, which during silver leaching will enter the stock solution. In course of time the sodium sulphate will therefore accumulate to such a degree in the stock solution that the latter will greatly lose in dissolving energy, a much longer leaching time will be required, and the percentage of extraction will suffer much. When such trouble arises operators usually try to "freshen up" the solution by adding more sodium hyposulphite to it. This, however, will benefit it only for a very short time, and the same trouble will appear again. Some add more hyposulphite every day, thus increasing greatly the cost of extraction without getting the solution back to its original energetic condition.

Calcium Sulphide as Precipitant.—The best method is to conduct the process so that no sodium sulphate will accumulate in the stock solution. This can be done by using calcium sulphide as precipitant. Sodium sulphate reacts with calcium sulphide, forming sodium sulphide and calcium sulphate, which precipitates, while the sodium sulphide acts as precipitant for the metal chlorides dissolved in the solution. On account of this reaction, which is of so great an advantage to the lixiviation process, I always advocated the use of calcium sulphide as precipitant.

The valuable effect of this reaction is especially felt if ore is treated which requires a large percentage of salt in roasting, whereby larger quantities of sodium sulphate are produced. Should so much sodium sulphate enter the stock solution that by the regular process of precipitation all the sodium sulphate is not decomposed, this salt will then gradually increase, notwithstanding the use of calcium sulphide, and will spoil the solution. It is therefore advisable to add at certain intervals an excess of calcium sulphide, and finish precipitation with silver solution. This will free the solution entirely from sodium sulphate.

If sodium sulphide is used this reaction does not take place, and not only will all the sodium sulphate dissolved during silver leaching accumulate and remain in the stock solution, but the amount will be increased from the precipitant, which always contains more or less sodium sulphate. In such works it is well to make provision for the manufacture of calcium sulphide, solely for the purpose of purifying the stock solution from time to time.

Treatment of the Precipitate.—The precipitate should be discharged from the precipitation tanks every other day. If it is allowed to accumulate for a longer time it changes from a flaky condition to a very fine powder, which settles slowly and will cause the solution in circulation to contain in suspension a considerable amount of exceedingly fine precipitate. The precipitate is discharged into a tank with a slowly moving agitator, varying in size according to the size of the works, from which the precipitate is drawn into a Montejus pressure tank in portions as required, and thence by means of compressed air it is forced into a Johnson filter press. Pressure tanks made of cast iron will resist longer the corroding action of the solution than those made of boiler iron, and they should be made so that the top or cover can be removed when required. The relief pipe should return to the sulphide tank with the agitator, because the air when relieved escapes with great force, and is apt to carry along some precipitate. Instead of compressed air, steam may be used, but the pressure will be limited by the pressure in the boiler, and often may not be sufficient. The feed pipe of the press should be in connection with water and air pipes, so that after the filter press is filled the precipitate can be washed by pumping warm water through it, and partially dried by forcing compressed air through it. When so treated the precipitate will come out of the press in hard cakes, permitting a clean handling of them. The press should be placed in a separate room with cement floor. Such a floor is easily kept clean and prevents loss of silver.

Instead of a pressure tank, a steam pump can also be used; only attention should be paid that the steam cylinder of the pump is larger than the pump cylinder, so that a pressure of 150 lb. or more can be produced with it. The pump cylinder should be provided with a bronze piston rod and liner, because iron is more or less affected by the solution and in course of time the inside of an iron cylinder becomes very rough. Much cleaner work can be done with a pressure tank, because there is always more or less leakage around the piston rod of a pump. The filtrate is conveyed to one of the lower solution reservoirs.

Removal of Sulphur.—The precipitant being a polysulphide, the precipitate will contain a large percentage of free sulphur, whether calcium or sodium sulphide is used. It is desirable to remove this free sulphur from the precipitate before the latter is subjected to a final treatment. The best method is to boil it with caustic soda in an iron tank, the caustic soda combining with the free sulphur and forming sodium sulphide, which serves as precipitant for the silver; but care has to be taken that no excess of caustic soda enters the stock solution, because it will exercise a decomposing action on the silver chloride in the ore. To avoid this the sodium sulphide solution thus obtained is conveyed to the boiling vessels in which the precipitant is manufactured. After boiling with caustic soda the precipitate shrinks much in volume and becomes very heavy. To separate it from the sodium sulphide solution after the main part has been decanted is a very slow process if done by common filters, and therefore it is much better to use a small filter press for this purpose. In works where no filter press is used, and a common filter has to be employed, these badly filtering sulphides can be made quick filtering by treating them with a strong silver solution to decompose all the sodium sulphide, which is the cause of the bad filtration. By this method 60% of the sulphur contained in the precipitate can be regained and brought into a state in which it can be used again as precipitant, thus greatly reducing the actual consumption of sulphur.

Another way of expelling and regaining the free sulphur from the precipitate is by distillation. The moist precipitate is discharged into retorts and heated, the sulphur vapors being conducted to brick chambers and condensed as flowers of sulphur. I used this method years ago on a large scale with satisfactory results as to the amount of sulphur regained, but the cast-iron retorts did not last long enough, principally on account of carelessness on the part of the men in charge, who overheated them; and as transportation of such heavy castings into the mountains was very difficult and expensive, I discarded this method. However, with careful firing, and in localities where transportation facilities are better, it can be applied to great advantage. While the extraction of the free sulphur will not be so complete as by boiling the precipitate with caustic soda, the operations are fewer. The product is dry and ready immediately for further treatment, while in the former method the product has to undergo the processes of filtering, washing, and drying.

A third method, by which, however, the sulphur is lost, consists in burning it off in a small reverberatory furnace. An actual roasting is not required; in fact it ought to be avoided to prevent loss by volatilization. The sulphides ought to be charged dry, but if they are charged moist they should remain undisturbed in the moderately heated furnace until dry, to avoid the generation of rapidly evolving steam, which is apt to carry away fine particles of the already dry part of the precipitate, thus causing a loss. Even if the precipitate has been previously dried in special ovens, the heat in the beginning has to be kept very low for some time to avoid mechanical loss by steam, because only seldom will the precipitate be perfectly free from moisture after leaving the drying ovens. Later the temperature is increased to ignite the sulphides. They commence to burn with a blue flame near the fire bridge, and the flame spreads gradually over the whole charge. When this takes place the fire has to be lowered to avoid overheating. No stirring should be done until the flame ceases; then a gentle stirring is given. This brings up new flames which, however, do not last long, but reappear if the charge is stirred again. Stirring is repeated until the flame ceases entirely. The temperature has to be kept so that when the flame ceases the charge is perfectly dark, which indicates that no actual roasting of the material took place and that only the free sulphur was burned off. If the precipitate is treated in this way no loss by volatilization will take place, and that is all which is required, because for the further treatment of the precipitate an actual roasting, a changing of the sulphides to oxides and sulphates, is not necessary; in fact, is hurtful. Mr. Stetefeldt, in his book on lixiviation, stated the loss to be as high as 6 and 12%, but this is not so.* However, there is a chance of loss by this method if it is not properly executed, especially if the precipitate contains antimony or is charged wet into the furnace.

The first method, *i.e.*, boiling with caustic soda, is the most rational and entirely excludes any loss of silver, and offers the additional advantage of regaining about 60% of the sulphur.

Refining the Precipitate.—No improvement in the final treatment of the precipitate has been made. The refining is still done with litharge on a lead bath in the cupelling furnace. Other methods have been tried, but so far not with much success. Stetefeldt's method of charging the roasted sulphides into a Roessler converter and furnishing the sulphurous acid gas for the converter by roasting another charge of dried sulphides did not work, because the Roessler converter failed to produce sulphuric acid.

Experiments to bessemerize the sulphides failed on account of too large a

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* Stetefeldt based his statements on reports received from Cusihuiriachic, while the works were under the management of Mr. Dagget. Later it developed that the great loss attributed to the burning of the sulphides was caused by the dishonesty of the man in charge of that part of the process, who stole systematically part of each charge.

loss of silver by volatilization. The bullion was very base, and had to be refined either by cupellation or by electrolysis. The method of melting the sulphides and casting them into anodes for electrolytic treatment has not been sufficiently investigated, but most likely it will prove to be the one most suitable.

TROUGH LIXIVIATION.—The principle of trough lixiviation is based on the fact that all soluble salts dissolve quickly and thoroughly in a swiftly moving stream of the solvent. If the solvent is made to flow in a triangular trough with sufficient inclination, and the roasted ore is gradually charged into this stream, the latter will carry the ore along, imparting to the particles a rolling motion. The liquid, however, will move faster than the solid, and the latter will be brought in rapid contact with a large volume of the solvent, and all soluble salts will go in solution in a very short time; in fact, in a fraction of a minute. It is well known that a liquid loses its dissolving energy in proportion as it becomes concentrated with the substance it dissolves. If, therefore, a salt is continually brought in contact with fresh solvent, the latter will act on the salt with its greatest energy. Moreover, the time of dissolving is proportional to the surface the salt offers to the solvent.

Both these principles are highly developed in trough lixiviation, while in tank lixiviation this is not the case. The unfavorable conditions of the latter may be seen if the progress of the process is closely observed. When, after base-metal leaching, a stream of sodium hyposulphite is allowed to enter on top of a charge it takes the solution about one hour to pass through 2 ft. of ore of average filtering capacity and to appear at the outlet. The movement of the solvent through the ore is therefore exceedingly slow as compared with the swift current in a trough. The solvent, by passing the ore particles so slowly on its way down, gradually becomes more concentrated and proportionately loses in dissolving energy; the air, of which more or less is always contained in a charge of ore, even when under prolonged treatment, reduces the surface of contact between the solvent and the ore. The unfavorable conditions for quick solution in tank lixiviation are felt still more with badly filtering ore; in fact, they may prove serious impediments to the success of the treatment.

We have seen from a preceding section of this paper that during base-metal leaching the water becomes so concentrated with chloride salts that silver chloride is dissolved, which necessitates a separate treatment of the base-metal solution. In trough lixiviation a dissolving of silver by the base-metal solution can be entirely avoided. The proportion of water and ore has only to be so regulated that the resulting solution is dilute enough not to dissolve any silver chloride. After the ore has settled and the solution has become clear it can be run to waste without any further loss of silver. This is decidedly a great advantage of trough lixiviation, doing away with the tedious operation of obtaining the silver from the base-metal solution.

In silver leaching in the trough it is also necessary to maintain a certain proportion of ore and solvent, so that the resulting solution will still possess a strong dissolving energy for silver chloride. This proportion varies according to the character of the ore, and has to be ascertained for each ore, lead ores requiring a larger volume of the solvent than other ores. If the right proportion is maintained the silver is almost instantaneously dissolved, taking place while the pulp passes through 12 ft. of trough, or in about 4·70 seconds.

Practice at Cusihuiriachic.—After my paper on "Trough Lixiviation" was read before the American Institute of Mining Engineers at the Boston meeting, February, 1888, and since my extensive experiments on a large scale with the San Francisco del Oro ore at Parral, the results of which were published in the *Engineering and Mining Journal*, February 16, and the following numbers inclusive to March 16, 1889, I had opportunity to apply trough lixiviation on a large scale. The Don Enrique Mining Co. at Cusihuiriachic had accumulated a large dump of second-class ore, which contained about 25 oz. silver per ton, but filtered so badly on account of the large amount of porphyry in the gangue that it was not profitable to work it by tank lixiviation. After the large lixiviation works of this company were destroyed by fire I undertook to work this second-class ore by trough lixiviation, and treated 12,000 tons by this method at a good profit.

Utilizing the results of my experiments at Parral, I adopted short troughs, which enabled me to find enough room and grade in the old North Mexican mill buildings, which were formerly used for tank lixiviation. I constructed two sluicing vats with central discharge and flat, funnel-shaped filter bottoms, which were placed about 10 ft. above the rim of the old leaching vats, of which there were eight, arranged in two rows. Each vat measured 14 ft. in diameter and 3½ ft. in depth. These tanks were used as settling tanks, and were connected each with the other by a 4-in. pipe inserted near the rim, thus forming a circuit of all the vats, so that if one tank was filled the solution could flow into the other, and from there into the next and so on. The filter bottoms of these vats I covered with sheeting and a 4-in. layer of washed river sand. On the cooling floor a small hopper with funnel-shaped bottom was erected and covered with an inclined screen of ⅛-in. mesh, to prevent the lumps from entering. The hopper discharged to a short screw conveyor, and the conveyor into the cups of a belt elevator. The elevator lifted the roasted ore and discharged it into a short triangular trough, in which a stream of water was running. The trough was so arranged that the pulp could be conveyed to either of the two sluicing tanks.

The speed of the screw conveyor being always the same, the feed of roasted ore into the trough was uniform, and the desired proportion of ore and water was easily regulated by the stream of water. When the first vat was filled the pulp was made to enter the second sluicing vat. While the second vat filled the liquid in the first vat became clear and was drawn off by siphons, so that when vat No 2 was full vat No. 1 was ready to receive again the stream of pulp. This was repeated until both vats were fairly filled with washed ore. After the first charge of pulp the outlet under the filter of each vat was opened, discharging a clear stream while filling was in progress. This increased the filling capacity of the vats. When both tanks were charged the solution above the ore was allowed to drain, and then water was applied to displace the solution absorbed by the ore.

The precaution was taken to insert from above the sluicing hose into the central discharge opening before the vats were charged. When the valve below was opened and the pump set to work the sluicing commenced in the center close to the filter, and the stream of sodium hyposulphite soon made there an excavation. By drawing up the hose gradually the ore continued caving until a cylindrical opening through the ore of about 2 ft. in diameter was made. By directing the stream judiciously a pretty constant proportion of ore and solution can be maintained; but this can be accomplished more easily and accurately if the sluicing is done with a small stream, say through a 1-in. hose, and by supplying the deficiency in solution from the upper storage tanks, allowing a stream, which can be regulated, to flow into the trough close to the vat.

The silver leaching troughs were not longer than necessary to reach to each of the eight settling vats. Over the center of each settling vat the trough was intersected by a tight box 18 in. square and 12 in. deep, with a 3-in. round opening in the bottom, which could be closed with a wooden plug. To prevent splashing during charging a 4-in. canvas hose was fastened around the opening in the bottom, reaching down to within a few inches below the rim of the settling vat.

Soon before sluicing commenced the plug in the box above the first tank was removed and a piece of filter-cloth, kept in place by several bricks, was spread directly under the canvas hose, to protect the sand filter. The pulp, dropping always on the same place in the vat, the ore, or rather the residues, because the silver is already extracted when the pulp drops into the vat, will form a cone, which, however, will never project much above the surface of the solution, because the material in the solution, being loose and lighter than the material above the solution, will slide down. Thus a tank of 14 to 16 ft. diameter will be charged pretty evenly. To fill the lower space around the periphery a short trough is placed under the drop, and by changing the position of the trough gradually the stream can be directed to all points. Long before the vat is filled with residues the solution will reach the level of the communicating pipe and flow into the next tank (No. 2) and then into No. 3 and No. 4. When tank No. 2 is filled with solution the outlet under the sand filter is opened and a clear stream of solution will discharge. The same is done when No. 3 is filled with solution, and so on. The filtration was very free and the volume of solution entering vat No. 3 was much reduced, and still more so the overflow into No. 4, so that the solution seldom occupied more than three vats besides the one which was undergoing charging. This gave ample time to treat and discharge the residues, and to renew the filter, if necessary, before the vat was again required in the circuit.

When tank No. 1 was filled sufficiently with residues the stream of pulp was hanged to flow into vat No. 2 by opening the corresponding plug hole and losing the one above No. 1. The communicating pipe between vat No. 1 and No. 2 was closed, and the solution allowed to drain and then displaced by water. The leaching with water has to be continued until the outflowing tream does not show any reaction for silver. Before the pulp was directed to vat No. 2 the outlet under the filter of this vat was closed and not opened again until the vat was partly charged with residues. When vat No. 1 was disconnected from the circuit and No. 2 subjected to the operation of charging, the overflowing solution moved one tank farther and commenced to fill No. 5; when, in course of the operation, the overflowing solution reached vat No. 8, vat No. 1 was empty and prepared to receive the overflowing solution.

Samples of the pulp taken at the drop showed that all the silver chloride was extracted while the pulp was flowing through the trough, and that the ore actually dropped as spent residue into the vat. A similar experience was realized with regard to the base-metal salts. After the pulp (ore and water) passed through the short trough from the elevator to the sluicing vat, I found by a sample taken at the drop that all the heavy metal salts soluble in water had been dissolved, and that only some sodium sulphate still remained. This salt, however, can only be removed from the ore by prolonged leaching, and will be found in like quantity after base-metal leaching in tanks.

The base-metal solution was conveyed to four large tanks in order to prevent the loss of any ore which by careless decantation might have been drawn out by the siphons, or if by want of time (there were only two sluicing vats) the solution had to be decanted before it was perfectly clear. These tanks were not communicating, but each was filled separately to give a better opportunity for settling. When clear the solution was discharged without any further treatment. Regular samples of the base-metal solution were taken, but I never found it to contain any silver.

Advantages of Trough Lixiviation.—It can clearly be seen that trough lixiviation offers many advantages over tank lixiviation, especially in large works, or if badly filtering ore has to be treated. In large works, where 20 or more leaching vats are in operation, each one in a different stage of the process, much attention is required to avoid costly mistakes, while in trough lixiviation care has to be given to only a few tanks. If the ore filters badly it will take a very long time to extract the silver by leaching in tanks, while in troughs the silver as well as the base-metal salts dissolve almost instantaneously, and the effect of the bad filtering will be felt only while the charge is draining and the solution is being displaced by water. It makes the treatment of very badly filtering ore possible, which otherwise could not be treated by lixiviation. The extraction of silver from lead-bearing ores is usually slow and requires a large plant. By trough lixiviation the silver from these ores can be just as quickly extracted as from other ores, and requires only a larger volume of solvent to be used in sluicing. This system is of special advantage when large quantities of ore have to be treated, because it can be done with a comparatively small plant. The system is continuous, and the precipitator receives for precipitation a silver solution of uniform strength, which facilitates quick and exact precipitation. It is strange that so little attention is paid to this method by those engaged in this branch of metallurgy.

THE RUSSELL PROCESS.—The Russell process, which was so elaborately and well written up, and about which so many statements of excellent results were published, and which was in consequence thereof introduced at several places with a large expenditure of capital, has not proved a success. As this process attracted much attention and has found its way into all metallurgical text and hand books, it is interesting and instructive to investigate the cause of the failure.

It was claimed that the extra solution, a solution of a double salt of cuprous hyposulphite and sodium hyposulphite, manufactured by adding a solution of copper sulphate to a solution of sodium hyposulphite, exerted a highly energetic dissolving and decomposing action upon metallic silver, silver sulphide, silver minerals belonging to the group of antimonial and arsenical sulphides, and other silver combinations. Based on this property of his extra solution, he claimed that silver ores treated by his process required a less careful chloridizing roasting, or only an oxidizing roasting, and that even raw sulphuretted ores could be successfully desilverized.

These claims attracted general attention, and if they had been true in the sense in which they were given out this process would have marked a decided step forward in the hydrometallurgy of silver; but they were based on results obtained on a very small scale and under conditions which are very costly and not practicable to create and maintain on a large scale, while the results, even under such conditions, especially with regard to raw ores, proved to be not good enough to justify the application of the process in practical metallurgy. If ½ oz. of raw sulphuretted ore is treated in a beaker with a large excess of a 32% extra solution for 12 hours with frequent stirring, as is done in laboratory experiments, more or less silver will be dissolved. Such a strong solution of sodium hyposulphite will also dissolve some silver, but it is not practicable to work with such concentrated solutions, since they cannot be maintained without a very large consumption of sodium hyposulphite and copper sulphate. Besides, raw sulphuretted ores do not filter well, and the ore would have to be treated in agitating tanks, and the separation of the solution from the residues would have to be effected by means of large filter presses. These manipulations, together with the very large consumption of chemicals, would cost more than roasting with salt. The weakest part of this method, however, is that the extraction at the best is so inferior and incomplete that its application is entirely out of the question.

More favorable results can be obtained by treating raw oxidixed ores, or sulphuretted ores which were first subjected to a thorough oxidizing roasting, wherein there are not so serious difficulties preventing its application on a large scale, though only in exceptional cases will it be rational to employ it. A dilute solution can be used, and often 50 to 70% of the silver will be extracted. But similar results can be obtained by using a solution of sodium hyposulphite. Some of the San Francisco del Oro ore, a highly sulphuretted lead-zinc ore, was roasted oxidizingly and a part treated with sodium hyposulphite and a part with Russell's extra solution. The roasted ore contained 29·3 oz. silver per ton. By leaching with sodium hyposulphite 17·21 oz. silver were extracted, while with Russell's extra solution the extraction gave 17·64 oz. or 0·43 oz. silver per ton more. But an increased extraction of less than half an ounce of silver per ton does not justify the extra consumption of 7 lb. of copper sulphate and 5 lb. of sodium hyposulphite. It would be folly to oxidize an ore for lixiviation and obtain an extraction of only 50 to 70% instead of chloridizing it and obtaining an extraction of 90% and more, especially since by the modern method of chloridizing roasting the loss of silver by volatilization is greatly reduced, and does not much exceed the loss occurring in oxidizing roasting.* The claim that ores require only an oxidizing roasting, if treated by the Russell process, is therefore a mistake, and cannot be verified by actual and satisfactory working results.

I mentioned that only in exceptional cases can lixiviation of raw ores be executed successfully. One of the cases is, when in oxidized ore the silver occurs as chloride. However, it is more advantageous to subject such ore first to a short red heat to melt the silver chloride. Chloride of silver, as it occurs in nature, is very dense and dissolves very slowly, but if the ore is heated the silver chloride will melt and impregnate the surrounding ore particles, in which condition it offers a large surface to the solvent and permits a much quicker extraction. Oxidized ores in which the silver occurs as antimoniate can also be successfully leached. Low grade oxidized ores, which will yield 60 to 70% by leaching raw, can also be successfully treated if, on account of the small tenor of silver in the ore, the increased amount extracted by chloridizing does not exceed the cost of roasting. However, the use of sodium hyposulphite for such ores will be found more economical than the use of the extra solution, the additional cost of chemicals not being covered by the slight gain in extraction.

The claim that silver ores require a less careful chloridizing roasting because the extra solution dissolves the unchloridized part of the silver is also an illusion, as has been demonstrated by many failures on a large scale. The following is a case in support of my assertion:

In Sombrerete, Zacatecas, Mexico, the lixiviation process with sodium hyposulphite was in successful operation for years, until a new company was organized to work the property on a larger scale. A new and large mill was erected to suit the requirements of the Russell process. The success in the old mill was based on a good chloridizing roasting in reverberatory furnaces. The results, however, were entirely different when the new mill was set in operation. In this the ore was roasted in a Stetefeldt furnace, which is not suitable for such a heavy sulphuretted ore, and the chlorination was not satisfactory. This gave an opportunity to demonstrate the claim that the extra solution exerts an energetic dissolving and decomposing action on the unchloridized part of the silver. However, the extra solution failed to react, and after 11 months of unsuccessful trials the company failed and the property changed hands. Though there was an abundance of ore and the mill was arranged for a daily working capacity of 60 tons, during those 11 months not more than 5,777·58 tons of ore were treated, notwithstanding that the ore is quick filtering and permits a quick extraction. The small tonnage was caused by the failure of the Stetefeldt furnace to chloridize well and the failure of the extra solution

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* See my article on "Chloridizing Roasting of Calcareous Silver Ores" in THE MINERAL INDUSTRY, Vol. V., 1896.

to extract the unchloridized silver, which necessitated many trials and endeavors to improve the chlorination. These trials were connected with much loss of time without leading to success. These 5,777 tons of ore contained 191,536·99 oz. silver, while the production amounted only to 111,301·7 oz., which gives an actual working result of not more than 58·1% extraction.

The new company abandoned the Stetefeldt furnace and the Russell process, built a suitable number of reverberatory furnaces and adopted the common lixiviation process with sodium hyposulphite with great success. The mill is still in operation and treats regularly 60 to 80 tons of ore per day.

The Cusi company at Cusihuiriachic, Chihuahua, Mexico, had a similar experience. The old lixiviation process was for years in successful operation on a scale of 50 to 60 tons per day. Induced by the glowing representations of the advantages of the Russell process, the company adopted it, but after 1½ years' trial, with heavy financial loss, the process was discarded and lixiviation with sodium hyposulphite was resumed. Long before the Russell process was adopted in Cusi I tried the extra solution on several tank charges of chloridized Cusi ore, but with no charge did I succeed in extracting more silver than could be extracted with the common solution. The statements of the working results and cost of the Russell process compared with ordinary lixiviation at the Cusi mill as contained in Stetefeldt's book on lixiviation are misleading. It is hard to understand how an ore can be treated cheaper by the Russell process than by the ordinary, when the former embodies the latter from beginning to end and requires an additional application and consumption of blue vitriol and sodium hyposulphite. Still it is stated that the cost of treatment in Cusi was less by $1·29, and the net profit showed 50% in favor of the Russell process. If these statements were true the company would never have discarded the Russell and returned to the ordinary lixiviation process.

The principal claims for the Russell process, viz., that the extra solution exerts a most energetic dissolving and decomposing action upon metallic silver, silver sulphide, silver minerals belonging to the group of antimonial and arsenical sulphides, and other silver combinations, and enables the successful treatment of raw sulphuretted silver ore, have not proved true. In all localities in Mexico where the Russell process was introduced it has had to give way to the ordinary process, and so also has been the experience in New South Wales.

GRAPHITE.

Crystalline graphite was produced in the United States in 1898 at Ticonderoga, N. Y., and in the vicinity of Chester Springs, Chester County, Pa. The reopening of the latter mines led to a considerable increase in the domestic production, while the output at Ticonderoga was itself larger in 1898 than in 1897. Amorphous graphite was produced only in Rhode Island. The production of the so-called "Baraga" graphite of Michigan is not included in our statistics of graphite production, this substance being classed by us with slate pigment.

THE GRAPHITE INDUSTRY IN THE UNITED STATES.

Year.	Production. (a)		Imports.		Consumption.	
	Pounds.	Value. (b)	Pounds.	Value.	Pounds.	Value.
1893	883,912	$39,731	28,882,000	$865,379	29,764,912	$905,110
1894	770,846	34,689	11,640,000	225,720	12,410,846	260,409
1895	377,450	17,286	20,124,160	260,111	20,516,168	277,751
1896	405,006	18,225	33,824,000	437,189	34,229,006	455,414
1897	993,138	44,691	19,113,920	270,952	20,107,058	315,643
1898	1,647,679	148,291	30,199,680	743,820	31,847,359	892,111

(a) Production of refined graphite only; of amorphous graphite there were produced 840 short tons ($4,700) in 1895, 574 short tons ($3,850) in 1896, 1,200 tons ($11,400) in 1897, and 1,200 tons ($11,400) in 1898. (b) Nominal.

Market Conditions.—Owing to the large amount of cast steel, requiring graphite crucibles for its production, used in the manufacture of projectiles during the Spanish war, the domestic consumption of graphite increased heavily. The large demand for this purpose cleared the European market of its supply and taxed the resources of Ceylon to keep up with the current consumption. Prices more than doubled. Plumbago dust sold at 1·4c. per lb. at the beginning of the year and at 3c. at the end. Chips rose from 2@3c. to 4·75@6c., and lump (which is the grade required for crucible purposes) rose from 3·5@4c. to 7·5@9c. It is not believed that the graphite mines of Ceylon, which are controlled by natives, can produce much more mineral than at present. These mines occur in the low, unhealthy part of the island where English people cannot live safely, wherefore the business is allowed to remain exclusively in the hands of the natives.

New York.—The only producer of graphite in this State, and for a long time the only producer in the United States, was the Hooper property (owned by the Joseph Dixon Crucible Co.) which is located about 5 miles west of Lake George, in the town of Hague, Warren County. The graphite is of the foliated variety, occurring in minute scales in the cleavage planes of a seam of gray quartzite. The rock mined contains an average of about 10% graphite, of which only half is won by the process of extraction. There are two mines, both situated on the same bed on different sides of a fault. Above and below the graphite-bearing bed is a laminated garnetiferous gneiss, which is underlain by a massive mica gneiss. Overlying the garnetiferous gneiss is a bed of light-colored quartzite which is entirely free from graphite. The gneiss dips about 20° south. The graphite beds do not conform to this dip, having a low pitch to the east. The graphite-bearing quartzite is from 1 to 5 ft. thick in one mine and from 2 to 15 ft. in the other. The mines are opened by adit levels. The graphite is separated by crushing and washing in a mill near the mines and is further refined at works at Ticonderoga.

The discovery of veins of graphite on Split Rock Mountain, near Essex, was reported late in the year, and it was said that explorations with a view to opening them would be undertaken in 1899.

Pennsylvania.—Prof. T. C. Hopkins contributes the following notes as to the occurrence and production of graphite in the vicinity of Chester Springs, Chester County: "The Philadelphia Graphite Co. owns a mine about one mile east of Chester Springs, which was opened in 1897, and was productive in 1898. The graphite is of the crystalline variety, occurring richly impregnating beds of mica schist, of which the graphite forms about 50%. There are two beds, an upper one about 4 ft. thick and a lower about 6 ft. thick, which are worked through adit levels. The mineral is trammed to a mill nearby, where it is crushed and washed in a log washer, the washed graphite being further ground and screened into different grades. The product in 1898 realized from $75 to $150 per ton.

"At Byers Station, a few miles south of Chester Springs, graphite was mined in 1898 by Pettinos Bros., who do not grind and grade the mineral like the Philadelphia Graphite Co., but simply wash the crude material, which is shipped to Bethlehem for foundry facings. At Pikeland, about a mile north of Chester Springs, graphite was formerly mined in considerable quantities and a good crushing and washing plant was erected there, but it has been idle for several years. The success of the above-mentioned concerns led to considerable prospecting for graphite in the Pickering Valley, and some new discoveries were reported, although none became productive in 1898. The works of Pettinos Bros., at Byers, were destroyed by fire early in January, 1899."

Rhode Island.—The Rhode Island Graphite Co. reported that its production was about the same in 1898 as in 1897. The product was valued at $8 to $10 per ton for lump and $30 per ton for pulverized, f. o. b. at Providence. The company is contemplating an extension of its plant, and if this is made it is expected that between 3,000 and 5,000 tons will be mined in 1899.

South Dakota.—Arrangements were made to open a graphite mine near

Custer, the product of which is to be shipped to Cincinnati for use in foundry facings, etc., but no production was reported in 1898. In the vicinity of Custer there are said to be several promising veins of graphite, in one of which high-grade mineral occurs. In width these veins average about 4 ft. On Castle Creek, in Pennington County, about 25 miles N.W. of Custer, there is a large vein containing about 40% graphite. On one side of this vein there is 10 ft. of graphitic slate. Eight miles N.E. of Custer there are several veins of graphite containing high-grade mineral, which is of the crystalline variety and can be easily separated from its gangue.

Wyoming.—According to Wilbur C. Knight, State Geologist, some progress was made in the development of the graphite deposits near Wheatland, in Hallack Cañon, and it is expected that shipments of the product to Chicago will be made in 1899.

WORLD'S PRODUCTION OF GRAPHITE. (IN METRIC TONS.)

Year.	Austria	Canada	Ceylon.	Germany.	Great Britain.	India.	Italy.	Japan.	Russia.	Spain.	United States. (*b*)	Totals.
1893......	23,807	*Nil.*	21,900	3,140	*Nil.*	(*a*)	1,465	26	311	*Nil.*	400	51,283
1894......	24,121	63	10,718	3,133	*Nil.*	1,623	1,575	1,091	(*a*)	10	350	42,683
1895......	28,443	199	13,711	3,751	41	(*a*)	2,657	77	(*a*)	*Nil.*	171	49,050
1896......	35,972	126	10,463	5,248	*Nil.*	(*a*)	3,148	215	(*c*)	*Nil.*	184	55,356
1897......	38,504	396	(*c*)	3,861	*Nil.*	61	5,650	(*c*)	(*c*)	(*a*)	450	48,922

(*a*) Not reported in the government statistics. (*b*) Crystalline graphite. (*c*) Statistics not yet published.

Bavaria and Bohemia.—According to Dr. Weinschenk, quoted again further on, the graphite deposits of Bavaria are found N.E. of Passau, near the Austrian frontier, the gneisses of the Bayerischer Wald being rich in occurrences of graphite. In payable quantities, however, the mineral is restricted to a comparatively small area. Approaching the frontier granite takes the place of the gneiss, and simultaneously with this change the graphite disappears, reappearing further N.E. in the gneisses around Schwarzbach in Bohemia. Thence following generally a northeasterly strike, the occurrences are traced in varying number and importance as far as Krumau. The Passau graphite occurs in large scales, easily separable from the matrix, and in its refined condition furnishes excellent material for crucibles of the best quality. On the other hand, the Bohemian graphite occurs either in very minute scales or in a compact form, and is in part applied to the manufacture of pencils, in part to that of crucibles, but the latter are of an inferior quality to those of Passau.

The methods of refining vary in the two districts. In Passau, the graphite being scarcely at all brittle, the method is to crush the whole of the graphite-bearing rock, so that the brittle constituents are reduced to a fine dust, while the graphite scales remain unchanged. The dust is then blown or sifted out, and the mass of material left behind contains about 90% of graphite, the remaining impurities being chiefly mica minerals; but all the constituents which would endanger the refractory qualities of the graphite are found to have been removed.

The more compact Bohemian graphite suitable for pencil manufacture is marketed without treatment, but the less pure, dense varieties are ground ine, and by settling processes freed from the heavy mineral constituents. The levigated graphite is accumulated in settling tanks, from which the sediment is transferred to filter presses and squeezed under 88 lb. per sq. in. The resulting cakes, still containing upward of 20% water, are dried for 24 hours at 90 to 100° C. The refined graphite is not very much richer in carbon than the crude mineral, which assays 50 to 60%.

The country in the neighborhood of Passau is very rough and transportation, which is done at present by wagon, is costly, wherefore the mines are not exploited so extensively as might be done under better conditions. The connection of Passau by railway with the main lines of Bavaria is now contemplated, although no definite plans have yet been formulated. In Bohemia there are three graphite mines in operation, one near Schwarzbach, owned by Count Schwarzenburg; one near Murgan, owned by peasants, and one near Krumau. The mineral of the last place is of inferior quality. The larger part of the production of Bohemia is exported.

Canada.—At the present time graphite is produced only in Ontario, where a deposit in Brougham township, Renfrew County, which was discovered in 1895, is being operated by the Ontario Graphite Co. This deposit exists in the form of a bed from 8 to 10 ft. in thickness in crystalline limestone. The mineral is for the more part amorphous, but a flaky structure is observable in certain portions of the deposit. The mine is situated about 12 miles from the railway, at Calabogi. Graphite mines have been from time to time exploited in Ottawa County, Quebec; but apparently with poor success, since the Walker Mining Co. closed down in July, 1896, and the North American Graphite Co. suspended work in July, 1897. In New Brunswick work has been carried on at intervals near St. John, where graphite in a finely divided state is disseminated through the country rock, and at a few points is found in beds of economic importance.

Artificial Graphite.—An important event in the graphite industry of the United States in 1898 was the manufacture of an artificial graphite direct from coke, which it is thought can be put on the market at a price to compete with the natural product. During the year about 200,000 lb. of carbon rods were graphitized for electrolytic alkali manufacturers in the United States, England and Germany. This reversed the direction of the trade in carbon electrodes, Europe now buying in America, whereas formerly America bought in Europe. It is anticipated that in the near future nearly all electric motor brushes will be made out of this artificial graphite, which has been found to possess not only high lubricating qualities but also an exceedingly high conductivity. The artificial graphite can also be substituted for the natural product in the manufacture of lead pencils, stove polish and paint. This business was inaugurated by the Carborundum Co., operating under patents* granted to E. G. Acheson, but now it has been taken over by the Acheson Graphite Co., which intends to erect works at Niagara Falls, N. Y.

According to the Acheson patents, carbon in the form of coke, charcoal or

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* United States Patents Nos. 542,982, 568,323 and 617,979, covering the purification of carbon and the manufacture of graphite and articles thereof.

lampblack is granulated, mixed with an oxide or metallic salt, such as oxide o sulphate of iron, moistened with water containing a little sugar or other binding material in solution, and pressed into the desired shape. These shapes are then placed in an electric furnace, embedded in fine carbon, which is surrounded by a layer of amorphous carborundum, sufficiently thick to prevent the rapid radiation of heat. Current is turned on the furnace and under the influence of the high temperature the carbon is more or less completely graphitized; it has been found advisable to leave a portion of the carbon unconverted, since this tends to make the article stronger.

The manufacture of carbons for electrical purposes in the United States has passed into the hands of a combination known as the National Carbon Co., incorporated under the laws of New Jersey with $10,000,000 capital stock, of which $4,500,000 is preferred 7% stock, and $5,500,000 common stock. This company took over the business of the National Carbon Co. of Cleveland; the American Carbon Co. of Noblesville, Ind.; the Faraday Carbon Co. of Jeannette, Pa.; Crouse, Tremaine & Co. of Fostoria, Ohio; the Washington Carbon Co. of Washington, Pa.; the Phœnix Carbon Co. of St. Louis; the Partridge Carbon Co. of Sandusky; the Solar Carbon Co., near Pittsburg; Brush Carbon Co., Standard Carbon Co.; Thompson-Houston Carbon Co., Fremont Ohio; and the Globe Carbon Co. of Ravenna. In 1898 the National Carbon Co. is said to have made an output of 120,000,000 carbons, and the other companies enough to give a total of 200,000,000 carbons. The United States Carbon Co., of Cleveland, Ohio, did not go into the consolidation. Owing to the prosperity in the electric business in 1898, and the increase in electric lighting, electro-chemical work, etc., the business in carbons was very active and further improvement is looked for in 1899.

GEOLOGICAL OCCURRENCE OF GRAPHITE.

Dr. Ernst Weinschenk* considers that all graphite deposits are divisible into two great groups, and these again into several sub-divisions according to type localities. The following is a summary of his classification:

I. *Vein-like Occurrences.*—These, generally speaking, are the purer of the two, and therefore of considerably greater industrial importance.

(*a*) *Ceylon.*—Veins of scaly or fibrous graphite run through granite or closely associated rocks. The country rock is often highly decomposed, and then consists mainly of kaolin and similar decomposition products.

(*b*) *Borrowdale* (England).—Fine-scale graphite in veins in "greenstone porphyry." The gangue material is chiefly calcspar, brownspar and quartz, wherein are nests and lumps of very fine graphite, especially suitable for the manufacture of pencils.

(*c*) *Batugol* (Province of Irkutsk, Siberia).—Finely fibrous graphite, purer than that of Borrowdale. The veins run through a granitic or dioritic rock,

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* Der Graphit, seine Wichtigsten Vorkommnisse und seine Technische Verwerthung, *Sammlung gemeinverstaendlicher wissenschaftlicher Vortraege*, new Vol. XIII., p. 295, Hamburg, 1898; Zur Kenntniss der Graphitlagerstaetten *Abhandlungen der Mathematisch-Physikalischen Classe der Königlichen Bayerischen Akademie der Wissenschaften*, 1898, Vol. XIX., pp. 509–564.

while in the closely adjoining limestone (altered by contact metamorphism) are great lumps of pure graphite, suitable for pencils only.

II. *Bed-like Occurrences.*—(*a*) Passau.—In the easternmost corner of Bavaria, bounded on the south by the Danube, and by the Austrian frontier on the east, are gneisses and gneissose rocks impregnated with scaly graphite, which at times appears as lenticular masses of rich mineral. These occur chiefly in the immediate neighborhood of intercalations of granular limestone, altered by contact metamorphism. Both the graphite-bearing rock and its near neighbors are highly decomposed, so that kaolin and other decomposition products are found in intimate association with the graphite deposits. The genetic connection of these with the Ceylon type is very close.

(*b*) *Schwarzbach-Krumau* (Bohemia).—This appears to be the most widespread of any. The lenticular form of the deposits, their geological relationship with limestone intercalations, their frequent association with kaolin and other decomposition products, connect them closely with the Passau type, from which they are differentiated by the more compact, less crystalline character of the graphite.

(*c*) *Paltenthal-Kaisersberg.*—On the northern border of the central zone of the Styrian Alps is a highly metamorphosed system of Carboniferous shales, clay-slates, limestones, and conglomerates with coal seams, the coal of which has passed into graphite, which preserves completely, in some cases, the appearance of the coal from which it is derived. It is very compact, very pure, and often extremely hard. It is used for crucibles.

GYPSUM.

The production of gypsum in the United States is steadily increasing, especially in the Western States. There is a considerable production of this rock in New York and Virginia, but the Atlantic coast derives its supply chiefly from Nova Scotia and New Brunswick. A noteworthy feature of the gypsum industry in recent years is the increasing proportion of the product converted into stucco or plaster of paris.

The total production of crude gypsum in 1898 was 348,686 tons, against 300,369 in 1897. Of the product in 1898 about 287,581 tons were calcined, and of that in 1897 about 248,630, the yield of plaster being about 80%. Crude gypsum is worth about $0·90 per ton at the quarries in New York and $1·25 in Michigan. Ground gypsum for land plaster sells for about $2 per ton in New York. Calcined gypsum varies from $3 to $5 per ton in the Eastern and Central States; in Colorado, Utah and California it is worth about $8 per ton. Prices were generally lower in 1898 than in 1897, the average for the whole product having been $3·81 against $4·05 in the previous year. The total value of the gypsum and plaster production as marketed in 1898 was $989,589, against $889,177 in 1897.

PRODUCTION OF GYPSUM IN THE UNITED STATES. (*a*)
(In tons of 2,000 lb.)

States.	1896.	1897.	1898.	States.	1896.	1897.	1898.
California	1,638	2,750	3,875	Oregon	1,800	*Nil.*	150
Colorado	1,600	1,190	1,570	South Dakota	7,825	6,000	8,750
Ind. Ter. and Oklahoma	11,736	11,120	15,229	Texas	9,965	17,161	24,417
Iowa	18,361	24,478	38,338	Utah	1,200	2,500	*e* 3,000
Kansas	49,500	62,556	49,720	Virginia	5,444	7,150	8,125
Michigan	68,180	95,040	*e* 100,000	Wyoming	*Nil.*	3,875	30,635
Montana	385	425	400				
New York	22,923	47,164	46,477	Total	221,898	300,369	348,686
Ohio	21,341	*e* 19,000	*e* 23,000				

(*a*) Statistics reporting the amount quarried. (*e*) Estimated.

GYPSUM IMPORTED INTO THE UNITED STATES. (IN METRIC TONS.)

Year.	Ground or Calcined.			Unground.			Value of Manufactured Plaster of Paris.	Total.
	Quantity.	Value.	Per Metric Ton.	Quantity.	Value.	Per Metric Ton.		
1893	3,417	$31,670	9·26	166,929	$180,254	$1·08	....	211,924
1894	2,050	16,823	8·17	165,100	179,237	1·08	....	196,060
1895	3,348	21,526	6·42	195,630	215,705	1·10	$10,352	247,583
1896	3,348	22,058	6·58	183,165	193,549	1·06	11,722	227,329
1897	2,707	17,028	6·29	165,812	178,686	1·08	16,715	212,429
1898	3,021	18,500	6·12	168,723	181,364	1·09	40,979	240,843

TECHNOLOGY OF GYPSUM.*

BY G. P. GRIMSLEY.

THE composition and properties of gypsum were first investigated in 1765, when Lavoisier found that it was composed of lime and sulphuric acid. Later the theoretical composition was determined to be 46·6% sulphuric acid, 32·5% lime, and 20·9% water, corresponding to the formula $CaSO_4+2H_2O$. Lavoisier determined that when gypsum was heated the water was removed at two different stages. This result was confirmed by Payen, and especially by Le Chatelier. The latter found that on heating gypsum up to 200° C. there was a constant rise in temperature with two exceptions. The first halt was at 128° C., and the second at 163° C. These halts were regarded as due to absorption of heat at the time the water was removed. Gypsum heated above the 128° C. temperature showed a composition of 6·7% water and 93·3% calcium sulphate, corresponding nearly to the formula $(CaSO_4)_2+H_2O$. A series of experiments showed that the loss in weight in gypsum heated to the above temperature was equal to one and one-half equivalents of water. The ordinary plaster of paris of commerce contains about 7% of water, and so has the formula given above, *i.e.*, $(CaSO_4)_2+H_2O$. No change in composition was detected after the second halt at 163° C., and the change at this point was probably physical rather than chemical. No further change was noted in the plaster with increased heat, except that it set much more slowly until about 360° C., when it was found to be overburned or dead burned, and it did not harden or set when water was added. The temperature of burning is thus shown to be a very important point in plaster manufacture.

FIG. 1.—EARLY TYPE OF GYPSUM KILN.

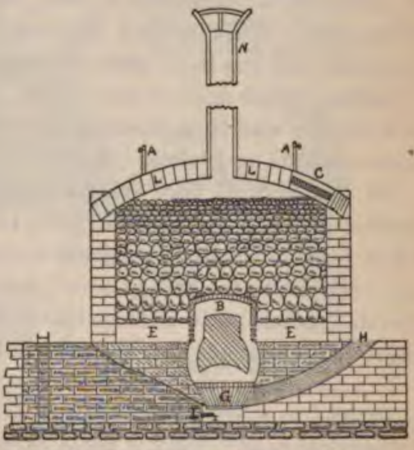

FIG. 2.—DUMESNIL'S GYPSUM KILN.

Process of Plaster Manufacture.—In the early methods of the manufacture of plaster of paris the gypsum was quarried and burned directly in kilns of simple pattern. One of the early types is illustrated in section in Fig. 1.† The

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* Published by permission of the University Geological Survey of Kansas.

† Taken from *Wagner's Chemical Technology*, 1889.

walls (AA) are constructed of brick or stone and connected by a low arch openings through it (aa) for ventilation. The gypsum blocks were place this kiln through the door at R. At the bottom were placed a series of s chambers (xx) for the fuel. Very often these fire boxes or flues were structed of the larger blocks of gypsum. Wood was used as fuel, and vapors and flames passed upward between the blocks of gypsum and through the openings at aa.

Later Dumesnil constructed an improved kiln, which gave better contr the heat. In this type, shown in Fig. 2,* there is a central firepit (G) with chamber (B) above, which is connected with radiating flues (EE), constructe the larger fragments of gypsum rock. Above these flues the stone is arrange layers containing smaller and still smaller fragments toward the top. In arch (L) forming the top of the kiln are flues (A, A, N), controlled by da ers. The material is introduced through the opening at C, and rem through a door at the side. Coal is used as fuel and added at H. The ash is located at I. The kiln is 20 ft. in diameter and 13 ft. in height to the of the arch. This method of burning is said to be economical and unifor its work, and has been used especially in Europe.

In the plaster mills of the United States the gypsum rock is usually gro to a powder and then calcined in kettles. This method is thought to gi more uniform and better quality of plaster.

On the ground floor of the mill are placed the crusher and nipper. crusher reduces the lumps of rock to 3 in. or 4 in. size, and these pieces taken by the nipper or cracker, which is set in the floor just under the crus and reduced to the size of small gravel. This falls into the buckets of a c elevator, whereby it is raised to a bin on the second floor, whence it pa through a spout into an ordinary buhrstone mill, where it is ground to pow

In several of the Michigan mills buhrstones are not used, and the gyp is ground to powder by revolving cages (Fig. 3), which are provided with cylindrical bars. Two of these cages fit together, one inside of the other, revolve at high speed in opposite directions. The gypsum is fed into center and is carried by centrifugal force between the bars, which beat material to powder by friction of one particle against another and against bars. The advantages claimed for this machine are: no choking or clog of the machine, larger capacity for a given horse power, and small dange breakage.

From the buhrstone mill, or the disintegrator last described, the sum flour passes into another chain elevator, and is carried to the to the second story of the mill into the storage bin, located just over kettle. It is thence run into the calcining kettle, which has a tem ature of about 212° F. or higher, about 1·5 hours being required t this to a depth of 5 ft. The calcining kettle is shown in Fig. 4. Coal is as fuel. The heat passes under the kettle proper and upon one side, th through the flues, two or four in number, as shown in the horizontal sec on the line AB, and out on the other side through the chimney. The d

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* From *Redgrave's Calcareous Cements*, 1895.

tions are shown by the arrows in the vertical sections. The bottom of the kettle directly above the firebox is made of cast iron or steel. It is convex in shape, and can be removed when it has burned through. In some cases this bottom is made in sections, which may be renewed part at a time. The average kettle is 8 ft. in diameter and 6 to 8 ft. deep, constructed of iron and surrounded by a wall of stone 2 ft. thick, with an air space between the wall and the kettle.

Such a kettle holds 7½ tons of ground rock. This amount can be calcined in three hours. The rock loses about one-fourth of its weight by expulsion of water, which passes out through the vapor stack connected with the kettle. Consequently there remains about 6 tons of plaster. During the process of calcination the whole mass is stirred by horizontal revolving arms attached to the vertical shaft shown in the accompanying engravings. This stirrer makes about 15 revolutions per minute, and serves to keep the plaster from burning on the bottom of the kettle.

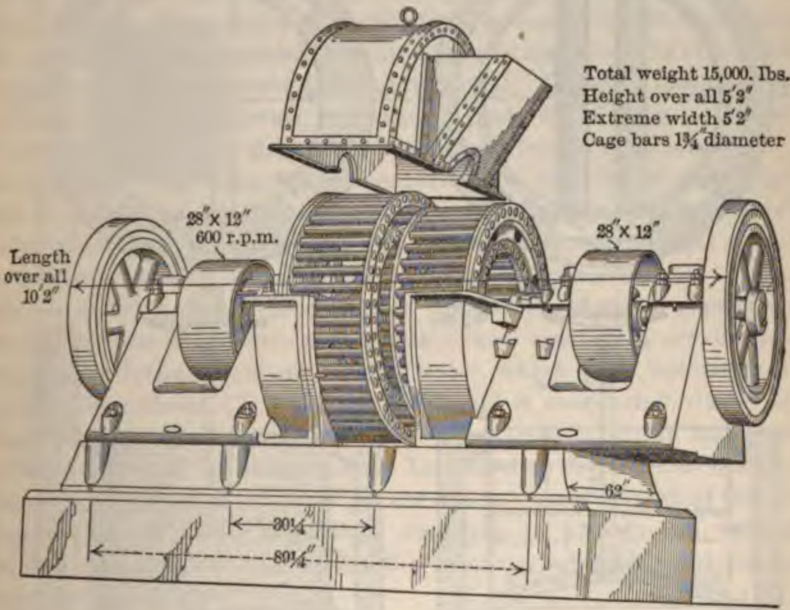

FIG. 3.—THE STEDMAN DISINTEGRATOR.

In an hour after the kettle is filled the temperature reaches 230° F. (110° C.), and the mass is seen to be boiling vigorously as the water is driven off. When the temperature increases to about 270° F. (132° C.) the material settles down solidly, leaving 12 to 16 in. of vacant space at the top, and the steam ceases to rise. This temperature agrees very closely with that determined by Le Chatelier in the laboratory as the first halt in the rising temperature when gypsum is heated.

At about 290° F. (143° C.) the mass boils up again, throwing part of the material over the edge of the kettle, and when a temperature of near 340° F. (171° C.) is reached the plaster is rapidly withdrawn through a gate near the kettle bottom, as shown in Fig. 4, and is run into a fireproof bin on the ground. The kettle is then refilled. The whole process takes about two and one-half hours, and three kettles are usually burned in a day, requiring about 1,400 lb. of the best coal. In some parts of the country oil is used as fuel with considerable saving of labor.

The cost of manufacture of cement plaster will come under $3 per ton—80c. for mining, 30c. for fuel, 50c. for labor at mill and $1·25 for office force and agents, making a total cost (estimated) of $2·85 per ton.

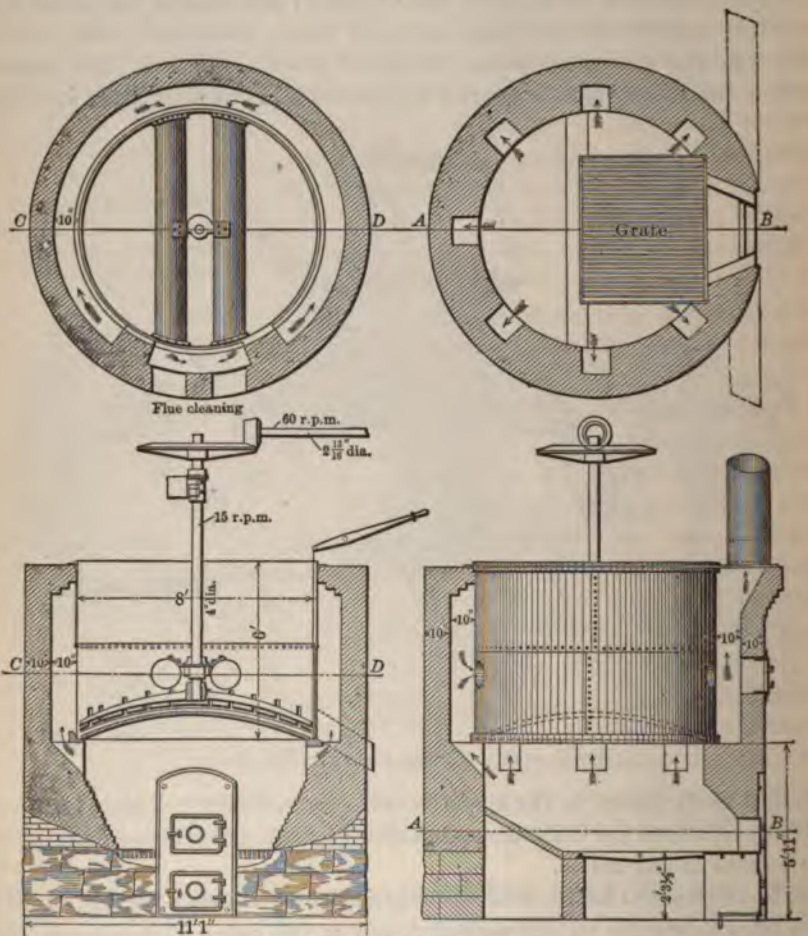

FIG. 4.—GYPSUM CALCINING KETTLE.

During the process of calcination some manufacturers add a "retarder" about one-half hour before the operation is completed, in order to hold back the time of setting. Ordinary gypsum plaster will set in six to ten minutes, and retarders are added in sufficient amount to hold back the set from two to six hours, or in extreme cases for 24 hours. In the early history of the plaster industry glue water was added by the workmen, but this was troublesome, and poor work often resulted from poor mixing of the materials. At the present time patent retarders are sold and used under various names. These are added in the kettle and thoroughly mixed through the action of the revolving stirrers. Sorghum, glue, dextrine, tankage from packing houses, sodium carbonate and other materials are used in these retarders. The exact influence of such mixtures is not known, but they seem to hinder in some way the rapid crystallization of the plaster. Whether this weakens the plaster or not is a disputed question among architects and plasterers. However, in all probability a good retarder does not injure the plaster. These retarded plasters and those naturally retarded through their chemical components are sold under the name of "cement wall plaster."

After the hot plaster passes from the kettle to the floor bin it remains about an hour to cool partially, and then it is raised to the second story. It finally passes down into a horizontal cylindrical trommel, 40 in. in diameter and 10 ft. long, slanting downward $\frac{3}{8}$ in. to the foot, covered with 40-mesh brass wire cloth. The screenings remaining behind, averaging about 1% of the total, are carried back to the buhrstone mill and reground. The fine plaster is run into 100-lb. sacks or 250-lb. barrels, and then is ready for shipment. When water is added to these plasters they unite with it and form a solid mass.

Set of Plaster.—Lavoisier, in his experiments in 1765, showed the set of plaster of paris to be due to a process of crystallization. These experiments (microscopical) have been repeated and verified a number of times since. The process of set, however, still remains unknown to most persons.

Landrin, in 1874, noted three steps in the process of set of plaster: (1) The plaster on contact with water unites with it and assumes a crystalline form; (2) the plaster dissolves partially in the water, which then becomes saturated with calcium sulphate; (3) a part of the liquid is evaporated, due to the heat set free in the chemical combination. A crystal is then formed and determines the crystallization of the entire mass. Le Chatelier showed that plaster would set in a vacuum and that evaporation was not an essential part of the process. He argued that the plaster of paris is dissolved in part in the water, which diminishes the solubility and makes the solution supersaturated, so that the gypsum crystallizes out.

So far as I have been able to determine, these are the only attempts made to explain the set of plaster. My own experiments in part agree with those given; and the set of plaster is undoubtedly a process of crystallization, but just what determines or starts the crystallization is the difficult part of the problem. Microscopical study shows that the process of burning gypsum not only changes the chemical composition but also the physical character. After calcination the irregular particles of ground gypsum are broken into fine granules of nearly uniform size and shape. As the material is heated the water in the gypsum is changed to steam, and expanding, breaks the crystals into finer particles.

As we have seen, Landrin shows that secondary crystals are formed by evaporation of the liquid through heat set free by chemical combination; while Le Chatelier regards the solution as supersaturated by dissolving part of the gypsum. Now while the gypsum particles are broken by the expanding steam it is possible and very probable that some very fine particles remain as true crystals, and would serve as centers for crystallization in the solution of water and plaster of paris, just as a crystal of alum will start crystallization in a saturated alum solution. Crystallization once started would go on rapidly, and the set plaster is a crystalline network. Retarders added interfere with the formation of the network and delay it.

Under the microscope, when water is added to the calcined plaster, small needle-like prisms are seen forming. As these become more abundant they unite and form a more or less solid mass. The change is shown in Fig. 5. Chemical analysis shows the set plaster to be of nearly the same composition as the original gypsum rock.

Hardening of Cement Plasters.—It is well known that if gypsum is heated and placed in a solution of alum, and then again heated to redness, the plaster on setting is very much harder than the ordinary plaster. Payen thought the hardening was due to the formation of a double sulphate of potash and lime. Landrin found by chemical analysis that there was not even a trace of alumina or potash in these plasters. He regarded the hardening as due to the action of aluminum sulphate and potassium sulphate on the gypsum, whereby all the calcium carbonate was changed into sulphate, giving the alum plasters their great purity. In place of alum, borax, sulphuric acid and other substances have been used. Such plasters command a higher market price and are sold under various patent names.

FIG. 5.—PLASTER CRYSTALS FORMING.

Hardening of Gypsum Rock.—Many attempts have been made with more or less success to change the soft gypsum rock into a hard rock resembling marble or chalcedony. The hardened gypsum is polished and used for ornamental work in place of marble. Up to the present time this work has not attracted very much attention, but now a company is organized to manufacture this product in Colorado under a secret process. The gypsum so hardened shows an increased weight, and is compact and takes a fine polish. The banded gypsum gives onyx effects, and the promoters of this work claim their product is much cheaper than onyx or marble. The patented processes consist of drying the rock by hot air to eliminate all moisture and then treating with acids and aluminum sulphate solutions. The more recent work, however, is not patented, and no facts are given out by those engaged in the manufacture. It seems asif there might be a good field for this new branch of the gypsum industry.

IRON AND STEEL.

THE same conditions which made 1897 a year of activity in the iron trade all over the world were intensified in 1898, especially in the United States, which had not only to supply an enormous domestic demand, but also a large amount for export to make up for deficiencies in Europe. This happened both in the trade in raw iron and steel and in that of finished products of all kinds. The American production was the largest on record, and during the closing months of the year the capacity of furnaces and mills was taxed to the utmost. It is difficult to specify any single source of demand which calls for the great increase in production, there being general prosperity in nearly all branches of industry which require iron and steel. The railways were large buyers, but their demand is no longer the predominating influence in the trade that it once was. The increase in new mileage was not large, but the consumption of steel for this purpose is much less in importance than the consumption for renewals; the importance of the demand for the latter purpose is increased, moreover, by the substitution of heavier rails; only a few years ago 56-lb. rails were considered heavy, while now 80 and 85-lb. rails are common; some roads are using 100-lb., and 115-lb. rails have been rolled.

The year 1898 was remarkable for the large consolidations of business interests in the iron and steel industry. The Federal Steel Co. was formed by the combination of the Illinois Steel Co., which owned works at Milwaukee, Wis., South Chicago and Joliet, Ill.; the Minnesota Iron Co., which owned mines on the Vermilion and Mesabi ranges in Minnesota; and the Elgin, Joliet & Eastern Railroad, a belt line around the city of Chicago, constructed chiefly to serve the different works of the Illinois Steel Co. The Federal Steel Co. also purchased the plant of the Lorain Steel Co. at Lorain, Ohio, which included extensive works built by the Johnson Steel Co. and two large blast furnaces now under construction. The Lorain works are a type of the latest and most improved design, and are excellently situated for competition in the Eastern market. The American Steel and Wire Co. took over nearly all the important wire-making works of the United States, including the plants of the Washburn & Moen Manufacturing Co. at Worcester, Mass., and Waukegan, Ill., and that of the Cleveland Rolling Mill Co., of Cleveland, Ohio. The American Tin Plate Co., which is referred to especially under the caption "Tin," elsewhere in this volume, acquired practically all the tin-plate works of the United States. An attempt to form a steel rail combination failed through the refusal of the Carnegie Steel Co. to enter into it, and the announcement of this failure was followed by the placing of a large tonnage of steel rails for delivery in 1899 at $17@$18. Early in 1898 the independent furnaces in the Pittsburg district and the Shenango and Mahoning valleys, which make Bessemer pig iron, formed a combination under the name of the Bessemer Association. These furnaces have occupied a somewhat precarious position heretofore, owing to the tendency of the large steel makers to build their own blast furnaces and control their own supply of pig iron.

Iron Ore.—The American production of iron ore did not increase in proportion to that of pig iron, partly on account of the use of a larger proportion of high-grade ores and partly on account of reduction in stocks on hand. The supremacy of Lake Superior iron ore was emphasized still more decidedly, 75% of the make of pig iron in 1898 being from these ores. The only other single important source of supply was the Southern States, whence 16·5% of the pig-iron product was derived. Eastern Ohio and other local ores supplied 7·5%, while the make of pig iron from imported ores amounted to only 1% of the total. Statistics of the iron ore used as a flux in silver-lead smelting are not obtainable.

IRON ORE MINED AND CONSUMED IN THE UNITED STATES.

District.	1897.	1898.
Lake Superior	11,500,667	14,029,683
Southern States	4,283,700	4,980,000
Other States	1,537,600	1,678,500
Total	17,321,967	20,798,183
Add increase in stocks	995,000	None.
Total mined in United States	18,316,967	20,798,183
Add decrease in stocks	....	787,348
Add importations	489,970	187,219
Total consumed	17,811,937	21,772,750

The tendency to consolidation and ownership of the mines by the large steel-producing companies continued, the most noteworthy instance in 1898 being the purchase of the Norrie and other Gogebic mines by the Oliver Iron Co., which is a branch of the Carnegie Steel Co. This company also extended its interests in the Mesabi range. In the South the most important event of the year was the opening of the great brown ore deposits at Leeds, in Alabama, which are to be worked on an extensive scale, chiefly for the manufacture of basic pig iron for the new steel works at Ensley, Ala. In the East there were no changes of importance. The imports of iron ore in 1898 amounted to only 187,219 long tons, against 489,970 in 1897, the principal reason for the falling off being the closing of the Cuban mines on account of the war with Spain. There was some talk of importing iron ore from Newfoundland, where extensive beds of high-grade Bessemer ore are said to exist, but nothing was done beyond the shipment of one or two trial cargoes.

Pig Iron.—The details of the production of pig iron, which in 1898 was the largest in the history of the United States, are shown in the subjoined table. With respect to the first of the series, attention should be directed, as heretofore, to the fact that the division in the make of pig iron according to the kind of fuel used is no longer satisfactory, nearly all of the furnaces which use anthracite coal employing a certain proportion of coke mixed therewith; probably it will not be many years before the use of anthracite will be discontinued. A more satisfactory division is according to the kind of iron made, which appears in the second table. This table shows the large proportion of the American make of pig iron which is intended for the production of steel; the proportion will probably be increased still more when the Southern steel works are in full operation.

PRODUCTION OF PIG IRON IN THE UNITED STATES. (IN LONG TONS.)

According to Fuel Used.	1896.	1897.	1898.
Anthracite	1,146,412	932,777	1,203,273
Coke	7,166.471	8,464.692	10,273.911
Charcoal	310,244	255,211	296,750
Totals	8,623,127	9,652,680	11,773,934

Kind of Iron.	1896.		1897.		1898.	
	Tons.	%	Tons.	%	Tons.	%
Foundry and forge iron	3,499.899	40·6	3,127,010	32·4	3,437,337	29·2
Bessemer pig	4,654,955	54·0	5,795,584	60·0	7,337,381	62·3
Bassic pig	336.403	3·9	556,391	5·8	785,444	6·7
Spiegeleisen and ferro-manganese	131,940	1·5	173,695	1·8	213,769	1·8
Totals	8,623,127	100·0	9,652,680	100·0	11,773,934	100·0

States.	1893.	1894.	1895.	1896.	1897.	1898.
Alabama	726,888	592,392	854,667	922,170	947,831	1,033,676
Colorado	45,555	73,669	58,508	45,104	6,582	(*a*) 141,010
Connecticut	12,478	7,416	5,615	10,187	8,336	6,336
Georgia	39,675	40,268	31,034	15,593	17,092	13,762
Indiana	5,567	....	....	....	....	....
Illinois	405,261	604,795	1,006,091	925,239	1,117,239	1,365,898
Kentucky	47,501	33,854	63,780	70,660	35,809	100,724
Maryland	151,773	5,600	10,916	79,472	193,702	190,974
Massachusetts	7,853	156	4,710	1,873	3,284	3,661
Michigan	117,538	95,171	91,222	149,511	182,578	147,640
Minnesota	10,373	....	....	....	....	....
Missouri	32,360	6,522	27,518	12,548	23,883	(*b*)
New Jersey	74,305	63,273	55,502	59,153	95,696	100,681
New York	191,115	175,185	181,702	206,075	243,304	228,011
North Carolina	2,843	....	323	2,151	....	....
Ohio	875,265	900,029	1,463,789	1,196,326	1,372,889	1,986,358
Oregon	4,739	1,000	....	....	....	....
Pennsylvania	3,643,022	3,370,152	4701,163	4,024,166	4,631,634	5.537,832
Tennessee	207,915	212.773	248,129	248,338	272.130	263.439
Texas	6,257	4,671	4,682	1,221	6,175	5,178
Virginia	302,856	298,086	346,589	386.277	307,610	283,274
West Virginia	81,591	80,781	111,968	108,569	132,907	192,699
Wisconsin	181,772	91,595	148,400	158,484	103,909	172,781
Totals	7,124,502	6,657,388	9,446,308	8,623,127	9,652,680	11,773,934

(*a*) Production of Missouri included. (*b*) Production included in Colorado figures.

The monthly production of pig iron did not vary much in 1898. January opened with a capacity of furnaces in blast reported at 227,150 tons a week, which gradually increased to 235,350 tons in March, the maximum for the first half of the year. A fear of over-production then arose, in view of the continuance of low prices, and some furnaces were blown out, with the result that the production fell off slowly until August, when a minimum of 208,150 tons was reached. It rose slowly in September and October, and then more rapidly, as stocks of unsold iron diminished and the consumptive demand increased, until at the end of the year it amounted to 238,000 tons a week. With the last figure we probably approached nearly the limit of production, for although there were at that time a considerable number of furnaces out of blast, the more part of these were old furnaces of small size or obsolete type, which cannot be operated at a profit, and will probably never be blown in again, except when a large advance in price may give them a fitful period of prosperity. The productive capacity of the United States is being increased, however, by the construction of several modern furnaces of huge size, which when completed will probably enable us to turn out 15,000,000 to 16,000,000 tons per annum if the demand requires it. The consumption of pig iron in the United States in 1898 was about the same as the production; the exports amounted to 253,077 long tons and the imports to 25,137, but the difference was fully offset by the reduction in stocks on hand, which were 240,000 tons less on December 31 than at the beginning of the year.

PRODUCTION OF STEEL IN THE UNITED STATES. (IN TONS OF 2,240 LB.)

Kinds.	1894.	1895.	1896.	1897.	1898.
Bessemer	3,571,313	4,909,128	3,919,906	5,475,315	6,609,017
Open-hearth	784,936	1,137,182	1,298,700	1,631,843	2,230,292
Crucible	51,702	67,666	62,500	65,600	89,747
Miscellaneous	4031	858	900	1,750	3,801
Total tons	4,412,032	6,114,834	5,582,606	7,174,508	8,932,857
Total metric tons	4,482,592	6,212,671	5,366,518	7,289,300	9,075,783

The production of Bessemer steel increased 1,133,702 tons, or 20%, in 1898, as compared with the previous year. In 1898 Pennsylvania made 57·5% of the total; Ohio, 22·5%; and Illinois, 16·7%. These three States made 90·7% of all the Bessemer steel, only 9·3% being made in other States. The production of all kinds of Bessemer steel rails by the producers of Bessemer steel ingots in 1898 was 1,955,427 long tons, against a similar production in 1897 of 1,614,399 and 1,102,892 in 1896. The maximum production of Bessemer steel rails by the producers of Bessemer steel ingots was reached in 1887, when 2,044,819 tons were made. The following table shows the production of rails as compared with that of Bessemer steel ingots for six years past:

Year.	Tons Ingots	Tons Rails.	Per Cent.	Year	Tons Ingots	Tons Rails.	Per Cent.
1893	3,215,686	1,036,353	32·2	1896	3,919,906	1,102,892	28·1
1894	3,571,313	904,020	25·3	1897	5,475,315	1,614,399	29·4
1895	4,909,128	1,266,081	25·8	1898	6,609,017	1,955,427	29·6

The American Iron and Steel Association report separated for 1897, for the first time, the production of Bessemer steel rails weighing 45 lb. and less than 85 lb. to the yard from those weighing less than 45 lb. and over 85 lb. This separation continued for 1898, as follows:

	Under 45 Lb.	45 Lb. and Less than 85	85 Lb. and Over.	Total.
Pennsylvania	67,558	670,290	314,923	1,052,771
Other States	52,368	712,343	137,945	902,656
Totals	119,926	1,382,633	452,868	1,955,427

This shows the continued increase in the use of heavy sections of rails. No less than 23·2% of all the rails made were over 85 lb. to the yard, while 70·7 were between 45 and 85 lb., the rails of less than 45 lb. section being 6·1% of the total. The 100 lb. section is now in use on several lines, while some experimental lots of 115 lb. have been rolled.

The production of rails from open-hearth steel amounts to only a few thousand tons, that description of steel being used for structural purposes almost entirely. Of the open-hearth steel made in 1898 a total of 1,559,412 tons (70·1%) was basic, and 670,880 tons (29·9%) acid steel. The proportions in 1897 were 65·6 and 34·4% respectively. The increase in basic steel in 1898 over the previous year was 47·7%, while that in acid steel was 19·6%.

The production of direct steel castings (included in the table) is increasing. The total of such castings in 1898 was 110,787 tons, of which 18,460 tons were acid and 92,327 tons basic steel.

The open-hearth steel made in 1898 was produced by 65 works in 13 States—Massachusetts, Connecticut, New York, New Jersey, Pennsylvania, Kentucky, Alabama, Ohio, Indiana, Illinois, Wisconsin, Missouri, and California. By far the largest production was in Pennsylvania, where 1,817,521 tons were made. The output of that State included 496,213 tons, or 75·2% of the total acid steel. Of the basic steel Pennsylvania made 1,321,308 tons, or 85·1% of the total; Illinois, 143,891 tons, or 9·2%; leaving but 5·7% made in all the other States.

The production of crucible and other special steels shows a moderate gain. Of the total steel made in 1898, 74% was Bessemer; 25% open-hearth; and 1% crucible and special steels.

Tin Plates.—The production of black plates for tinning in 1898 was 876,954,424 lb., almost all of steel. The growth of this industry is shown by the fact that in 1892 only 40,478,516 lb. were made. The production of tin plates in the United States in 1898 was 732,290,285 lb. The imports of tin plates were 149,576,525 lb., or only a little over one-fifth of the domestic production.

Exports of Iron and Steel.—The exports of iron and steel from the United States in 1898, including manufactures, amounted to $82,774,958, against $62,737,250 in 1897. During the closing months of the year numerous orders were booked for delivery in 1899. In the exportations in 1898 there were large increases in rails, sheets, steel shapes, wire nails and other products. It is noteworthy that these exports are now being made to countries which themselves are large producers of iron and steel, a large amount going to Great Britain, while Germany takes some of these products, notwithstanding the high tariff duties which she imposes.

EXPORTS OF THE PRINCIPAL KINDS OF IRON AND STEEL. (IN LONG TONS.)

Articles.	Exports.			
	1895.	1896.	1897.	1898.
Pig iron	26,164	61,071	262,686	253,057
Bar iron	3,329	3,505	4,493	5,498
Rails	15,507	76,131	148,221	291,038
Hoop and band iron	198	268	1,423	1,795
Steel ingots, billets, etc	3,148	8,855	6,356	28,600
Iron and steel plates and sheets	1,235	2,620	9,119	27,075
Wire rods	....	....	....	18,510
Wire and wire rope	29,674	38,043	53,075	74,695
Nails and spikes	10,228	14,854	24,036	29,449
Structural iron and steel	....	....	....	34,098

Changes in Iron and Steel Manufacture.—Many improvements were made in 1898, chiefly in the direction of economies in production by the consolidation of plants and increased size of furnaces. Progress in the metallurgy of iron and steel is referred to especially in a paper by H. M. Howe, further on in this volume. The two large furnaces of the Carnegie Steel Co. at Duquesne, Pa., which were rated nominally at 500 tons of pig iron per day, have been successfully worked up to 600, and for short periods to 700 tons a day, a hitherto unprecedented capacity. At the Lorain works of the Federal Steel Co. two blast furnaces, each of 500 tons per day nominal capacity, are now under construction. The Ohio Steel Co. is also building two very large furnaces. The Jones & Laughlins Steel Co. began the reconstruction of its Eliza furnaces at Pittsburg, Pa., and the addition of a new stack, which will make a total of four, and give a capacity of 2,400 tons of pig iron per day. The building of these enormous furnaces has brought into greater prominence the casting machines for making pig. The Uehling machine, which is already in successful use in the United States, is now being introduced abroad. This machine and also the Laughlin, which is claimed to be an improvement, are controlled by the Carnegie Steel Co. The so-called direct process, *i.e.*, the transfer of the molten iron directly from the blast furnace to the steel converters, is coming largely into use in the United States. Other economies are being investigated and carried into practice, while the iron and steel industry is also benefited by the improvements in the manufacture of coke through the introduction of by-product ovens, of which several new plants were installed in 1898. The utilization of the waste gases from blast furnaces, which has been the subject of important experiments in Great Britain, Belgium and Germany, has not yet been undertaken by American iron masters. The Bessemer process continues to be by far the most important steel-making process in the United States, although the open-hearth process, both acid and basic, is making steady gains. The basic process will receive a considerable impetus from the success of the furnaces built in 1897 by the Birmingham Rolling Mill Co. at Birmingham, Ala., and the new works which the Tennessee Coal, Iron and

Railroad Co. has under construction at Ensley, Ala. The erection of similar works at other points in the Southern iron district may be expected.

PRODUCTION OF PIG IRON AND STEEL IN THE PRINCIPAL COUNTRIES. (IN METRIC TONS.)

Year.	Austria-Hungary		Belgium.		Canada.	France.		Germany.		Italy.		Russia.	
	Pig Iron	Stee.	Pig Iron.	Steel.	Pig Iron.	Pig Iron	Steel.	Pig Iron	Steel.	Pig Iron.	Steel	Pig Iron	Steel.
1894	1,054,520	a490,000	810,940	396,914	45,827	2,077,647	663,264	5,559,322	a2 700,000	10,329	54,614	1,312,760	422,500
1895	1,075,000	a495,000	829,135	455,550	38,434	3,005,889	899,676	5,788,798	a3,941,300	a10,530	50,314	1,454,298	574,112
1896	1,130,000	a520,000	982,780	598,755	42,164	2,333,702	1,159,970	6,360,982	4,297,447	11,280	59,500	1,629,810	a 625,000
1897	1,905,000	553,000	1,024,666	616,604	41,500	2,472,143	1,281,595	6,888,087	5,091,294	12,500	57,250	1,867,000	821,000
1898	1,250,000	605,500	982,748	653,130	42,250	2,534,427	1,441,633	7,402,717	5,734,307	12,850	58,750	2,228,850	1,095,000

Year.	Spain.		Sweden.		United Kingdom		United States.		All Other Countries.		Totals.	
	Pig Iron.	Steel.	Pig Iron.	Steel.	Pig Iron	Steel.	Pig Iron.	Steel.	Pig Iron	Steel.	Pig Iron.	Steel.
1894....	260,000	70,000	459,132	205,865	7,364,745	3,050,000	6,050,000	4,482,592	a350,000	a250,000	26,058,970	12,785,749
1895 ...	206,430	65,000	455,200	231,900	8,022,006	3,444,201	9,597,449	6,212,671	a375,000	a275,000	29,858,439	16,659,005
1896....	246,326	104,577	466,400	250,600	8,700,220	4,306,211	8,761,197	5,366,518	395,000	290,000	31,009,831	17,581,181
1897....	282,171	121,100	533,800	268,300	8,930,086	4,559,736	9,807,123	7,289,300	450,000	310,000	33,505,076	20,979,179
1898....	261,799	112,605	510,550	289,750	8,769,249	4,639,042	11,962,317	9,075,783	545,000	355,000	36,507,487	24,060,500

(a) Estimated.

The increase in the total pig iron production was 3,002,411 tons, or 9%; in steel, 3,050,853, or 14·5%. The proportion of increase was much greater in steel than in pig iron, showing that a larger proportion of the raw iron produced was converted into steel last year than ever before. The iron production of 1898 was by far the largest ever recorded; as compared with 1894, only five years ago, the gain was 10,448,517 tons, or 40·1%. The gain in steel during the five years was even more marked, the total for 1898 being greater than that of 1894 by 11,274,751 tons, or 88%. The great increase in the proportion of pig iron converted into steel is due to the improvement and cheapening of the processes of steel production and the greater uniformity and certainty of the results obtained; steel having become for most purposes of construction the material which can be supplied in large quantities at a more moderate cost and of more uniform quality than wrought iron. Steel is also replacing cast iron for many purposes. Thus steel castings—and especially of open-hearth steel—are now largely used in locomotive and stationary engine construction for many parts where cast iron was formerly employed. In the smaller finished products, such as hardware, stamped steel is replacing malleable iron castings. Tin plates also are now made of steel almost entirely in this country, and to a large extent in Great Britain.

In 1898 the United States made 32·8%, Great Britain 24% and Germany 20·3% of all the pig iron; these three countries furnishing 77·1% of the world's supplies. In steel the United States reports 37·6%, Germany 23·9% and Great Britain 19·3%, the three countries supplying 80·8% of the total output. Of these three leading countries Great Britain is nearly stationary at present, the increase coming in larger part from the United States, but in a very considerable quantity from Germany.

The production of pig iron in 1898 required the mining and handling of approximately 70,000,000 tons of ore; and it is not surprising that new supplies of such ores are being sought for in many parts of the world.

The activity in iron production and consumption in almost all civilized countries is extending into the present year; and it is not at all improbable that in 1899 the world's production may reach a total of 40,000,000 tons of pig iron and 28,000,000 tons of steel.

Belgium.—The production of pig iron in Belgium in 1897 and 1898 was as follows, in metric tons:

Year.	Foundry Iron.	Forge Iron.	Steel Pig.	Totals.
1897...	78,410	426,332	517,659	1,022,401
1898...	89,575	302,975	590,198	982,748

The production of wrought or puddled iron in 1898 was 509,160 metric tons; an increase of 30,302 tons over 1897. The production of steel ingots in 1898 was 653,130 tons, a gain of 36,526 tons; of finished steel in various forms, 558,995 tons, an increase of 33,264 tons over 1897.

France.—The pig iron production for 1898 was 2,534,427 metric tons; 549,014 tons being foundry iron and 1,985,413 tons forge and steel pig. The increase over 1897 was 62,284 tons, or 2·6%. The production of steel ingots was 1,441,633 tons, an increase of 160,038 tons, or 12·5%; 905,995 tons being Bessemer steel, 512,445 tons open-hearth, and 23,193 crucible and other special steels. The output of finished steel included 651,733 tons of rails, 222,054 tons bars, angles and shapes, and 264,846 tons sheets and plates; a total of 1,138,633 tons. The production of wrought or puddled iron was 801,543 tons, chiefly plates and merchant bar; this was an increase of 17,577 tons.

Of the pig iron product 2,508,177 tons were made with coke or coal as fuel, and 6,754 tons with charcoal.

Germany.—The production of pig iron in Germany, including Luxemburg, is reported by the Association of German Iron and Steel Makers in the table below in metric tons:

Description.	1897.		1898.		Changes.
	Metric Tons.	%	Metric Tons.	%	Metric Tons.
Foundry iron	1,126,408	16·4	1,301,768	17·6	Inc. 175,360
Forge iron	1,619,556	23·5	1,564,149	21·1	Dec. 55,407
Bessemer pig	577,947	8·4	534,674	7·2	Dec. 43,273
Thomas (basic) pig	3,565,156	51·7	4,002,126	54·1	Inc. 436,970
Totals	6,889,067	100·0	7,402,717	100·0	Inc. 513,650

The total increase shown in 1898 over the previous year was 7·5%, a very good proportion of growth. The more important points in the statement are that foundry iron showed the largest proportion of increase, which is an exception to the course of production for several years past, and that the main increase otherwise was in basic pig. Forge iron and Bessemer pig showed actual decreases. We have again evidence that in Germany, as elsewhere, the tendency to substitute steel for wrought iron has received no check. Moreover, in Germany the use of basic steel, the result of the nature of the local ores, continues to increase steadily, while that of acid steel, both converter and open-hearth, is falling off.

The production of iron ores in 1898 was 15,893,246 metric tons, including Alsace-Lorraine, which produced 5,949,777 tons, and Luxemburg, which produced 5,348,951 tons. The steel production of Germany was as follows:

	1897.		1898.	
	Metric Tons.	%	Metric Tons.	%
Basic converter	2,234,214	63·5	3,606,737	62·9
Basic open-hearth	1,304,423	25·6	1,459,159	25·4
Total basic	4,538,637	89·1	5,065,896	88·3
Bessemer	552,657	10·9	668,411	11·7
Totals	5,091,294	100·00	5.734.307	100·00

The total increase in 1898 was 643,013 tons, or 12·6%. In 1897 for the first time, and to a greater extent in 1898, Germany produced more steel than Great Britain, although the output of pig iron in the former country was less by 1,900,388 tons in 1897 and 1,228,434 tons in 1898. This shows how much larger proportion of the German output of pig iron is converted into steel, and to how much greater extent the manufacture and use of wrought iron continues in Great Britain.

Great Britain.—The pig iron production in 1898 reached a total of 8,631,151 long (8,769,249 metric) tons, while that for 1897 was 8,789,455 long (8,930,086 metric) tons; so that there was a decrease in 1898 of 158,304 long tons, or 1·8%. The production was divided as follows: Forge and foundry irons, 4,408,017; Bessemer pig, 3,273,232; basic pig, 730,028; speigeleisen and ferromanganese, 219,874; total, 8,631,151 tons. The decrease in production was due entirely to the long strike of the coal miners in South Wales, which forced most of the furnaces in that district to go out of blast for a time.

The steel production amounted to 4,565,986 long tons, an increase of 80,025 tons, or 1·8% over 1897. Of this total 2,806,600 tons (61·5%) were open-hearth and 1,759,386 tons were Bessemer steel. The open-hearth process has been for years making substantial progress, the preference evidently increasing yearly. By far the greater part of the steel made is acid steel, basic metal increasing in proportion very slowly, as shown by the following table:

	1897.		1898.		Changes.
	Tons.	Per cent.	Tons.	Per cent.	Tons.
Open-hearth:					
Acid steel	2,393,718	92·0	2,590,512	92·3	Inc. 196,794
Basic steel	208,088	8·0	216,088	7·7	Inc. 8,000
Total open-hearth	2,601,806	100.0	2,806,600	100.0	Inc. 204,794
Bessemer:					
Acid	1,374,339	73·0	1,255,252	71·3	Dec. 119,087
Basic	509,816	27·0	504,134	28·7	Dec. 5,682
Total Bessemer	1,884,155	100·0	1,759,386	100·0	Dec. 124,769

Of the total steel made in 1898, therefore, 3,845,764 tons, or 84·2%, were acid, and 720,212 tons, or 15·8%, basic metal.

There were in existence in 1898 in Great Britain 80 Bessemer converters. The average number at work was 62, of which 42 were acid and 20 basic. The average yearly production of acid steel was, therefore, 29,887 tons to the converter, and of basic steel 25,207 tons.

Of the Bessemer steel made, 751,591 tons, or 42·7%, was converted into rails. The remainder of the production was used for a variety of purposes, 189,316 tons being made into bars and 164,124 tons into plates and angles. While Bessemer steel is the material chiefly used for rails, open-hearth steel is the kind in demand for boiler and ship plates and for structural material of all kinds.

The production of puddled or wrought iron in 1898 was 1,115,699 long tons, against 1,288,159 tons in 1897; a decrease of 172,460 tons, or 13·5%. The heaviest decrease was in ship and boiler plates, in which wrought iron is being each year replaced by steel to a greater extent.

The iron ore consumption is estimated as follows: Mined in Great Britain, 14,000,000 tons; purple ore from pyrites, 480,000 tons; imported, 5,468,395 tons; total, 19,948,395 tons. About 70% of the ores used were domestic.

Russia.—The production of pig iron in Russia showed a greater gain than ever before in a single year. The total in 1898 was 2,228,850 metric tons, an increase of 371,850 tons, or 20% over 1897. The production of wrought iron in 1898 is estimated at 510,500 metric tons; of steel, 1,095,000 tons.

The iron industry in Russia has been very profitable for several years, owing to the large expenditures of the government for railroad material and the preference given to Russian works in the distribution of contracts. A large amount of French and Belgian capital, with some German money, has been invested in Russian iron works during the last three years. The growth of the industry has been chiefly in South Russia and Poland.

Spain is important chiefly because of the large supplies of iron ore furnished to foreign countries. The total production of iron ore in 1898, reported by Señor Adriano Contreras in the *Revista Minera,* was 7,125,600 metric tons; which was less than in 1897 by 294,168 tons, or 4%. The consumption of iron ore in Spain in 1898 was 554,772 tons, or only 7·8% of the total, the remainder being exported. The output of iron and steel in various forms is given in the table below, in metric tons:

	1897.	1898.	Changes.
Pig iron	282,171	261,799	Dec. 20,372
Bessemer steel ingots	68,500	54,500	Dec. 14,000
Open-hearth ingots	52,600	58,105	Inc. 5,505
Finished iron and steel	163,500	154,910	Dec. 8,590

Of the pig iron produced last year 16,800 tons were made with charcoal; the rest with coke or coal—chiefly coke. The exports of pig iron were 46,105 tons, against 43,943 tons in 1897, showing an increase of 2,612 tons, or 6·1%. The destinations of the iron ore exported were as follows:

	1897.	1898.	Changes.
Great Britain	5,091,027	4,748,557	Dec. 242,470
Germany	1,058,694	1,193,924	Inc. 125,230
France	435,972	399,424	Dec. 36,548
Belgium	224 776	201,693	Dec. 23,083
Austria	10,350	8,650	Dec. 1,700
United States	59,243	5,792	Dec. 53,456
Italy	....	20	Inc. 20
Sweden	4.526	....	Dec. 4.526
Totals	6,884,588	6,558,060	Dec. 326,528

The chief producers of iron ore were the Orconera Co. and the Société Franco-Belge, both of Bilbao. The reduction in exports was due to a diminished supply of ore rather than to any decrease in demand. The Spanish ores continue in request, since their reputation as good Bessemer ores is well established.

THE IRON MARKETS IN 1898.

The most remarkable feature of the iron markets in 1898 was the continuance of low prices in the face of a strong and increasing demand. Contracts were made in 1898, moreover, which covered deliveries over a considerable part of 1899, but since the beginning of 1899 there has been a sharp rise in prices.

Alabama.—During the first quarter of 1898 the Alabama iron market was active, but the war with Spain interfered with the export trade from April to August, although the furnaces remained in blast. On the whole the iron trade was highly prosperous. The shipments for export were very large and the development of this branch of the trade was important, largely owing to the cheap freight rates quoted from Gulf and Atlantic ports, pig iron being taken as ballast for cotton cargoes. This enabled the Alabama manufacturers to monopolize practically the American export trade in pig iron. The rate on Birmingham iron to Chicago was $3·10, and to New York $3·25, while the rate to Liverpool was the same as to New York, that to Bremen a little cheaper, and that to Genoa only 55@60c. per ton more. To Yokohama $8 was quoted. The total shipments from the Birmingham district amounted to 800,000 tons, an increase of 100,000 tons over 1897. The exports in 1898 were 222,000 tons, shipments being made to Germany, Great Britain, Italy, Japan, Holland and other countries. Besides the pig iron a large amount of cast-iron pipe, finished iron and machinery were turned out. The domestic trade for pig iron was good during the entire year and prices were satisfactory. There was not so much competition as during the previous year. Besides the large open-hearth steel plant under construction at Ensley, a steel wire and rod mill was begun by the same company at that place.

Chicago.—The most noteworthy feature of the iron and steel trade in 1898 was the enormous tonnage of business done at prices which showed little fluctuation. Consumers of all classes used great quantities of material, and the demand increased steadily from the first to the last of the year. In December the rate of consumption in Chicago territory was larger than ever before. The increase in the use of iron and steel in this territory last year was fully 25%

over that of any previous year. The demand for foundry iron was more than sufficient to take the output of the local furnaces, and enabled the outside producers to get a much larger business than ever before. The range of values was fairly steady throughout the year. Prices of local coke iron in January were on a basis of $10·50 for No. 2 foundry, from which they advanced in March to $11. The latter price remained unchanged for the remainder of the year. Southern irons experienced rather more fluctuation, but it was within narrow limits. The lowest price was $10·10 for No. 2 foundry, in July, and the highest was $10·85 for the same grade in December. The local furnaces were in blast throughout the year, with the exception that Iroquois furnace was out of blast for a few weeks in the fall for repairs. Foundry interests did a much larger business than ever before, as is evidenced by the large shipments of iron made in the year. Malleable foundries were particularly active. In steel and finished material an excellent business was done. Sales were heavier, in most lines, than in any previous year, and increased steadily. It is remarkable that while orders were large enough to keep mills busy for the entire year, and in many cases business was turned away, prices were steady throughout the year. This is explained by the desire of the mills to secure a large trade at fair prices rather than small sales at much higher prices. In bars the tonnage was very large. Bar iron sold from the low price of 1c. up to 1·1c. in the year. Good sales of rails were made at the first of the year, and sufficient business was obtained from time to time to keep the mills fully occupied. Prices were maintained at $20, Chicago, until September. In structural material the year was quiet and steady. Comparatively little large building was done, but general business, in the smaller class of buildings, bridges, railroad work, etc., took enough tonnage to make the year's business larger than that of any previous year. The Chicago drainage canal also furnished a large tonnage to the structural mills. Business in plates was heavy through all the year, some months the mills having to refuse offered orders because of inability to meet the shipping requirements of the buyers. Local mills were in operation throughout the year, and did a larger business than ever before.

MONTHLY AVERAGE PRICES OF IRON AND STEEL, CHICAGO, 1898.

	Jan.	Feb.	Mch.	Apr.	May.	June.	July.	Aug.	Sept.	Oct.	Nov.	Dec.
Northern, No. 2, Foundry	10·50	10·50	11·00	11·00	11·00	11·00	11·00	11·00	11·00	11·00	11·00	11·00
Southern, No. 2, Foundry	10·35	10·35	10·35	10·35	10·35	10·20	10·10	10·25	10·60	10·60	10·60	10·75
Bar Iron	21·00	21·00	20·50	21·00	21·00	21·00	20·50	21·00	21·00	21·00	21·00	20·00
Tank Plates	23·00	22·50	22·00	22·75	23·00	22·75	23·00	23·50	24·00	24·00	24·00	25·00
No. 27 Sheets	44·00	42·50	41·00	40·25	40·00	39·00	39·00	40·00	41·50	42·00	40·75	39·50

Cleveland.—Early in the season an association was formed by the agents handling the "old range" Bessemer ores. Following were the prices fixed for the season: Specular and magnetic ores, Bessemer quality, $3·25@$3·65; specular and magnetic ores, non-Bessemer quality, $2·50@$2·75; hematite ores, Bessemer quality, $2·55@$3·25; hematite ores, non-Bessemer quality, $2·10@$2·25. In some instances the prices were advanced over those of 1897, and the average price of Bessemer ores was slightly higher than during the preceding year.

The season opened early and the agents had made extensive arrangements for breaking all records. During the first month of navigation over 1,000,000 tons of ore were brought down to the Lake Erie docks. For several months, or until it was seen that the shippers would be unable to move all their ores by season charters, there were practically no wild rates.

IRON ORE SHIPMENTS FROM UPPER LAKE PORTS.

Ports.	1896.	1897.	1898.	Ports.	1896.	1897.	1898.
Escanaba	2,321,931	2,302,121	2,803,513	Duluth	1,988,932	2,376,064	2,635,462
Marquette	1,564,813	1,945,519	2,245,965				
Ashland	1,566,236	2,067,637	2,391,088	Total by Lake	9,644,036	12,215,645	13,655,432
Two Harbors	1,813,992	2,651,465	2,693,246	Rail Shipments	290,792	253,993	374,250
Gladstone	220,887	341,014	335,955				
Superior	167,245	531,825	550,403	Total shipments	9,934,828	12,469,638	14,029,683

SHIPMENTS OF LAKE SUPERIOR IRON ORE BY RANGES FOR FIVE YEARS.

Year.	Marquette.	Menominee	Gogebic.	Vermilion.	Mesabi.	Totals.
1894	2,058,683	1.139,273	1,810,290	948,514	1,792,172	7,748,942
1895	2,097,838	1,923,798	2,547,976	1,077 838	2,781,587	10,429,037
1896	2,603,839	1,560,467	1 799,971	1,088,090	2,882,079	9,934,446
1897	2,715,035	1,937,913	2,258,236	1,278,481	4,280,873	12,469,638
1898	3,195,039	2,527,274	2,498,462	1,265,142	4,613,766	14,029,683
Totals from opening of mines	52,448,261	27,458,715	25,545,485	11,763,858	16,969,222	134,117,863

While the activity of the movement by water to the Lake Erie ports was almost unprecedented, a new record was being established in shipments from the docks forward to the furnaces. At the opening of the season, May 1, there were 3,167,915 tons of ore on the Lake Erie docks. By December 1 there were 5,136,407 tons on the docks. Consequently the shipments forward to furnaces reached a total of 9,059,829 tons. The corresponding amount for 1897 was 7,453,648, while for 1896 it was 5,021,146. An analysis of the figures shows that the tendency to ship ores direct to the furnaces is growing rapidly. The Lake Erie ports received 11,028,321 tons, as against 10,120,906 tons in 1897. Nearly 3,000,000 tons found their way to places other than Lake Erie ports.

IRON ORE RECEIPTS AND STOCKS AT LAKE ERIE PORTS.

Ports.	Receipts at Ports.			Stock on Docks Dec. 1.		
	1896.	1897.	1898.	1896.	1897.	1898.
Toledo	301,794	416,438	414,012	151,959	194,644	146.568
Sandusky	58,667	79,792	136,200	59,491	84,786	48,500
Huron	226,515	198,231	126,755	200,075	230,029	139,982
Lorain	191.445	355,188	536,086	231,288	317,509	324,034
Cleveland	2,313,170	2,456.704	2,645,318	1.419,311	1,478,355	1,175,970
Fairport	941,446	1,008,340	912,879	773,905	825,312	719,794
Ashtabula	2,272,822	3,001,914	2,684,563	1,441,666	1,835,694	1,732,671
Conneaut	327,623	495,327	1,404,169	275,800	360,895	288,101
Erie	847,849	1,311,526	1,092,364	355,222	484,871	439,167
Buffalo Tonawanda	545,101	797,446	1,075,975	82,267	111,660	121,620
Totals	8,026,432	10,120,906	11,028,321	4,954,984	5,923,755	5,136,407

Lake Freight Rates.—While the average of lake freight rates was in the neighborhood of 10% higher in 1898 than during the preceding season, the advance was of no material benefit to the vessel owners, for the reason that there were fewer charters on the market. Most of the ore was brought down in the large 6,000-ton ships which were put in commission in 1897, season charters having been given the fleet owners. The smaller vessel owners, therefore, had but few opportunities to secure charters until very late in the season, when the shippers were in a measure at their mercy. This accounts for the apparent advance in the lake freight rates. The following shows the average daily rate during 1898: Iron ore, Escanaba to Ohio ports, 50·8c.; head of Lake Superior to Ohio ports, 61c.; Marquette to Ohio ports, 59·8c. During 1897 the average daily rates were: Escanaba, 45·3c.; head of Lake Superior, 57·2c.; Marquette, 54·6c. The average daily rate during the past ten years is: Iron ore, Escanaba to Ohio ports, 67c.; head of Lake Superior, 94c.; Marquette, 83c.

Pig Iron.—The feature of the pig iron market in Cleveland during 1898 was its stability. During the preceding year pig iron prices varied from $9·25 to $11·40. During 1898, however, the price was never below $9·75 and never above $11·25. The average price paid for Bessemer pig iron during January was $10·37½. February was a weaker month, but during the remaining months of the year, excepting March, the average price was higher than during the corresponding months of the preceding year. In April the price advanced to $11 a ton, and from that time on till the close of the year the market was firm and the volume of business transacted was larger than during the corresponding months for many years previous.

PRICES OF BESSEMER PIG IRON AT CLEVELAND DURING 1898.

Month.	Highest.	Lowest	Average.	Month.	Highest.	Lowest.	Average.
January	$10·50	$10·25	$10·37½	July	$11·00	$10·50	$10·80
February	10·75	9·75	10·30	August	11·25	10·75	10·90
March	10·75	10·25	10·50	September	11·00	10·60	10·68
April	11·00	10·25	10·72	October	11·25	10·50	10·71
May	11·00	10·75	10·87½	November	11·00	10·75	10·86
June	11·00	10·75	10·87½	December	11·10	10·75	10·85

Pittsburg.—The Pittsburg iron trade in 1898 made a remarkable record, both in production and in consumption of iron and steel. All the local furnaces were in blast almost continuously throughout the year, and all the local mills were fully occupied. The year served to emphasize once more the unrivaled position of Pittsburg in the iron trade. The sales were the largest ever known, especially in the closing months of the year. In Bessemer pig the sales for the year aggregated 2,097,793 tons. The sales of billets and of mill iron were also very large. The total of all kinds of steel and iron shows an increase of 770,672 tons over 1897. The sales of Bessemer pig iron, steel billets, and mill iron were as follows:

	Bessemer Pig.	Steel Billets.	Mill Iron.	All Kinds.
January-July	898,318	317,250	100,700	1,776,111
July-December	1,199,475	383,421	137,050	2,406,126
Totals, 1898	2,097,793	700,671	237,750	4,182,237
Totals, 1987	1,669,700	668,170	240,904	3,411,565

The total sales of all kinds of iron and steel in Pittsburg in 1898, by weeks, were as follows:

Week.	Tons.	Week.	Tons.	Week.	Tons.	Week.	Tons.	Week.	Tons.	Week.	Tons.
Jan. 1...	82,125	Mar. 5...	70,530	May 7...	52,105	July 9...	60,075	Sept. 10...	85,380	Nov. 12...	53,170
" 8...	57,600	" 12...	96,725	" 14...	67,150	" 16...	54,155	" 17...	77,225	" 19...	56,305
" 16...	66,965	" 19...	91,330	" 21...	73,590	" 23...	66,210	" 24...	74,450	" 26...	75,225
" 22...	50,600	" 26...	71,340	" 28...	81,760	" 30...	53,650	Oct. 1...	92,260	Dec. 3...	87,935
" 29...	44,050	April 2...	83,575	June 4...	62,100	Aug. 6...	59,450	" 8...	64,165	" 10...	323,675
Feb. 5...	80,015	" 9...	26,975	" 11...	69,525	" 13...	148,896	" 15...	55,875	" 17...	96,150
" 12...	167,845	" 16...	39,110	" 18...	50,775	" 20...	178,425	" 22...	55,925	" 24...	96,875
" 19...	95,925	" 23...	38,578	" 25...	47,500	" 27...	167,600	" 29...	51,000	" 31...	42,525
" 26...	71,470	" 30...	46,848	July 2...	49,475	Sept. 3...	106,975	Nov. 5...	76,008		

DECEMBER PRICES FOR IRON AT PITTSBURG IN FIVE YEARS.

	1894.	1895.	1896.	1897.	1898.
Bessemer	$10·25	$12·75	$11·15	$10·25	$10·80
No. 1 Foundry	11·40	14·25	12·10	10·00	11·10
No. 2 Foundry	10·40	13·50	11·25	10·75	10·75
Mill Iron	9·25	11·75	9·80	10·25	10·65
White Iron	9·00	9·00	9·10	9·00	9·25
Mottled Iron	9·00	9·00	9·10	9·25	9·30
Silvery, No. 1	13·60	14·80	14·50	13·00	13·00
Charcoal, No. 1 Foundry	17·50	17·80	17·50	15·50	16·00
No. 2 Foundry	16·25	17·00	16·00	15·25	15·50
Cold Blast	23·50	23·00	23·25	21·50	21·50
Warm Blast	16·50	16·00	16·10	15·00	15·25
Muck Bar	18·40	21·00	19·75	18·75	18·85
Billets	15·15	16·50	16·00	15·50	16·50
Billet Ends	10·45	14·00	13·00	10·00	11·00
No. 1 Scrap	16·00	14·00	12·75	11·00	12·00
Steel Rails	32·00	28·00	25·00	18·50	17·50
Bar Iron	1·15	1·25	1·30	1·15	1·18
Iron Nails	1·00	1·25	1·60	1·30	1·30
Steel Nails	1·00	1·30	1·60	1·30	1·25
Wire Nails	1·00	1·30	1·55	1·45	1·50
Coke at Ovens	1·00	2·00	2·00	1·75	1·50
Ferromanganese, 80 per cent.	49·50	54·00	48·25	47·00	50·00
Old Iron Nails	12·50	15·00	13·00	14·00	14·00
Old Steel Nails	10·00	13·00	12·75	10·50	10·00

The following are the monthly prices of Bessemer pig in Pittsburg during the year: January opened $10·25; closed $10. February opened $9·75; closed $10·25. March opened $10·50; closed $10·65. April opened $10·50; closed $10·85. May opened $10·25; closed $10·35. June opened $10·35; closed $10·40. July opened $10·50; closed $10·30. August opened $10·40; closed $10·60. September opened $10·55; closed $10·55. October opened $10·50; closed $10·40. November opened $10·45; closed $10·40. December opened $10·50; closed $10·90.

Mill Iron: January opened $9·25; closed $9·10. February opened $9; closed $9·10. March opened $9·15; closed $9·25. April opened $9·25; closed $9·25. May opened $9·25; closed $9·10. June opened $9·25; closed $9·10. July opened $9·15; closed $9·25. August opened, $9·25; closed $9·40. September opened $9·50; closed $9·40. October opened $9·40; closed $9·30. November opened $9·35; closed $9·35. December opened $9·50; closed $9·65.

Steel Billets: January opened $15·25; closed $15·30. February opened $15·20; closed $15·30. March opened $15·50; closed $15·75. April opened $15·50; closed $15·40. May opened $15·25; closed $15·30. June opened $15·20; closed $15·15. July opened $15·10; closed $15·20. August opened $15·50; closed $16. September opened $16·35; closed $16·30. October opened $16·15; closed $15·75. November opened $15·60; closed $15·75. December opened $15·85; closed $16·65.

NOTES ON THE PROGRESS OF IRON AND STEEL METALLURGY IN 1898.

BY HENRY M. HOWE.

1. THE BLAST FURNACE.—(*a*) *Gas Engines for Blast Furnace Gas.*—One of the year's most important developments in the whole range of metallurgy is that of the direct use of the waste gases of the iron blast furnace in gas engines. Owing to the necessary conditions of the blast furnace, the gases which escape at its top are inevitably very rich in carbonic oxide, containing say from 20 to 26% of that gas. It is thus not so very far in value from common producer gas; and, indeed, it is richer than some producer gas. This blast furnace gas, made in enormous quantities, and formerly allowed to burn to waste, is now burnt to preheat the blast with which the blast furnace itself is supplied, and to raise steam both for driving the engines which furnish that blast, and for other engines if there be a surplus available for them. But it now seems probable that this surplus gas over and above that needed for heating the blast and driving the blowing engines will generally be used directly in gas engines, instead of being burned under boilers to generate steam as a source of power. Several papers on this subject appeared in 1897,* and in 1898 it was discussed elaborately by F. W. Lürmann,† and the results of important trials at Seraing were given by MM. A. Greiner and A. Witz.‡ The value of M. Greiner's paper is increased by an important contribution to its discussion by Mr. James Riley.

At first sight we naturally fear that if blast furnace gas be used direct in a gas engine, its dust and perhaps also its sulphuric acid will injure the cylinder, and that the fluctuations in its composition will cause serious trouble. But experience already goes very far toward dispelling these fears. Thus the little 8-h. p. gas engine at Seraing ran for four months without having its cylinder cleaned. In considering this is to be remembered that 20% of the ore charged at Seraing is fine ore, and that there are no large dust catchers; so that the dust trouble there should be more serious than in more modern works. Corrosion thus far has not been detected at Seraing; and the variations in the composition of the gas appear from M. Greiner's words to be a less serious thing when the gas is used in a gas engine than when it is burnt under boilers; for gas which is so poor that it ignites only with difficulty under the boilers will still work regularly in the gas engines.

The estimates of the quantity of surplus calorific power available over and above that needed for heating the blast and for raising steam for the blowing engines are surprising. M. Hubert thought that there would be 20 h. p. per ton of pig iron made per week; though M. Greiner seems inclined to divide this by two, the quantity of surplus power, 10 i. h. p. per ton of pig iron per week, the quotient of his division, is still very large, corresponding to some 75,000 i. h. p. for a pair of furnaces like those at Duquesne. Even Mr. James Riley estimates the surplus power as 6·73 i. h. p. per ton of pig iron per week.

Mr. Riley's estimate applies to two blast furnaces, each making 800 tons of

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* *Journal West of Scotland Iron and Steel Inst.*, V., 1898, pp. 49, 65; Hubert, *Annales des Mines de Belgique*, February, 1897, p. 37; Lencauchez, *Bull. Soc. de l'Industrie Minerale*, November, 1897.

† *Stahl and Eisen*, XVIII., p. 247, 1898.

‡ *Journal Iron and Steel Inst.*, 1898, I., p. 21, and 1898, II., p. 130.

Bessemer pig iron per week from hematite ore, with 20·5 cwt. of Durham coke per ton of iron made. From his calculations I take the following:

Total gas yield of furnaces per hour	1,729,862 cu. ft.
Calorific power, at 100 B. T. U. per cu. ft.	172,086,200 B. T. U.
Of this we must use for heating 103,040 lb. of blast to 1,210° F., or about ⅕ of the whole	33,488,000 B. T. U.
We further must use for raising steam for the blowing engines, or about ⅓ of the whole	54,200,000 B. T. U.
Balance available for use in gas engines, over and above the gas used in heating blast and raising steam for blowing engines	80,298,000 B. T. U.
Total, as above	172,986,000 B. T. U.

Available h. p., at 1 hour = 7,912 B. T. U., 85,298,000 ÷ 7,912 = 10,780 h. p.

Available horse power per ton of pig iron per week = $\frac{10780}{1600}$ = 6·73.

or four and a half times as much as if this surplus gas were burned under boilers. Here is the essence of the economy that this gas yields so very much more power in a gas engine than when used to raise steam as a source of power.

In reality Mr. Riley's case seems capable of even stronger presentation. For if the gas now burnt under boilers to raise steam for the blowing engines were instead used direct in gas blowing engines, and if the quantity of gas needed for generating the blast were thereby divided by 4·5, we should need for that purpose only 54,200,000÷4.5=12,040,000 B. T. U. instead of the 54,200,000 above allowed for this purpose, so that the difference, 54,200,000—12,040,000=42,160,000 B. T. U., must be added to the 85,298,000 above found available for gas engines for purposes other than blowing, thus increasing the power available from this source by about half, or to say 16,000 i. h. p. for a pair of furnaces each making 800 tons of iron a week, or say 10 i. h. p. per ton of iron per week.

If we apply these numbers without modification to a pair of furnaces making say 500 tons a day apiece, we reach the astonishing result that, after supplying and heating their own blast, they would yield a surplus of gas capable of generating 70,000 h. p. But great modification must be made. In our best American furnaces which make these great outputs, since the consumption of coke is smaller per ton of iron, the total amount of carbon in the gases per ton of iron is correspondingly smaller; and as the ratio of $CO:CO_2$ is doubtless smaller also, so that a smaller percentage of that smaller quantity of carbon is available for fuel as carbonic oxide in the furnace gases. Thus while about one-fifth of Mr. Riley's gas is needed for blast heating, some two-fifths of it is needed for this purpose in some of the best American practice. But even after making all such allowances there is an enormous surplus of gas available for power. It is said that at both Homestead and Duquesne about half the steam used for the rolling mills, etc., comes from the blast furnaces. And there can hardly be a doubt that the immediate future will see a very great economy brought about by the use of blast-furnace gas in gas engines.

(b) *Multiple tuyeres* have been experimented with much during the past year. The number has been in many cases raised from eight to 12, 16 and even to 24, with corresponding reduction of their diameter. The change should and probably does lead to greater uniformity of working in the different parts of the periphery of the crucible; but, unless the blast pressure is increased correspondingly, there should be slightly less penetration. In other words, for given initial velocity a 6-in. column of blast should penetrate further than a 3-in. one. In some cases this increase in the number of tuyeres has been beneficial, in others it has not; which means that the conditions in some furnaces are such that the benefits of multiplying the tuyères outweigh the harm. Further experience and evidence are needed to permit us to speak confidently on this subject.

(c) *Casting Arrangements.*—The old arrangement of casting the enormous output of the blast furnace, in some cases more than 500 tons per 24 hours, in small pigs in sand molds which have to be specially prepared by hand for every cast, their sand sticking to the pig iron and befouling it; of breaking these pigs off by hand from the main runner or sow; of lifting them by the ache of men's backs one by one and carrying them across the casting house to a railroad car; this old method, I say, has long been felt to be inexcusably barbarous. For many years iron molds have been in use about Pittsburg instead of sand ones, with a certain lessening of labor, though with a not inconsiderable expense in keeping up these iron molds themselves. Then came a great and important step of lifting up a whole runner or sow with its attached pigs, by a crane, which carried them to a breaker. This, in turn, broke the pigs in two and the sow into short pieces, by pressure from strong short hydraulic plungers, and delivered the fragments direct into a railroad car standing underneath, or in some cases upon a conveyor which carried them to the remelting cupola or other desired place.

But of far greater promise is the Uehling* metal-conveying apparatus, which in November, 1898, was taking care of more than half of all the Bessemer pig iron made in this country, and is expected to be in use for more than 70% of the total Bessemer pig iron production in May, 1899. One of these machines has been installed in England, and others are to be erected in Europe. In eight continuous months one of these casting machines at the Lucy furnaces handled 163,500 tons of iron, the total output of the two furnaces, without missing a single cast. The saving of cost was estimated at 10·6c. per ton,† which for a pair of furnaces like those at Duquesne would be at the rate of some $36,000 a year.

In the Uehling arrangement the cast-iron is tapped from the blast furnace directly into a large ladle, which is carried by rail to a central casting machine, placed so as to be used for a number of furnaces, indeed naturally for all the furnaces of the establishment. The casting arrangement consists essentially of a "Jacob's ladder," or common bucket elevator, of which the buckets and their supporting belt are replaced by a series of horizontal trough-shaped, open, cast-iron molds, their ends supported and carried by the two parallel endless chains which form the skeleton of the
very gradually, carries these molds successively un
ladle which contains the molten cast-iron from the
tips progressively, and fills the passing molds o
on slowly up the elevator, so slowly that by th

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* *American Manufacturer*, LXIII., p.

† E. A. Euhling, private communication

reached the top of the elevator the pig which it contains has solidified. When the mold passes the sheave at the top, the pig is dumped out upon another conveyor, which is horizontal and partly submerged in water. Here the pig cools off more quickly and is carried forward by this conveyor, and by it in turn dumped upon a railroad car. On its return passage the elevator which carries the molds is sprayed with water, and, if desired, with lime-wash. So, too, on its upward travel, while carrying the hot pigs, it may be sprayed to hasten their cooling.

This method has the advantage over that of casting the pigs attached to a sow and then breaking them up, that it avoids the large expenditure of power required for breaking up the several pigs and the sow. If the iron is to be used in small pieces, why not cast it in small pieces rather than in large ones, which later have to be broken up at considerable expense?

Mr. David Baker, of the South Works of the Illinois Steel Co., has also devised a very promising casting machine for the iron blast furnace.

2. Refining and Knobbling.—The refinery fire and the knobbling furnace are still in use to some extent in this country, though it may seriously be questioned whether they are not simply slowly departing relics of a by-gone day; in short, whether their real need and usefulness has not ceased. The refinery in effect removes the silicon of cast-iron by melting it and blowing upon its surface while covered with a layer of incandescent coke. The stream of air directed upon the surface of the metal gradually oxidizes and removes both silicon and carbon; the resultant silica of course rises to the surface, as does the resultant carbonic oxide. But, whereas the removal of silicon is effective, that of carbon is not, because as fast as we burn carbon out by the blast of air, so fast is it re-absorbed from the layer of incandescent coke which floats upon the molten metal. This, then, is a mode of getting a cast-iron which consists almost solely of iron and carbon. It is of course perfectly white, and is almost and perhaps quite identical in composition, properties, and value with washed metal, by which one naturally supposes that it will eventually be replaced after the usual considerable period of friction and inertia which exists in such cases; if indeed the demand for refined iron does not first completely cease.

In the knobbling furnace the refined cast-iron is melted down one or more times under a cover of charcoal, and is simultaneously decarburized by the xygen of a blast of air, and also by that of the rich iron oxide slag which esults, and the product is charcoal wrought-iron made from refined iron.

Not many years ago much of this charcoal iron was used for wire for cutting to screws. Its great purity, that is to say, the freedom from sulphur and hosphorus, was its advantage over puddled iron, while its advantage over ft steel was that the screw-[illegible]hine cut the thread in it much more sily and much more acc[illegible] soft steel. The thread of soft steel nds to clog and [illegible]gh one and perhaps through th of these [illegible]venly cut, and to cause the al to go[illegible]ns in such cases, when the rew[illegible]pects as good as the ol[illegible] they found or made a way—whether by changing the speed, the cutting angle, the lubrication, or other details of their screw-cutting machines—to use this steel. Therefore I understand that the knobbled iron is now used to only a very moderate extent for screw-cutting. For very high conductivity iron wire it may have the advantage of having less manganese than it is convenient or easy to leave in open-hearth or Bessemer steel.

The refinery and the knobbling fire are still used to a very considerable extent for making wrought iron boiler tubes. This is due to the not uncommon belief that wrought iron, at least in the condition of boiler tubes, resists corrosion better than soft steel. But admitting this to be the case, it is not very easy to see why knobbled iron should be greatly better for this purpose than the puddled iron, which is certainly very much cheaper. By careful selection of the pig iron to be puddled and by careful puddling, puddled iron should be as free from phosphorus and sulphur as knobbled iron actually is. And one would suppose that it ought to be practicable to give puddled iron practically the same percentage and distribution of cinder as knobbled iron. In short, it is not easy to see what the essential difference between the two is, except that knobbled iron is made in smaller units and perhaps with greater opportunity for careful manipulation.

3. PUDDLING.—Far from this process being a moribund one, we find new puddling furnaces erected by some of the most intelligent and progressive of crucible steel makers, for the purpose of supplying themselves with extremely pure wrought iron. Right or wrong, the belief is still tenaciously held that crucible steel made from wrought iron, whether puddled iron or knobbled iron, is incomparably better than crucible steel of like composition, made in exactly the same way from scrap Bessemer or scrap open-hearth steel. It is very difficult to understand this, though it would, of course, be idle to say that it is an error. Yet there are certain considerations which suggest themselves to the mind of the bystander.

In the first place it is very difficult to explain, or put into words, or give any numerical measure of the relative superiority of two pieces of tool steel, both good, but one only very good, the other remarkably good. As the bystander cannot understand the basis of the discrimination which the crucible steel expert makes, it is not unnatural that he should be inclined to wonder whether that discrimination is really wholly trustworthy, and whether the crucible steel maker does not perhaps deceive himself. This of course is a dangerous line of reasoning, for proceeding along it a man who was no judge of pictures might deny any difference in the merits of two pictures because he was unable to appreciate or understand the connoisseur's grounds for preference. But in the present case, even though we may not be crucible steel experts, we have a considerable knowledge of closely related subjects, and therefore consider ourselves in pretty fair position to pass upon the merits of any process of testing or discrimination between different grades of tool steel. If we do not know it we ought to be able to understand it. Hence there are probably not a few of us who have a lingering suspicion that, after all, open-hearth or Bessemer steel, if of the same composition as wrought iron (which is rarely the case), might, when remelted in crucibles, give equally good crucible steel.

4. DIRECT PROCESSES.—In 1889 I pointed out, among other conditions, that direct processes should be especially applicable (1) to rich ores; (2) to cheap ores, and (3) where fuel is dear. To these I might have added, where water power is cheap, and thus available as a source of heat and reducing power. These conditions coexist in Sweden, and we find here very serious consideration given during the past year to the development of direct processes.

In this country, too, more than one eminent metallurgist, not of the visionary type with which we have had in the past so often to connect direct processes, but of the class of successful administrators, has taken up the problem in great earnest. Their plans are hardly yet ready for publication, but one at least of them has very great promise of success.

5. THE IRON FOUNDRY.—A perusal of the discussions which have taken place before the various foundrymen's associations as to the relation between chemical composition and physical properties of cast iron do not produce the impression that this relation has yet been mastered at all fully by our iron founders, and one is at first inclined to doubt whether most of them are in a position to draw much valuable information from analysis. However, this is probably due in large part to the fact that, in the desire to increase their knowledge, they frankly dwell on what they do not know rather than on what they do. A very encouraging bit of information is furnished by the *American Manufacturer*,* which finds that, out of the 87 founders who replied to its circular letter, 64% actually buy on analysis, and that some of the others are inclined to.

The paper of Mr. Guy R. Johnson, on "The Relation between the Composition and the Properties of Cast Iron," read at the Stockholm (August) meeting of the Iron and Steel Institute, deserves careful study.

Dr. Richard Moldenke, of Pittsburg, has taken an important step toward putting foundry practice on a more scientific, *i.e.*, a more intelligent basis, in devising a modification of the Le Chatelier thermo-electric pyrometer, especially adapted to measuring the temperature of molten iron, for instance in the foundry ladle. To make such a measurement the thermo-electric junction or "couple" of the pyrometer must be brought to the temperature of the molten iron itself, and therefore must be immersed in it, with great danger of destruction. The essential feature of Dr. Moldenke's device is a special clay tip for protecting the couple. This and other apparently excellent contrivances should greatly facilitate the application of this admirable instrument to metallurgical purposes; but great care will, I fear, still be needed in its use, and in incompetent hands it may be very misleading.

Dr. Moldenke has published a long and important series of observations of the melting points of cast iron, steel and iron alloys of various compositions, agreeing well with those of our earlier determinations which we had reason to believe true.

6. THE BESSEMER PROCESS—(*a*) *Car Casting.*—In the Bessemer process a still further extension of the car casting system, so ably worked out by F. W. Wood,

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* *American Manufacturer*, Feb. 17, 1899, p. 161, Vol. LXIV.

has been made. It has been introduced even at the great South Works of the Illinois Steel Co., where the old swing crane system had reached its highest development and economy. There a single man, managing one ingot crane with its three movements of lifting, rotating and racking (*i.e.*, moving radially), could strip three molds simultaneously, set them down on their proper car, then lift from the casting pit three ingots at once, and set them on their proper car. The high development of the swing ingot crane led to very great economy at South Chicago; but the introduction of the car casting system has led to still further economy, not so much in the number of men employed (which is not much lessened) as in their rate of pay, since with the car casting system neither great skill, intelligence, nor heat-resisting power is required.

(*b*) *Recarburizing.*—Carborundum has come into use as a substitute for ferrosilicon as recarburizer. It appears to offer the same advantage over ferrosilicon that aluminum offers over ferro-aluminum, viz., that the active agent, silicon, is not diluted by a great quantity of iron. For instance, it has been found extremely convenient to add aluminum in small pieces, either in the very molds themselves or in the ladle just before teeming, a thing which would be manifestly impossible to do with low grade ferro-aluminum. Much the same advantage appears to hold true of carborundum. It would often be possible to add this substance to the molten steel after it had left the furnace or converter, without the danger of chilling, and with much greater uniformity of the resulting metal, than if ferro-silicon were added.

(*c*) *Baby Bessemer.*—The past year has shown a good deal of activity in the exploitation of the Tropenas Bessemer converter, which differs strikingly from most of the preceding small converters in that its claims are intelligible. I do not in the foregoing refer to the Walrand process, of adding a little high silicon iron at the end of the blow, for here we have not simply a special converter, but a special process, worthy of that name. Whatever may be its actual value, it is certainly wholly rational.

The Tropenas converter admits two blasts of air, one just below the upper surface of the molten metal, and one just above it, with the aim of burning there the carbonic oxide formed in the bath itself by the action of the blast upon the carbon of the metallic iron. This idea is certainly wholly rational. To what extent this combustion occurs in the Tropenas converter direct experiment alone can show, and although such an experiment ought to be easy to conduct, I have not yet heard that it has been made. And while the metal in the Tropenas converter may be made exceedingly hot, a sufficient reason for this may be that the tuyeres which blow into the iron, enter it very near its upper surface, and therefore presumably burn a large amount of iron, and thereby generate a very great quantity of heat. I have seen as yet no strong evidence that any material part of the heat is furnished by more complete burning of the carbonic oxide to carbonic acid than occurs in other converters in which the blast is in like way introduced a little below the upper surface of the molten iron, such as the Clapp-Griffiths. We reasonably fear that the very high temperature claimed is obtained at the sacrifice of oxidizing, and thus wasting, a very large percentage of iron.

7. ROTARY OPEN-HEARTH FURNACES, *i.e.*, those of which the melting chamber or laboratory part rotates nearly or quite about its longitudinal axis, so as to lower the tap-hole during pouring, are coming into somewhat more general use in this country. These furnaces offer two classes of advantages: First, those more or less directly connected with the management of the tapping hole; and second, those connected with holding the furnace in an inclined position during or between heats.

Tap-hole Advantages.—Here a saving of time, labor, and material, a greater yield, a longer life, and a more accurate control are claimed. As the closing of the tap-hole of a rotary furnace need receive but little attention, a saving of from 30 to 60 minutes between heats in the time required to make it up is claimed.

Material.—It is asserted that 25% of the refractory material used for making up the bottoms between heats is required for the tap-hole. Nearly all of this may be saved in a rotating furnace, so light may the stopping of the tap-hole be.

Yield and Life.—Considerable roughnesses form on the bottom of the furnace. In a stationary furnace these dam back an appreciable quantity of metal first and last, which, oxidizing between heats, is in considerable part lost, or is effective only as so much ore would be, and moreover bores into the bottom, injuring it and shortening its life. But a rotating furnace may be inclined sharply at the end of the pouring, so as to pour the metal out more completely, thus increasing the yield and lessening the damage due to the boring action of the metal left behind in the furnace.

Control in Making High-Carbon Steel.—If the charge is to be arrested as soon as the carbon has fallen approximately to the percentage aimed at, a considerable variation is liable to arise if the furnace be stationary, from the variations in the length of time which is required for tapping out the charge, due in turn to the variations in the condition of the tap-hole; and these variations in length of time induce corresponding variations in the percentage of carbon in the resultant steel.

If decarburization be pushed, however, far beyond the point aimed at, and the charge then be brought back to the desired percentage of carbon by heavy recarburizing, *i.e.*, by introducing a large amount of the recarburizer, the carbon is liable to vary very greatly for another reason. If, on one hand, the tap-hole when opened is very small, so that the metal runs out slowly, it may reach the ladle too cold to take up fully the recarburizer present. If, on the other hand, the tap-hole opens very wide, the slag is liable to run out into the ladle in considerable quantities, and not only to cause with the carbon in the ladle a very violent reaction, but also to oxidize and thereby waste a large part of that carbon. With a rotating furnace, however, the metal can be poured in a uniform stream upon the carbon in the ladle, and just before the slag comes it can be tipped back, so as to give the metal in the ladle time to take up the carbon, before pouring into the ladle the last of the metal and the slag which would accompany it.

The Campbell and the Wellman Furnaces.—There are two well-known types of rotating furnaces, the Campbell, of Mr. H. H. Campbell, of Steelton, Pa., and the Wellman, of Mr. S. T. Wellman, of Cleveland, Ohio. The barrel-shaped Campbell furnace (Fig. 1), rests on a large number of rollers set in the arc of a circle which is centered at the axis of the furnace, *i. e.*, at a line running through the center of the single port at each end of the melting chamber. The piston of a hydraulic cylinder, grasping this furnace by means of a link, rotates it about this axis, with the result that whatever be the position of the furnace the ports are still in place, and at one end bring in gas and air, and at the other lead out the products of combustion; in short, that the heat may be on, no matter what the position of the furnace is.

The Wellman rolling furnace (Fig. 2), is of approximately the same shape as the Campbell furnace, but rests at its ends only, on a stationery rack *D'* into which gears the segment *D* of a pinion fastened to the furnace, with which it is concentric. The piston rods of two nearly vertical hydraulic cylinders, *E*, lay hold of this furnace, and, rotating it, roll it along this rack forward or backward as the case may be. The rack and pinion arrangement insures the return of the furnace accurately to its original position. Unfortunately, whenever the furnace tips or rolls, the connection of the ports is broken, and it is here that the Campbell furnace appears to have a material advantage over the Wellman, especially in baking the back wall and while charging ore, or "oreing."

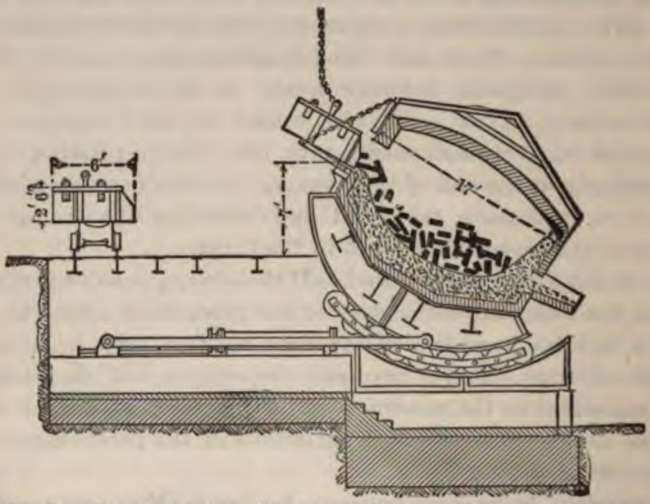

FIG 1.—THE CAMPBELL TILTING AND OPEN-HEARTH FURNACE.

Baking the Back Wall.—It is desirable to make the back wall, *i.e.*, the wall on the tapping side, as steep as practicable, so as to give as much room as possible in the furnace; but the basic materials do not readily permit us to make a very steep wall under ordinary conditions, because until baked hard they tend to slide down. In the Campbell furnace the back wall may be built at an angle of even 80° with the horizontal, by the simple expedient of so tipping the furnace while building up and baking the back wall, that the face of that wall, though so shaped that when the furnace is in its normal position it will stand at an angle of 80°, will during this time make an angle of only 20° with the horizon.

Oring.—While we are adding ore to the charge, the bath tends to boil seriously, and the rapidity with which we can add the ore, and consequently the rapidity with which we can bring the charge forward, is limited in part by this boiling, for we cannot allow the boil to be so active as to throw the metal out of the charging doors. Now the Campbell furnace during oring may be inclined so as to bring the charging doors above their normal position and thus to increase the amount of boiling which is permissible, and thus in turn to enable us to bring the charge forward more rapidly. This, as well as the power of inclining the furnace while the back wall is being baked, is of course due to the fact that the ports are always in position, and the heat therefore always on, even when the furnace is inclined. But if we are thus to incline the furnace during oring we must stop the taphole strongly, thus losing the first two of the "tap-hole advantages" already enumerated.

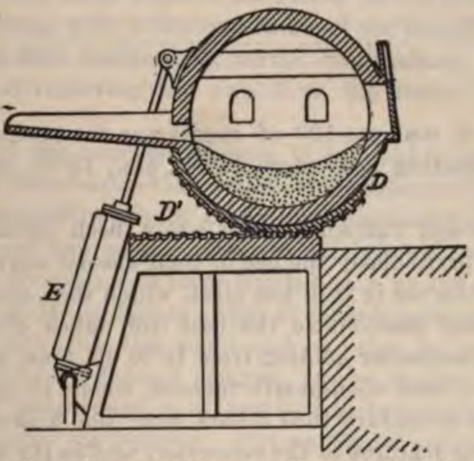

FIG. 2.—THE WELLMAN ROLLING OPEN-HEARTH FURNACE.

The Wellman furnace, on the other hand, has a certain advantage in that there is no machine-finished or faced work either under the furnace, as in the case of the Campbell furnace, or indeed elsewhere.

Both furnaces have considerable advantage over most stationary furnaces in that, if a serious breakout occurs, the metal runs into the pit below the furnace, where it can do no serious harm. Thus a 60-ton charge has run out of one of the Wellman furnaces through the bottom: but the steel was so easily accessible and the furnace so readily repaired, that the time lost amounted to only that of making one or two heats. In this respect the Wellman furnace appears to have somewhat greater immunity than the Campbell. It is very unlikely that a breakout in the Wellman furnace should interfere with the mechanism, while in the Campbell furnace the chance of damage seems to be appreciably greater. This immunity is due, of course, not to the rotation but to the suspension of these furnaces; in other words, a furnace which, though immovable, was perched as either the Wellman or the Campbell furnace is with a clear space beneath, would have practically the same immunity as regards breakouts.

Number in Use.—Of the Campbell furnaces there is the superb series of six, with a capacity of 50 tons each, at Steelton. Of the Wellman furnaces there are 21 in use, with a total capacity of 808 tons. The most important group is that which was building in 1898, ten 50-ton furnaces, at Ensley, Ala. Admirable as is the Wellman furnace, it would be unjust to suppose that the greater number of Wellman than of Campbell furnaces in actual use necessarily implied that the Wellman was in a corresponding degree the better furnace; for while the Wellman is actively pushed, the Campbell furnace is not.

8. DUPLEX PROCESSES (*a*) *Daelen and the Duplex Process.*—The duplex process aroused a good deal of interest in 1893, when Mr. Kupelwieser described the practice of this process at Witkowitz.[*] There molten pig iron direct from the blast furnace, and containing 0·80 to 1·20% of silicon, was desilicidized in an acid converter, and the resulting metal, still rich in carbon, was treated in a basic open-hearth furnace. In treating charges of

Desilicidized pig	56·16%
Solid pig	36·82
Scrap	7·02
	100·00

the consumption of coal per 100 of ingots was for producers, etc., 16%; for blowing engines, heating converters, ladles, etc., 15·5%; or a total fuel consumption of 31·5%.

Messrs. Daelen[†] and Pszczolka have lately been advocating this duplex process, and more particularly the use of their special converter for disilicidizing cast-iron, and the use in it of hot blast, which they obtain from the blast furnace plant. They desilicidize the cast iron taken direct from the blast furnace, in a basic converter holding from 10 to 12 tons, and run the desilicidized metal into a basic open-hearth furnace, where its conversion into steel is completed. The procedure thus differs from the Witkowitz practice essentially in the peculiar features of the converter, and in the use of hot blast. It has been in use during the past year at Krompach, in Hungary, where cast iron containing manganese 2·5% and silicon 0·7% is desilicidized and then transferred to the basic open-hearth furnace.

The loss in the converter is 7·29% and the further loss in the open-hearth furnace, if this desilicidized metal exclusively be used, is in the two cases given, 6·29 and 7·79%: so, that the total loss is approximately 13·8%. Naturally the desilicidized metal is brought forward in the open-hearth furnace much more rapidly than an ordinary charge of pig and ore would be, and actually considerably faster than a charge of pig and scrap. Mr. Daelen reports that a given open-hearth furnace, when running on this desilicidized metal, will turn out two or three times as much per diem as it would when running on pig and ore, and one half more than if running on pig and scrap. The fuel consumption is given as 15 to 20%, which thus is considerably below that reported for Witkowitz.

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[*] *Journal of the Iron and Steel Institute*, 1893, II., p. 6.

[†] Circulars and private communication, Jan. 10, 1899, R. M. Daelen, Düsseldorf.

(*b*) *In the Bertrand-Thiel Duplex Open-hearth Process*,* about half of the charge, containing most of the pig iron used, is desilicidized and superheated in a preparatory open-hearth furnace, while the other half, consisting chiefly of scrap iron, is melted and oxygenated in a lower or finishing open-hearth furnace. The two lots are then mixed together by running the charge from the preparatory furnace into and mixing it with that in the finishing furnace, the slag being dammed back in transit. The oxygenated charge of the finishing furnace now reacts rapidly on the carbon and phosphorus of desilicidized charge from the preparatory furnace, the more rapidly because both charges are very hot and very fluid. For instance, a 13-ton charge, consisting of 60% of pig (of say 2·5% phosphorus) and 40% of scrap, has its phosphorus reduced to between 0·6 and 0·9% in the preparatory furnace, and is then mixed in the finishing furnace with a 9-ton charge consisting of 90% of scrap with only 10% of pig, and therefore much oxygenated. The reaction is so rapid that the phosphorus falls in 15 or 20 minutes from between 0·35 and 0·53% to 0.02%.

In other words, the process consists essentially in oxygenating part of the charge, and then mixing with it the remainder of the charge (which meantime has in melting down been incidentally partly desilicidized, decarburized and dephosphorized), and removing the slag from the latter part in the act of mixing.

The philosophy of the process appears to be that a bath of pig iron is dephosphorized and decarburized more rapidly by means of a large addition of oxygenated metal, especially if both baths are superheated, than it can be by feeding ore. It is not very easy to see why this should be the case, as the rapidity of working by ore seems to be limited only by the consideration that the ore must not be added so fast as either to cause the metal to boil out of the doors, etc., or to decarburize the metal and so raise its melting point faster than the temperature of the furnace can rise, and so in turn cause the metal to set on the bottom of the furnace, or at least lead to its being too cool to tap when its composition is proper for tapping. But we should suppose that these considerations would limit in the same manner, and indeed in the same degree, the rapidity of the action of the oxygenated metal which, in the Bertrand-Thiel process, takes the place of the ore of the common open-hearth process. In other words, we should suppose that a given rapidity of decarburizing would cause just as violent boiling if induced by oxygenated metal as if induced by oring; and hence that, the speed of oring being limited by the danger of damage from excessive boiling, no more excessive boiling, and hence no more rapid decarburization would be permissible with the oxygenated metal of the Bertrand-Thiel process. So, too, of too rapid decarburizing.

The normal procedure appears to be to have three furnaces working in conjunction, viz., two preparatory furnaces and one finishing furnace. It is estimated that with two 10-ton preparatory furnaces and one 20-ton finishing furnace, ten 20-ton heats should be made every 24 hours (five heats in each preparatory furnace and 10 in the finishing furnace), or say 200 tons a

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* J. Hartshorne, *Transactions American Institute Mining Engineers*, XXVI., p. 380, 1897, and June, 1898. *Stahl und Eisen*, XVIII., January, 1898, p. 86; February, 1898, p. 146.

day, or 1,200 tons per week. If these furnaces were run separately and made 14 heats per week, their output would be 560 tons per week. In other words, by working three furnaces in this special conjunction instead of working them independently, their output may be doubled, a claim which is certainly very surprising, and of the greatest importance if even half true. But, despite the trustworthiness and responsibility of those who make this claim, the apparent opportunities for self-deception are so great that there is great and natural skepticism. The claims as now made turn on the number of heats which can be made per week per furnace, a very variable thing, varying with the relation of the weight of the charge to the area of the furnace, with the proportion of scrap to pig iron in the charge, with the percentage of phosphorus, etc. In Westphalia some furnaces turn out six heats in 24 hours; in this country a fraction over two heats is good work. Yet this does not imply that our Westphalian friends are proportionally more skillful than we. Now, one working in Westphalia and making 36 charges a week may adopt a modification which will raise his output to 42 charges, an increase of 17%. If he applies the same modification to our practice he may reasonably hope to increase our output 17%, or to raise it from 14 to 16 heats per week. But he cannot expect to increase our output to 42 charges per week, or 200% (unless he simultaneously lighten the charges and change their nature, making them consist almost entirely of scrap), simply because he gets 42 heats a week in Westphalia. He must remember that we deliberately use larger charges in proportion to the size of our furnaces and less scrap iron; and that his modification will not lessen the effect of this difference between our practice and the Westphalian. I give this only as one instance of the many ways in which we may be deceived, to show how careful we should be in arguing from comparative data on this subject.

So, without attempting to say in what way these claims may, in all good faith, have become exaggerated, without indeed asserting that they are exaggerated, I do feel that there is easy opportunity in some such way as I have pointed out for self-deception; and, while recognizing clearly the trustworthiness of those who have made the claims, I feel that we should be very deliberate in accepting at their face value these remarkable results for a process which does not on its face explain why it ought to accomplish them. But even if we discount them very greatly, they are extremely important, especially in view of the standing of those who make them. I am informed that the Creusot works is adopting the process, after thorough experimenting.

(c) *Comparison of these Two Processes.*—We may call the Bertrand-Thiel process a two-phase open-hearth process, and the Daelen-Pszczolka process a Bessemer open-hearth process. While the claims actually made for the Bertrand-Thiel are very great, the Daelen-Pszczolka seems to me the one from which we should naturally expect most. One of the great merits of the Bessemer process is its very small fuel consumption; its two great defects are that its loss of iron is very great, and that its product is not generally as good as that of the open-hearth process. Now we should expect the Daelen combination to steer well between these difficulties. Arresting the blow early, it should get the advantage of the heat from the oxidation of the silicon, a fuel which costs nothing, since it must be burnt out in any event; but it should avoid the rapid oxidation and loss of that expensive fuel, iron, which occurs toward the end of the blow. Here, then, it should have the merit of the Bessemer process and avoid its defects. Further, while it is not easy to say on account of precisely what feature of the Bessemer process Bessemer steel is not generally as good as open-hearth steel of like composition, I for one involuntarily suppose that this usual inferiority is connected with something or other at the end of the blow, whether it be the high temperature or the oxygenation and nitrogenizing of the bath at the very end of the blow, without fair opportunity for remedying the damage by the long rest of the open-hearth process. This injury (if such it be) of the Bessemer process, also, Daelen's procedure should avoid. At least there is enough here to make us watch the experiment with great interest.

Against this the Bertrand-Thiel process may claim greater flexibility, because each open-hearth furnace may be used for either phase of the process, and wider applicability. It appears to be applicable to any works large enough to keep three open-hearth furnaces busy, while the Daelen-Pszczolka would be used to its best advantage if there were a considerable number of open-hearth furnaces to take the metal prepared by the converter. Americans, accustomed to have the Bessemer blows succeed each other at very short intervals, may easily exaggerate this consideration. Thus there are successful Bessemer works in Sweden in which a pair of converters makes only two heats per 12 hours during much of the time, and normally makes only four heats per 12 hours.

9. Rolling Mill.—In rolling mill practice two interesting events of the year have been the considerable extension of the scope of the Bedson or continuous mill, as modified by Mr. Charles H. Morgan, not only to merchant iron and skelp, but even to small billets; and the growing use of the continuous heating furnace, in which an ingot or billet is charged at the cold end and gradually pushed or rolled toward the hot end. The continuous mill has taken billets, 4×5¾ in. direct from the blooming mill, and rolled them into billets 1½ in. square, and it is even believed that 6×6 or 7×7 in. square blooms can thus be rolled directly into 1½ in. billets. Mr. Morgan's form of the continuous furnace, with a pair of water-cooled pipes running lengthwise of the hearth over which the billets are pushed, seems a particularly happy arrangement, extremely economical of labor and fuel, and well designed to lessen the loss by oxidation. The normal loss by oxidation in the Morgan furnace appears to be about 2%, and the fuel consumption 150 lb. per 2,240 lb. of product.

In some cases the journals of rolls have been provided with roller bearings; but while this must be regarded as a promising matter, some considerable difficulties in connection with it appear to remain.

Of the Garrett wire-rod mills, three new ones were contracted for in 1898, making 16 in all in this country, with a total capacity of about a million tons per annum, beside one in France (Lazare, Weiller et Cie, Havre). The output of these mills is very great; 2,000 gross tons have often been made in one week in a single Garrett mill, and 228 tons of No. 2 wire rod or 205 tons of No. 5 wire rod have been made in a single mill in 11 hours. Mr. Garrett informs me that such a mill requires from 3,500 to 4,000 h. p., and six men at the rolls.

10. ARMOR AND SPECIAL STEELS.—(*a*) *Armor Plate.*—The adoption of the Krupp secret process for carburizing, both from its secrecy and from the importance of its results, awakened great interest. So far as I can make out it is something over and above the Harvey process. That is to say, I understand that the armor plate is carburized deeply by the Harvey process, and that in addition to this it undergoes a special treatment, the invention and secret of Krupp.

As so often happens in cases of this kind, the perfection of armor plate for the sides of vessels seems likely to lead to its extensive adoption for the walls of banks and safe deposit vaults. It is very generally felt that it is not enough to protect these vaults against the attacks of individual burglars and small gangs, and that the amount of wealth concentrated in them is so enormous as to require protection against mobs armed with cannon. On this account the plan of protecting them by means of heavy Harveyed armor plate is receiving serious attention; and it is even conjectured that the consumption of armor plate for this purpose may, during the period required for the equipment of our present vaults, exceed its consumption for the protection of vessels.

(*b*) *Nickel Steel.*—That the Bethlehem Iron Co. should have turned out only 700 tons of nickel steel forgings other than armor plate in 1898 is at first a little disappointing in view of the wonderful properties and advantages of this substance. Yet while of course it is no longer, strictly speaking, in an experimental stage, it is in the stage of forming acquaintances. Every user wishes to assure himself perfectly by somebody else's experience that the new material is really going to fill his wants. Again, while the uses to which it has been put are very important, they do not call for very great quantities of material. The chief uses appear to be for crank shafts of various kinds, piston and connecting rods, crank and wrist pins.

(*c*) *Manganese Steel.*—Most interesting developments in the use of this material has been made by the Hibbard & Rodman Safe Co., which has made some very remarkable safes, and had them tested exhaustively with the most gratifying results. The safe itself is cast in a single very massive piece out of manganese steel; and the door, also of manganese steel, is ground into the doorway with a water-tight fit, so accurately that nitro-glycerine cannot be worked into the joint. In addition to this there has been a great increase in the use of manganese steel for the toothed rolls used for crushing anthracite coal.

11. METALLOGRAPHY.—(*a*) *The Metallographist.*—In metallography the event of the year has been the establishment, by the Boston Testing Laboratories, of an admirable quarterly review of metallography and allied subjects, *The Metallographist*, under the able editorship of A. Sauveur. The fact that metallography has become so important a specialty as to warrant the publication of a magazine devoted expressly to it, speaks volumes. The clearness and

lity with which Mr. Sauveur in his previous writings had treated most subjects, which were the objects of animated and even bitter controves this new undertaking every promise of usefulness and success. mise is strengthened by the many brilliant names among his list of contributors, which represent this country, Great Britain, France, Germany, and Russia. *The Metallographist* was warmly welcomed, and has justified the high expectations that were entertained for it.

*William C. Roberts-Austen** describes his new and extremely sensitive ial arrangement of the Le Chatelier pyrometer, and illustrates the ıg cooling curve of electro iron obtained with it. The curve is ole for having, in addition to the already well-known retardation r_2 and Ar_3 of Osmond, and 1130° C. of Ball, two new retardation ne between 450° and 500° C. and one at 270° C. These two points es to be connected with the presence of hydrogen; and he makes the t announcement that the three best-known points, Ar_1, Ar_2, and Ar_3, ed in magnitude by repeated heating and cooling in vacuo, and may ed to their original magnitude by charging the iron with hydrogen. uch to suggest that these molecular changes in iron are influenced resence of small quantities of hydrogen.

ad and the Heat Treatment and Structure of Very Soft Steel.—Two Mr. J. E. Stead† contain a great amount of very valuable information ıg the effect of heat treatment on the structure and properties of very , containing say 0·025 to 0·12% C.

noff had early shown that as the temperature rises above $Ac_{1,2,3}$ (say the size of the grain progressively increases, and this had become a everyday knowledge. So had the well-established fact that reheat- c_{123} steel which had been made coarse grained by higher heating, vonderful effect of "refining" it, that is, of breaking up the grain, it became extremely fine or even porcelanic. These, then, were d facts: that heating to Ac_{123} refines, and that heating beyond that ure makes the grain coarser and coarser.

rn from Mr. Stead's investigations, however, that this law holds true teel containing a considerable amount of carbon, say 0·11% or upward, he exact limit is not yet known. If the carbon be extremely low, say less, the structure appears to follow a radically different law. The ws progressively coarser as the temperature rises from about 500° C., ad of being refined continues growing as the temperature passes Ac_1, (if indeed this point exist in this very low carbon steel), and beyond inues growing, apparently until the temperature reaches Ac_2, say From this point on no material change in size appears to take place reach a temperature of about 900° C. (Ac_3), when the coarse grain is p and becomes refined. We thus have very sharp contrasts between classes of steel. High-carbon steel is refined at Ac_{123}, and made higher temperatures. Steel with less than say 0·11% carbon is made

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h report of the Alloys Research Committee of the Institution of Mechanical Engineers.

rnal Iron and Steel Institute, 1898, I., p. 145, and 1898, II., p. 137.

coarse at about Ac_1, and is refined at a much higher temperature, say 900,° Ac_3.

It may not be possible to give a complete explanation of this at present, but the following considerations, in part speculative, may suffice to group the facts together in a simple way, for provisional use only.

α ferrite, the normal or low-temperature ferrite, grows coarser as the temperature reaches and passes the temperature of molecular mobility, 500° C., and this growth continues undisturbed past Ac_1 (say 700° C.) until, at Ac_2 (say 750° C.), α changes to β ferrite. This does not grow further with further rise of temperature, but preserves the grain-size of the α ferrite unchanged until the temperature reaches Ac_3, (say 900° C.), when the change from β to γ ferrite greatly refines the grain. But the resulting fine grain of the γ ferrite is not greatly changed by further rise of temperature.

In almost carbonless metal, such as Mr. Stead worked with, consisting almost solely of ferrite, these tendencies are in no way interfered with, and the curve which I have sketched in Fig. 3 A, mostly from Mr. Stead's data, represents them in their simplicity.

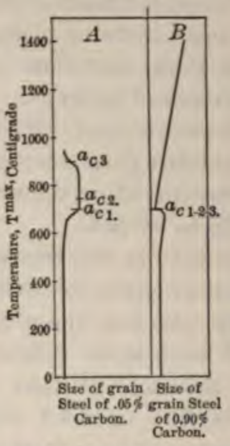

FIG. 3.—CONJECTURED RELATION BETWEEN TEMPERATURE AND SIZE OF GRAIN OF STEEL.

If, however, there be much carbon and hence much pearlite present (Fig. 3 B), while the ferrite causes the same coarsening at and above 500° C. (witness the considerable growth which here occurs on reheating refined steel), this growth is suddenly arrested when Ac_1 (say 700° C.) is reached, by the abrupt change of much or even the whole of the mass to the condition of martensite, a change which sometimes completely obliterates all pre-existing structure. Hence the refining on heating to Ac_1. On further rise of temperature the martensite in turn grows progressively into coarser and coarser grains.

What the law is for intermediate percentages of carbon remains to be seen. In connection with the refining of the grain of very low carbon steel at 900° C., noticed by Mr. Stead, it may be well to mention that some late experiments in my laboratory showed unexpectedly that steel containing 0·35% of carbon was slightly refined at about 900°.

An inference from the foregoing is, that while steel containing much carbon is best annealed at about Ac_1, this is a most objectionable temperature for annealing very low carbon steel, because it is the temperature which leads such steel to crystallize coarsely, and hence to become brittle. In particular the embrittling of tin-plate bars and the "rectangular brittleness" of thin steel sheets by such annealing are most interestingly described by Mr. Stead.

Embrittling Tin Plate Bars at 700° C.—Several tin-plate bars containing from 0·08 to 0·12% carbon, when annealed for 48 hours at about 700° C. in a closed box, became brittle, so that when bent 60° and straightened again they broke. Some indeed were much more brittle than this implies. This brittleness appears to have been due to the extremely coarse structure of the exterior of the bars, where the individual grains were as much as half an inch in diameter; for on removing the coarse exterior with a planer the interior part was found to be strong and tough. Moreover, this coarsening and brittleness were removed, apparently completely, by simply heating the bars to 900° C. and allowing them to cool naturally. After annealing the exterior of the bars contained only traces of carbon, and this may explain why the decarburized exterior was made coarse and brittle at 700, while the more highly carburized interior was not.

Rectangular Brittleness in Annealed Sheets.—More striking still is the "rectangular brittleness" which Mr. Stead discovered in annealed sheets of this extra soft steel. He found that about 1% of all the sheets of dead soft steel, between 10 and 20 B. W. G., were "rectangularly brittle" when annealed, although after rolling but before annealing none of them were brittle, and although this brittleness never occurs in thinner sheets, of 22 to 30 B. W. G. Moreover, that not only 1% but all annealed sheets of this carbon-content and between 10 and 18 B. W. G. developed this peculiar rectangular brittleness if hammered between 400° and 500° C. and then cooled slowly.

What, now, is "rectangular brittleness?" It seems that these very low-carbon steel sheets, when annealed, often have a very coarse structure, or, more explicitly, consist of coarse grains, each of which has a very strongly marked cubic cleavage, so that it is readily split along that cleavage in either of two directions at right angles with each other, although in other directions that same grain may be very ductile. When, as is usual, the adjoining grains differ in orientation, or in other words when their cleavage is not parallel, a rift started along a weak cleavage plane of one grain does not readily propogate across the adjoining grains, but is likely to be arrested on reaching the edge of the grain in which it started. But if the cleavage of the great majority of the various grains throughout the sheet is approximately parallel, then such a rift, started in a weak cleavage plane of one grain, readily passes along the cleavage plains of the adjoining grains, and so on across the sheet, which thus has extreme clĕavage brittleness in that direction; and as, according to crystalline symmetry, the cleavage is alike in two directions at right angles with each other, the whole sheet breaks readily in these two directions and is called by Mr. Stead "rectangularly brittle."

Now it happens in about 1% of the cases that the orientation or direction of the cleavage is nearly constant throughout the whole, or at least through large areas of the sheet, and thus it happens that about 1% of the plates made are rectangularly brittle. In these cases the cleavage and its consequent brittleness are at an angle of 45° with the length of the sheet, *i.e.*, with the direction of rolling.

The individual grains are often very large, sometimes half an inch, or even an inch, in diameter.

As we should expect, the rectangular brittleness disappears together with the coarse structure, on simple heating to 900° C.

We have already seen why the brittleness is only in the direction of the cleavage, and noticeable only when the cleavage of the different grains is parallel; and also why the brittleness and the coarseness of grain are broken up at 900°. It remains to point out briefly why the cleavage should be at an angle of 45° with the direction of rolling, and why the brittleness should be developed, not in the rolling but in the annealing.

If the powerful forward pull of the rolls were to develop a cleavage in any direction, that direction would naturally be at an angle of 45° with that pull, for the same reason which causes the rupture-facets of a tensile test-piece to lie at an angle of 45° with the pull which tears that test-piece in two.

The rolling having developed or determined the orientation of the cleavage, it is not surprising that, when the grains in annealing grow coarser, they should adhere to the orientation given in rolling. That is to say, the direction of rolling determines the injurious orientation, and the coarsening effect of the annealing at 700° follows the evil orientation already set up. The direction of the cleavage is due to the rolling; the coarseness to the annealing.

The Equiaxing Temperature of Steel.—The grain of steel normally is equiaxed; that is to say, the axes of any individual grain are approximately equal. When the metal is worked cold the grains are of course distorted and drawn out in one direction more than in the other. When the metal is heated to a red heat, and the molecules are thus made mobile, the grains resume their equiaxed state. It had been reported, on manifestly insufficient evidence, that this equiaxing could not occur below 750° Ac_2; and this critical point thus had been reported, on this same insufficient evidence, to be the "crystallizing point of iron." Mr. Stead completely disposes of this by showing that, if reasonable time be allowed, the grain becomes equiaxed even at 625° C.; and at a temperature barely visibly red, one at which no scale formed on iron during an exposure of two months, and evidently much below 600° C., the size of the grains of steel increased eight to ten fold, showing that crystalline changes can here occur.

Brittleness of Phosphoric Iron.—Mr. Stead gives a very interesting account of the structure of iron containing 0·75% of phosphorus. By blows he was able to break up the grains into almost perfect parallelopipeds, and the cubic cleavage appears to have been very strongly marked.

Burnt Iron.—An interesting study of the structure of "burnt" steel was made, and one which perhaps will enable us to give a good definition of that substance—a thing which we have not yet had. Mr. Stead corroborates M.

Osmond's observation that when, by extreme overheating, the granules have reached their maximum size, they separate, or yawn apart. He finds that gases formed in their interior then cause such pressure as to create blisters or excrescences (shall we not rather say that the metal is so weakened by its approach to the melting point that it is blistered by these previously imprisoned gases?) immediately below which the granules are very loosely held together, and their surfaces are more or less tarnished. Here then we may consider the steel as actually burnt, *i.e.*, dehiscent, distorted by the evolution of gas, and oxidized. But close by, where the temperature must have been almost the same, and where I should judge that the grain had been almost as coarse, but where no actual dehiscence, excretion or oxidation had occurred, on reheating to about 900° C. and cooling slowly the original fine structure was restored. Of course such reheating, although it broke up also the large oxidized grains which had been distorted by the escape of gas, and refined them, did not make good the crevices due to the separation of the grains, nor did it remove either the excrescences due to the escape of gas or the oxide films.

It has been the custom to speak rather loosely of overheated steel and burnt steel, without a definite basis of distinction between them. Making use of these observations of Messrs. Osmond and Stead, may we not say that "burning is overheating carried to such an extent that crevices open between the grains, or that excrescences are formed by the evolution of gas, or both; and that overheating, no matter how greatly it may enlarge the grain, is not burning unless one or both of these phenomena takes place."

He gives interesting data, showing how very injurious actual burning is, and how harmless, comparatively speaking, overheating is, even when it closely approaches burning. Thus in closely adjoining parts of the same "bulb angle," (of 0·235% of carbon), the burnt part had an elongation of only from 1 to 14% in 8 in., while the very coarse-grained but unburnt adjoining part had an elongation of 24% in 8 in., indeed was practically the same in both tenacity and elongation as the parts which had not been overheated.

This tallies well with the results of some experiments just finished in my laboratory by advanced students, in which heating steel of 0·35% carbon to 1300° C., and holding it at 1200° C., while it enlarged the grain greatly, yet did not seriously affect either the tenacity or the ductility.

*D. Carnot and Goutal** continue their researches into the condition in which silicon and chromium exist in iron and steel, by dissolving away part of the metal by different special solvents which fail to attack certain definite compounds. These remain behind as a residue and are thus isolated. They had already recognized the distinct substance FeSi in ferrosilicon, and they now find both in ferrosilicon, and also in three specimens of ferromanganese, the compound Fe_2Si, which Moissan had already made in the electric furnace. In silico spiegels, on the contrary, they find the compound $(FeMn)_3Si$, which is analogous to the Fe_3C of cementite.

Still more interesting are the substances which they find in chrome steel and

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* *Comptes Rendus*, May, 1898, p. 1,240.

ferrochrome. In the former, and also in a ferromanganese containing 0·97% of chromium, they find the compound $3\ Fe_3C,\ Cr_3C_2$, which is thus the familiar Fe_3C of cementite together with the carbide of chromium Cr_3C_2, prepared by Moissan in the electric furnace. In ferrochrome they find a compound made up of these two components, but in different proportions, viz., $Fe_3C,\ 3Cr_3C_2$. The difference between the two is obviously that, in the chrome steels containing but little chromium, we have but one equivalent of chromium carbide to each equivalent of iron carbide; whereas in the ferrochromes, rich in chromium, we have three equivalents of chromium carbide to each equivalent of iron carbide.

IRON ORES OF THE POTSDAM FORMATION IN THE VALLEY OF VIRGINIA.

In view of the great increase in the demand for iron ores it is strange that so little attention has been paid to the deposits of the Valley of Virginia. Some valuable facts in relation to these ores have been collected by Mr. Charles Catlett* who also notes the fact that during the years of depression in the iron trade 1893–97, no group of furnaces in the United States continued so steadily and generally in blast as that supplied by the Valley ores.

The ores of the Potsdam formation are found in a greater or less state of development along a definite horizon for a distance of about 150 miles. They are generally found in beds surrounded by heavy clays and shales, overlying the quartzite which marks the western boundary of the Blue Ridge. There are five mines of considerable size and reputation opened on some of these deposits—the Upton of the Crozer Iron Co., the Rorer, the Lynchburg, the Buena Vista and the Shenandoah Furnace Co's mines. Besides these there are a number of smaller mines. A comparison of analyses and of results obtained from these ores shows that their tenor in metallic iron is from 40 to 45%. It is quite possible that by care in picking, by jigging and by excluding recognized lean ore the average tenor of washed and prepared ores could be raised to 50% metallic iron; at least in some of the mines now opened. In places these ores are highly manganiferous, and there have been some deposits of high-grade manganese ore found in the same formation, and closely associated with iron ores. The silica in the ore occurs almost entirely combined, in silicates representing almost all grades from pure clay up to a clayey iron ore. So much clay is usually associated with the ores that washing is everywhere necessary. The phosphorus in the ores varies. In some of the mines near Roanoke it runs from 0·6 to 0·8%; further north it is much less. At the mines of the Shenandoah Furnace Co. large quantities of ore have been taken out in which the phosphorus varied between 0·20 and 0·25%. The iron made in the district has been entirely foundry and forge iron, no Bessemer pig having been turned out. Under the conditions of labor, etc., at present prevailing in the Valley of Virginia the cost of mining in most localities is not over $1 a ton of washed ore. The workable deposits are somewhat irregular, but their continuity has been proved by boring in many localities.

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MANUFACTURE OF NICKEL STEEL IN THE UNITED STATES.

BY TITUS ULKE.

THE use of nickel for improving cast iron was suggested first by Poole in 1845, and its addition to steel, it is said, was first proposed by Kelk in 1857. Percy, in his *Metallurgy*, published in 1864, devoted several pages to a discussion of the effect and value of using nickel with iron, and stated that Faraday and Stoddart had alloyed nickel with iron in various proportions. It is known also that in 1832 Wolf of Schweinfurt attempted to prepare such alloys industrially, and that ferronickel was first made in notable quantities in 1853. However, the first important American patents on the commercial manufacture of alloys of nickel and cast iron or steel were not granted until November 19, 1889, the patentee being the late Henri Schneider, then manager of the great Creusot works in France. It was at that time demonstrated by Schneider, Dr. Riley and others that a moderate amount of nickel (about 3·25%) so increased the toughness of steel of a given hardness or tensile strength as to greatly add to its resistance to cracking from the shock of impact of projectiles. The nickel, it was shown, acts as a hardener, replacing part of the carbon of the steel, and with a given tensile strength increases somewhat the elongation, and to a greater degree the contraction of area at the point of fracture. Its effect upon the elastic limit of steel is, however, of the greatest importance, since it raises this quality in a marked degree relatively to the tensile strength, and thus insures a combination of elastic strength and ductility, or toughness, unknown in any other simple alloy. Even the Krupp armor plate alloy is essentially a nickel steel, since it contains 3·5% nickel, 1·3% chromium, not less than 0·2% carbon, and not over 0·4% manganese, 0·07% copper, 0·03% phosphorus or sulphur, and 0·15% silicon; while the composition of the Creusot plates is similar excepting that tungsten, molybdenum or vanadium is substituted for chromium.

Schneider's Process.—Schneider recognized the extreme difficulty of obtaining a homogeneous mass by directly incorporating metallic nickel with steel and therefore made use of ferronickel, and thus laid the foundation of the present French practice. His patents (United States Nos. 415,655 and 415,657) cover the process of making a homogeneous alloy of steel and nickel, by charging either cold or molten ferronickel, composed say of 68% iron, 30% nickel, and 2% manganese and carbon, into a furnace containing the usual ingredients for producing steel and continuing the operation in the usual way. The ferronickel thus employed is formed by charging a suitable furnace with cast or pig iron, fuel and nickel (or a composition containing nickel) and melting them together under a cover of anthracite.

In making nickel steel free from flaws in a reverberatory furnace, it is necessary to avoid oxidizing the nickel before it forms the alloy with the iron, and to cause the incorporation to take place as soon after the commencement of the fusion as is practicable. Schneider protects the bath from oxidation by means of a layer of slag or cinder and takes special precautions to prevent redshortness in the metal before the final introduction of the recarbonizing, silicon bearing spiegeleisen or ferromanganese. The steel thus manufactured in France usually contains from 3 to 5% of nickel, a quantity amply sufficient to impart a remarkable degree of strength to the product, and specially adapting it for use in the construction of rifles and armor plate. The percentages of carbon, silicon and manganese in the steel can be regulated according to the degree of hardness required, but in every case, in order to obtain the best results, the product must be tempered in oil.

Marbeau's Process.—United States Patent No. 457,205 (August 4, 1891) to Henri Marbeau of Paris, France, covers the use of nickeliferous spiegeleisen, containing preferably 72% iron, 20% nickel, 5% manganese and 3% carbon, but free from sulphur and silicon, in the manufacture of nickel steel. According to the inventor, either an open hearth furnace or a converter may be employed in applying this process.

Wood's Process.—E. F. Wood, of Munhall, Pa., assignor to Carnegie, Phipps & Co., on June 14, 1892, obtained a United States patent (No. 476,913) for the process of making nickel steel, by charging bricks composed of nickel oxide and carbonaceous matter upon a furnace hearth (on which may first be placed a layer of limestone), superimposing pig iron thereon, and then heating the same, thereby causing reduction of the oxide and decarburization of the iron and production of nickel steel. The nickel oxide bricks are prepared by mixing the pulverized and dried nickel oxide with about one-third its weight of powdered charcoal or coke, and forming the mixture into a plastic mass with a sufficient quantity of some binding material, such as tar or sodium silicate. This plastic mass is then compressed into solid bricks in a hydraulic press. The compacting of the material is necessary, because the bricks can then be more readily kept immersed in the melted metal, with which they are brought into contact. An allowance of 10% is made for the metallic nickel which passes into the slag during the steel making and is thus temporarily lost.

The application of the above process at the Homestead works of the Carnegie Steel Co. is described as follows: An open hearth furnace is first suitably heated, and a proper proportion of the nickel oxide bricks, or nickel oxide powder put up in sheet iron boxes, is deposited on the hearth. The charge of pig metal is then mixed with the bricks or boxes in such a manner as to prevent them from rising to the surface of the metal as it melts. The open hearth process, is then carried on in the usual manner, *i.e.*, the pig metal is first decarburized, subsequently recarburized and finally purified by the addition of spiegeleisen or ferromanganese. In pouring, a very small quantity of aluminum is added to the steel to prevent it from rising in the mold. The above process formerly employed at the Carnegie works in making steel containing 3·25% nickel for government armor plate, has now been discarded in favor of the direct method of making nickel steel in which metallic nickel is used instead of the oxide.

Smelting in Cupola Furnace.—Nickel iron alloys can easily be prepared in a blast furnace, according to Riley and Wedding, from a mixture of oxide ores of iron and of nickel. The metal produced in this way is high in carbon, and is not suitable for conversion into malleable nickel-iron poor in carbon. On puddling such a carbon-nickel-iron, nickel oxides are formed, which are dissolved by the metal and render it brittle to a much greater degree than does dissolved oxide of iron. To make a malleable nickel-iron alloy, therefore, it is best to add metallic nickel to the molten malleable iron. Nickel and iron do not form true alloys by themselves, but only mixtures, and to render these mixtures homogeneous, some third metal, such as manganese, magnesium or aluminum, must first be added.

The Lechesne process for making nickel-steel, employed by the Société du Ferronickel, France, consists primarily in the simultaneous employment of nickel, manganese and aluminum, either with or without the addition of carbon. As an example, the following amounts are preferably used: Ferronickel (with 25% nickel) 100 parts, malleable iron or steel 400, ferromanganese (with 75% manganese) 3, and aluminum 2·5. The ferromanganese is added after decarburization, and the aluminum at the moment of tapping. The nickel steel alloys thus made are homogeneous, and solidify rapidly without blowholes.

Open-Hearth Process with Metallic Nickel.—In the United States at the present time nickel steel is made almost exclusively in the open hearth furnace by adding metallic nickel to the steel bath, practically the whole of the nickel, as well as that of any scrap nickel steel charged, being recovered. Metallic nickel offers this advantage over the oxide, that less nickel slags off. The open hearth heat lasts about six to seven hours, and a final addition of ferromanganese is made as usual. This process is the only one now employed at the Bethlehem and the Carnegie works in making nickel steel for armor plate, shafting and marine engine and rifle forgings purchased by the United States.

Production of Metallic Nickel.—In 1897 at least 400 tons of electro-refined nickel, it is stated, were consumed in the production of nickel steel in the United States alone. The commercially pure metallic nickel employed in steel manufacture should not contain over 0·1% copper and more than traces of sulphur. It is generally secured either in the electrolytic way, as in the United States, or is obtained as in Europe by reducing pure nickel oxide into metal. The nickel obtained by the latter method is not always a solid metal, but often a more or less agglomerated metallic powder, which can be melted into bars, according to the German nickel makers, Basse & Selve, by fusing it in the presence of a small quantity of nickel oxide. Whatever excess of oxide is dissolved by the molten nickel can then be reduced at the moment of casting by the addition of a little magnesium. Solid nickel has also been obtained by the simultaneous addition of both manganese and magnesium.

Uses of Nickel Steel.—Nickel steel is now largely used in the manufacture of high grade bicycle tubing, which practice was introduced, I believe, three years ago, by the Pope Manufacturing Co. At its works steel containing between 3 and 5% nickel, from the Bethlehem Iron Co., is received in the shape of plates about 2 ft. long, 1 ft. wide, and 0·1 in. thick. These plates are first punched into disks, about 1 ft. in diameter, in a blanking machine weighing 4 tons, and are then put through a number of heavy hydraulic presses, in which the disks are forced through dies by powerful rams. The first operation brings the disk to the shape of a hollow basin, the next makes it an elongated cup, the next increases its length and so on, until finally it is reduced to the form of a tube, 2 ft. or more in length. Then the rounded end of the tube is sliced off, and the nickel steel is in the form of a "billet," ready for the swaging machines and drawbenches. At the drawbenches the "billets" are placed in oil troughs and fed, one by one, to the dies, through which they are dragged by clutch tongs attached to powerful hydraulic pistons. Each tube undergoes successively a number of drawings, smaller and smaller dies being used unti the desired thinness has been obtained. Between each two drawings the tubes are sent to the annealing furnaces to be softened, so that they will reduce more easily. For the annealing furnaces, from 6 to 100 tubes, according to the size are placed in iron cylinders or "retorts," 9 in. in diameter and 14 ft. long and tightly screwed inside by end pieces. The retorts are then preferably exhausted of their air or filled with an inert gas and sealed up at the end This is done for the purpose of keeping the tubes from direct contact with the flames and oxygen, which would otherwise form a scale upon the surfaces, and necessitate the subsequent and injurious operation of "pickling" the tubes in large vats of acid in order to remove this scale.

The retorts, each of which weighs about 1,000 lb. with its load of tubes, are now brought on cars to the annealing furnaces and charged into the latter The annealing usually takes about 45 minutes, but as every tube has to be submitted to this process from six to a dozen times, it frequently happens that a period of two or three weeks elapses between the making of the steel cups and the final completion of the tubes. From the annealing furnaces, which usually hold three retorts each and are fired by naphtha gas aided by an air blast, the retorts are rolled on rails across to the muffles, where they are left to cool. This takes as much as five, six or seven hours. The heavy cylinders with their loads still inside, are finally swung across the room by means of an overhead trolley to the place where the tubes are to be piled, cleaned, straightened and cut into such lengths as are needed in making bicyles.

Next to its use in armor, perhaps the most important application of nickel steel is in the manufacture of propeller shafts, crank and thrust shafts, connecting rods, piston rods and other marine engine forgings. Heavy shafts of fluid-compressed nickel steel, hollow forged and oil tempered, are especially made in the Whitworth fluid compression and hydraulic forging plant of the Bethlehem Iron Co., at South Bethlehem. They are said to be far superior in their physical qualities to the forgings of mild steel heretofore used for the same purpose, and to fully justify their higher price. Examples of this application of nickel steel are to be found on the United States battleship Iowa and cruiser Brooklyn, and on the American Line steamships St. Louis and St. Paul. The shafting and engine forgings for the two new battleships Kearsarge and Kentucky are likewise of nickel steel.

Steel rich in nickel is considered to be practically non-corrodible, and therefore to be admirably adapted for parts of machinery which are usually kep bright, but which are liable, if made of ordinary steel, to rust quickly from the moisture of the hand, such as hand wheels, keys and levers.

LEAD.

The production of pig lead in the United States showed a large increase in 1898, the grand total of soft, desilverized and antimonial smelted from both domestic and imported ores and refined from imported crude bullion, amounting to 317,684 short tons, against 289,835 short tons in 1897. The production for 1898 was distributed as follows, the corresponding figures for 1897 being given in parentheses: Desilverized domestic lead, 169,364 (144,649); soft, 50,468 (45,710); antimonial lead, 8,643 (7,359); total domestic, 228,475 (197,718); foreign, 89,209 (92,117). The increase in the domestic production is traceable especially to the various lead-producing districts of Colorado, while Missouri also turned out considerably more than in the previous year. Strange to say, there was a small decrease in the output of the Cœur d'Alêne and Montana fell off considerably.

PRODUCTION AND CONSUMPTION OF LEAD IN THE UNITED STATES. (IN SHORT TONS.)

Year.	Stock of Refined, Jan. 1.	Produced from United States Ores.				Imported in Ores and Bullion.	Total Supply.	Consumed.	Exported in all Forms.	Stock of Refined, Dec. 31.
		Desilverized.	Soft.	(b) Antimonial.	Totals.					
1894....	5,991	119,486	37,044	4,337	160,867	69,741	236,599	191.146	40,000	5,453
1895....	(a) 12,634	119,057	32,797	5,000	156,854	100,471	269,959	232,854	18,130	(a) 18,975
1896....	(a) 18,975	135,332	33.428	5,932	174,692	80.159	273,826	203,819	56,983	(a) 13,024
1897....	(a) 13,024	144,649	45,710	7,359	197,718	92,117	302,859	211,345	60,353	(a) 31,161
1898....	(a) 31,161	169,364	50,468	8,643	228,475	89,209	348,845	246,989	78,168	(a) 23,688

(a) Includes lead in bond, which amounted to 7,345 tons at the end of 1898, 6,694 tons at the end of 1897, 4,124 tons at the end of 1896, and 9,865 tons at the end of 1895. (b) The entire production of antimonial lead is entered as of domestic production, although part of it is of foreign origin; it is, however, impossible to separate this in the statistics; owing to this inability the division of the American production between antimonial and desilverized is not quite accurate, though the error is not important.

In 1899 the production of lead will be controlled largely by the American Smelting and Refining Co., which was organized in March to take over the business of the smelters who have heretofore treated the more part of the silver-lead ore produced in the United States, British Columbia and Mexico, the consolidation taking in the United Smelting and Refining Co. (Helena and Great Falls, Mont.), Omaha & Grant Smelting Co. (Denver, Colo., Durango, Colo., and Omaha, Neb.), Pueblo Smelting and Refining Co. (Pueblo, Colo.), Colorado Smelting Co. (Pueblo, Colo.), Hanauer Smelting Co. (Salt Lake City, Utah), Pennsylvania Lead Co. (Mansfield, Pa.), Pennsylvania Smelting Co (Salt Lake City, Utah), Globe Smelting and Refining Co. (Denver, Colo.), Bi-Metallic Smelting Co. (Leadville, Colo.), Chicago & Aurora Smelting Co (Chicago and Aurora, Ill.), Germania Lead Works (Salt Lake City, Utah), an a majority of the stock of the Consolidated Kansas City Smelting and Refinin Co. (Leadville, Colo., El Paso, Tex., and Argentine, Kan.), the new concer being capitalized at $65,000,000, divided into 650,000 shares, of which one-hal is preferred stock entitled to cumulative dividends of 7% per annum. Provision was made for a working capital of $7,500,000 in cash and $2,000,000 in securities acquired from some of the companies purchased. The consolidation included practically all the silver-lead smelting interests of the United States with the exception of the Guggenheim Smelting Co., of Perth Amboy, N. J., and the Balbach Smelting and Refining Co., of Newark, N. J., which are refiners rather than smelters and work chiefly on Mexican silver-lead, the Philadelphia Smelting and Refining Co., of Pueblo, Colo., the Selby Smelting and Lead Co., of San Francisco, Cal., the Puget Sound Reduction Co., of Everett, Wash., and the works at Tacoma, Wash.

Arizona.—The lead production of this Territory has been small heretofore, but it is expected that it will be increased soon by the mines in the Chloride district, Mohave County, where there are large veins of silver-lead ore. Deposits of silver-lead ore are also being opened in the Santa Teresa Mountains, about 50 miles N. W. of Wilcox.

Colorado.—According to the statistics of H. A. Lee, State Commissioner of Mines, the production of lead in Colorado in 1898 was 56,708 short tons, against 40,400 in 1897. The production of Leadville was 17,973 tons, against 11,850 in the previous year. That of Pitkin County, chiefly Aspen, increased from 2,228 tons to 7,951; that of San Juan County increased from 4,010 tons to 7,330; while that of Hinsdale County (Lake City) increased from 2,775 to 4,914. The remainder of the production was distributed among most of the other counties of the State, Clear Creek (Georgetown), Mineral (Creede), San Miguel and Summit (chiefly Kokomo and Robinson) being the most important producers. Developments in these districts are referred to especially in the review of gold and silver mining elsewhere in this volume.

Idaho.—The production of lead in the Cœur d'Alêne in 1898 was 56,339 tons, against 57,777 in 1897. The output of silver was 3,399,524 oz. in 1898, against 3,579,424 in 1897. The statistics of silver and lead production in this State are based on direct returns from the mines, with an allowance of 6% for loss of lead in smelting and 2% for loss of silver. Outside of the Cœur d'Alêne the lead production of Idaho in 1898 was very small. The Clayton Mining and Smelting Co. operated a 50-ton furnace at Clayton, smelting ores chiefly from the Skylark and Red Bird mines owned by the company. Charcoal is used for fuel. The silver-lead is shipped from Blackfoot.

In the Cœur d'Alêne there was great activity in 1898 and in general these mines are showing up well. The production is made chiefly by nine concerns. The ore, which is low grade, is concentrated mechanically to a grade which yields an average of about 50% lead and 30 oz. silver; consequently at existing prices its lead value considerably exceeds its silver. There are few statistics as to the dividends of the Cœur d'Alêne mines, but it is believed they are generally profitable in spite of the rather unfavorable conditions imposed by the labor unions. The Tiger mine at Burke is opened to a vertical depth of 1,200 ft., while the Helena & Frisco is down 1,600 ft. The showings in the lowest levels are said to be as good as anywhere else in these mines. Several of the mining companies are now driving long crosscut adits to the veins to intersect them at much greater depths. The Cœur d'Alêne mines employ about 2,000 men, while fully 3,000 more obtain their living from them indirectly.

Iowa.—The lead mines in the vicinity of Dubuque made their usual small output in 1898. Most of the ore is reduced at a smeltery at Dubuque, where Scotch hearths are employed.

Missouri.—The production of lead in southeastern Missouri increased from 34,255 tons in 1897 to 35,769 tons in 1898. During 1898 there were six active producers in the district, and all of them were dividend payers. A good deal of new exploration work was undertaken and some new and important discoveries were made, so there is prospect of a further increase in production from this district, which is favored by the natural conditions. The best equipped mines of Bonne Terre and vicinity can produce lead at a cost of 2·25c. per lb. The Central Lead Co. purchased the property of the Theodora Lead Co. and built a railroad to the Theodora shaft, where hoisting machinery was erected. The National Lead Co. and Columbia Lead Co. which acquired properties in the vicinity of Bonne Terre in 1897 and proved their value by diamond drilling, were sinking shafts during 1898. Both companies expect to reach the ore in 1899. The Desloge Consolidated Lead Co. increased its capital stock to $1,000,000 and made various improvements on its property. New companies intending to operate in this district are the Elizabeth Lead Co. and the Union Lead Co. The Renault Lead Co. purchased the Palmer lead mines in Washington County, and made preparations to undertake operations on a large scale at greater depths than these mines have been opened heretofore. The Catherine Lead Co. has developed a promising property near Mine la Motte.

In the Joplin district (including southeastern Kansas) lead mining was not carried on so vigorously in 1898 as in 1897 on account of the increased attention devoted to zinc mining, wherein there was more profit than in lead mining because of the high prices prevailing for zinc ore. The sales of lead ore in 1898 amounted to 26,457 short tons, against 30,105 in 1897. The average monthly prices per 1,000 lb. were as follows: January, $22; February, $22·125; March, $23; April, $21·565; May, $21·75; June, $22·875; July, $23·60; August, $23·50; September, $23·315; October, $22; November, 20·875; December, $21·10. The average for 1898 was $22·25, against $21·38 in 1897.

Montana.—The production of lead in this State in 1898 was 10,702 tons, against 12,897 tons in 1897. However, the construction of the new branch of the Northern Pacific Railway through the Lo Lo Pass will open up a district in the Cœur d'Alêne and Bitter Root Mountains, which may increase the State's production of lead in 1899. The Iron Mountain mine has been closed, after being exploited to a depth of 1,700 ft., on account of the large expense of dead work required to continue operations advantageously, although at the bottom the vein was said to be as strong as ever. At Glendale new bodies of lead ore were opened in certain of the mines of the Hecla Consolidated Mining Co., which enabled that company to run two furnaces and continue the smelting of the high-grade silver ore, of which it still has a considerable supply. The erection of a 100-ton smelting plant at Twin Bridges, Mont., was begun, and it is expected that these works, which are to be equipped with two furnaces, will be in operation in 1899. The Basin & Bay State Mining Co., operating the Katie mine, built a fine smeltery of 150 tons per day estimated capacity at Basin. These works will furnish a market for the silver-lead ores of adjacent mines and may lead to a revival of interest in the district. There was considerable activity in 1898 in the Neihart district, where the ores are more valuable for silver than for lead.

Utah.—The lead-mining industry in this State has been disturbed considerably by an increase in the freight rates on ore and base bullion from Salt Lake City to Missouri River points. Previous to January 1, 1899, the rate on bullion to the Missouri River was $11 per ton, and to Chicago $13·50, the rate on ore being $9 and $11·50 respectively. On January 1 the ore rate was increased $1 per ton and the bullion rate $2 per ton. The production of lead in Utah increased largely in 1898.

LEAD PRODUCTION OF THE WORLD. (*a*) (IN METRIC TONS.)

Year.	Austria.	Belgium.	Canada.	Chile.	France.	Germany.	Greece.	Hungary.	Italy.
1893...............	7,212	12,006	968	73	8,119	94,659	14,534	2,514	[illegible]
1894...............	7,570	14,120	2,587	87	8,758	100,753	14,434	2,113	[illegible]
1895...............	8,085	15,573	7,467	93	7,627	111,058	19,845	2,277	[illegible]
1896...............	9,769	17,222	10,977	594	8,232	113,792	15,180	1,911	[illegible]
1897...............	9,860	17,028	17,698	370	9,916	118,881	16,486	2,527	[illegible]
1898...............	....	....	14,477	....	....	132,742	19,193	....	....

Year.	Japan.	Mexico. (*c*)	New South Wales. (*c*)	Russia.	Spain.	Sweden.	United Kingdom. (*b*)		United States.	Totals.
							Foreign Ores.	Domestic Ores.		
1893....	941	64,000	58,000	844	169,318	461	6,127	30,173	151,210	641,057
1894....	1,454	57,000	50,000	743	152 620	330	9,588	30,162	145,906	617,780
1895....	1,950	68,000	38,000	412	160,786	1,256	16,036	29,464	142,298	653,596
1896....	1,958	63,000	30,000	262	167,016	1,530	25,388	33,058	158,479	677,998
1897....	(*d*) 1,950	71,637	22,000	*d* 250	*e*189,216	1,480	33,058	26,942	179,369	741,070
1898....	....	71,442	....	....	*e*193,764	....	....	....	207,271	....

(*a*) The statistics for Austria, Belgium, Canada, France, Germany, Hungary, Italy, Japan, Russia, Spain and Sweden are from the official reports of the respective governments except where otherwise noted. Those for Greece are based on the authorities given under the general table of mineral production of Greece in a subsequent part of this volume. Those for the United States are from data collected by THE MINERAL INDUSTRY. Those for the United Kingdom as specified in note *b*.

(*b*) The production of lead in the United Kingdom is given in two columns. One gives the amount of lead derived from domestic ores, the yield of which is calculated at 95%, as reported in the official British blue books. The other column gives the production of lead in the United Kingdom from foreign ores smelted there. These figures, which are not reported in the official blue books, are obtained by deducting the production of British lead from the total output of the lead smelters of the United Kingdom as stated in the *Statistische Zusammenstellungen über Blei. Kupfer, Zink und Zinn* of the Metallgesellschaft, Frankfurt-am-Main.

(*c*) From the *Statistische Zusammenstellungen* of the Metallgesellschaft, Frankfurt-am-Main. These figures comprise only the lead exported to Europe and America. The actual production of lead in New South Wales is much larger than shown here, especially in late years, during which there have been large exports to China, but it is impossible to obtain statistics as to what it really amounts to.

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Washington.—The two smelters of Washington had a hard time in competition with the Denver and Pueblo buyers, not only for the British Columbia ore, but also for that of the Cœur d'Alêne district of Idaho and the Monte Cristo district of Washington. The Puget Sound Reduction Co. put in operation its new refinery and exported a considerable amount of desilverized lead to China. The Tacoma smelting works have been purchased by the London Prospecting and Development Co. Several promising silver-lead mines are being opened in Washington; among them the Boston mine in the Cascade district, and the Vandalia in Silver Creek; it is proposed to connect the latter by an electric tramway with the Great Northern Railway at Index.

PRODUCTION, IMPORTS, EXPORTS AND CONSUMPTION OF LEAD IN THE CHIEF COUNTRIES OF THE WORLD. (IN METRIC TONS.)

		Aust'a-Hungary. (*a*)	Belgium. (*b*)	France. (*c*)	Germany. (*d*)	Great Britain. (*e*)	Italy. (*f*)	Russia. (*g*)	Spain. (*h*)	Switzerland (*i*)	United States. (*j*)
1893	Production	9,726	12,006	8,119	94,659	36,300	19,898	844	169,318	*Nil.*	151,210
	Imports	5,925	38,028	77,679	23,857	191,261	1,657	28,712	*Nil.*	3,159	58,306
	Totals	15,651	50,034	85,798	118,516	227,561	21,555	29,556	169,318	3,159	209,516
	Exports	46	29,947	9,243	23,945	30,008	1,570	*Nil.*	158,737	89	31,752
	Consumption	15,605	20,087	76,555	94,571	197,553	19,985	29,556	10,581	3,070	177,764
1894	Production	9,683	14,120	8,758	100,753	39,700	19,605	743	152,620	*Nil.*	145,906
	Imports	8,842	43,491	84,674	24,208	164,451	1,499	30,755	*Nil.*	1,412	63,269
	Totals	18,525	57,611	93,432	124.961	204,151	21,104	31,498	152,620	1,412	209,175
	Exports	84	34,690	8,133	24,354	28,246	1,157	*Nil.*	158,735	....	36,288
	Consumption	18,441	22,921	85,299	100,607	175,905	19,947	31,498	....	1,412	172,887
1895	Production	10,362	15,573	7,627	111,058	45,500	20,353	412	160,786	*Nil.*	142,298
	Imports	8,974	45,594	67,183	28,449	165,531	947	29,556	*Nil.*	3,571	91,454
	Totals	19,336	61,167	74,810	139.507	211,081	21,300	29.968	160,786	3,571	233,752
	Exports	60	39,996	10,734	27,855	22,920	3,646	*Nil.*	151,129	298	16,447
	Consumption	19,276	21,171	64,076	111,652	188,111	19,147	29,968	9,657	3,273	217,305
1896	Production	11,680	17,222	8,232	113,792	57,200	20,786	262	170,790	*Nil.*	158,479
	Imports	7,221	35,221	80,572	33,016	170,484	1,166	31,384	*Nil.*	4,350	72,720
	Totals	18,901	52,443	88,804	146,808	227.684	21,952	31,646	170,790	4,350	231,199
	Exports	88	31,366	11,275	24,828	18,976	1,419	*Nil.*	166,124	244	54,752
	Consumption	18,913	21,077	77,529	121,980	208,708	20,533	31,646	24,666	4,106	176,447
1897	Production	12,387	17,023	9,916	118,881	60,000	22,407	....	....	....	179,369
	Imports	5,887	43,840	88,050	35,092	170,120	1,178	....	....	....	84,060
	Totals	18,274	60,863	97,966	153,973	230,120	23,585	....	....	....	263,429
	Exports	55	35,988	10,448	24,075	20,749	2,790	....	....	....	54,633
	Consumption	18,219	24,875	87,518	129,898	209,371	20,795	....	....	....	308,756

(*a*) From *Statistisches Jahrbuch des K. K. Ackerbau Ministerium* and *Magyar Statisztikai Evkönyv*,

(*b*) Production from *Statistiques des Mines, Minières, Carrières, et Usines Métallurgiques.* Imports and exports from *Annuaire Statistique de la Belgique.*

(*c*) From *Statistique de l' Industrie Minèrale.*

(*d*) Production, imports, and exports from *Statistisches Jahrbuch für das Deutsche Reich.*

(*e*) Production from statistics of the Metallgesellschaft, Frankfurt-am-Main. Imports and exports from Board of Trade returns.

(*f*) From *Rivista del Servizio Minerario.*

(*g*) From *Sbornik Statisticheskikh Svedenie o Gornozavodskoi Promyshlennostie Rossie v zavodskom Godu*, St. Petersburg,

(*h*) From the *Reports of the Comision Ejecutiva de Estadistica Minera.* Imports and exports from the *Revista Minera de Espana.*

(*i*) Brom *British Statistical Abstracts* and from figures furnished by the *Direction Génèrale des Douanes Fédérales*, Berne.

(*j*) The statistics of production are those collected by THE MINERAL INDUSTRY; those of imports and exports are from the reports of the Bureau of Statistics, Washington.

Australia.—The lead production of Australia continues to be derived chiefly from the mines of Broken Hill, in New South Wales. Reference to the pro-

[illegible] of this district was made specially in the review on gold and silver elsewhere in this volume. As indicated there, the production of the Broken Hill Proprietary Co., the most important producer, is decreasing. This company has now transferred all its smelting operations to Port Pirie, the last of the furnaces at Broken Hill having been blown out April 16, 1898. The plant at Port Pirie consists of eight 120-ton furnaces and three of 80 tons per day capacity. The average yield of the ore smelted during the half year ending May 31, 1898, was 7·4% lead and 13·16 oz. silver per ton. The production of refined lead was 15,803 tons and of antimonial 344 tons. The cost of mining and smelting the ore was $10·18 per ton, including depreciation of plant. A noteworthy feature of the lead industry in Australia in 1898 was the large shipments to Great Britain, whither 42,230 long tons were sent in 1898, against 16,394 in 1897. For several years previous a large part of the Australian lead production has been shipped directly to China and Japan. There was increased activity in the Zeehan and Dundas silver-lead mines of Tasmania, to smelt the ores of which the Tasmanian Smelting Co. is erecting works. The exportation of silver and silver-lead ore from Tasmania in 1898 amounted to 15,320 long tons. There are no precise data as to the lead production of this colony, which is the most important after New South Wales. Two smelting works were erected near Zeehan in 1892, but they ran only for a short time. A small amount of ore was also smelted by Kennedy & Sons, of Hobart. However, up to the present time the total quantity of bullion exported from Tasmania has been only 1,430 long tons. At present all the marketable ore is exported. The Western and Montana companies ship directly to Genoa or England, but most of the Zeehan mines sell their ore to the local agents of foreign smelters, the principal buyers being the Hamburg Metal Co. and the Queensland Smelting Co. The Tasmanian government has no record of the silver and lead contents of the ore exported, and it is difficult to make a close estimate, since the grade varies much. The average tenor in lead is probably between 55 and 60%, however, while the silver ranges from 30 to 200 oz. per ton. There is not much gold contained in this ore.

Austria.—According to the report of the Bleiberger Bergwerks-Union for 1897 the production of the Bleiberg district since 1887 has been, in metric tons:

	1888.	1889.	1890.	1891.	1892.	1893.	1894.	1895.	1896.	1897.
Bleiberg	3,209	3,205	2,995	3,136	3,183	2,921	3,495	3,074	3,310	3,090
Kreuth	1,614	1,725	1,684	1,704	1,119	1,051	1,024	1,095	1,319	1,345
Total	4,823	4,930	4,679	4,900	4,302	3,972	4,519	4,169	4,629	4,436

Two American hearths at Kreuth produced 7,184 metric centners of lead out of 1,054·3 tons of concentrates, a yield of 68·14%, at a cost of 1 florin 17 kreutzers per metric centner of lead. Two Belgian reverberatory furnaces at Gailitz produced 1,555 tons of lead out of 3,232 tons of concentrates, or 48%, although about 1,400 tons of residues with about 40% lead remained. This uneconomical method of smelting was abandoned at the end of the year and the well-proved American method was reinstated. The Pilz furnaces produced 767 tons of lead from 1,783 tons of reverberatory furnace residues and oxidized lead products. The cost of this combined method was 3·98 florins per metric centner of lead. The cost of smelting in two American hearths at Miess was 1·01 florins per centner of lead. At Windisch-Bleiberg a Carinthian reverberatory furnace was in operation, producing 431 centners of lead at a cost of 2·01 florins for smelting and a yield of 58·3%. At Kappel-Feistritz the cost of smelting in a Carinthian furnace was 2·14 florins per metric centner.

Canada.—The production of lead in Canada in 1898 was 15,958 short tons, practically all of which was mined in British Columbia. The entire production of this province was imported into the United States for reduction, the smelters of Denver and Pueblo being the most active in the market. A small part of the ore, however, was shipped to Everett and Tacoma, Wash. Of the local works the Trail smeltery, which is now in the hands of the Canadian Pacific Railway Co., was idle, but extensive improvements were made with the intention of resuming operations. At Nelson one furnace was run by the Hall Mines, Ltd., on ores mined in that vicinity. This company intends to enter more extensively into the smelting of silver-lead ores from the Slocan district. The small works at Pilot Bay, Kootenai Lake, owned by the Braden Bros., were not in operation in 1898.

Chile and Bolivia.—Silver-lead ores are smelted at Antofagasta, where there are two works, one at the suburb Bella Vista, owned by the Antofagasta Smelting and Amalgamating Co., and the other at Playa Blanca, owned by the Compañia des Minas da Huanchaca. The ores are obtained from San Felipe, between Valparaiso and Santiago, Coquimbo, Serena, Vallenar, Caracoles, Sierra Gorda in Chile, and at Huanchaca, Oruro and elsewhere in Bolivia. The more part of the ore supply for the Bella Vista works comes from Caracoles, which is situated about 150 miles inland from Antofagasta. The ores are carried on mules to Sierra Gorda, a distance of 30 miles, and thence by railway to Antofagasta. The ore is chiefly free-milling silver chloride with a calcareous gangue. The lead ores received at Bella Vista are chiefly carbonates and sulphates from Chile and sulphides from Bolivia.

China.—According to the report of the French Commercial Mission the Province of Yun-Nan now exports annually 2,600 tons of lead. Silver-bearing galena occurs in this province in veins in limestone.

Greece.—The production of silver-lead ore at Laurium in 1897 was 9,660 metric tons; galena concentrates, 6,915; mixed zinc-lead ore, 3,084; lead fume, 2,508. This does not include 409,085 tons of low-grade ore containing 3 to 5% lead which was dressed and yielded 16,074 tons of pig lead containing from 2,100 to 2,915 g. of silver per ton. In 1898 the production of soft lead in Greece was 305 metric tons; silver-lead 18,888, and lead fume, 2,655.

Mexico.—The production of lead in Mexico in 1898 was 78,751 short tons, against 78,966 in 1897, about 90% of the production each year being exported to the United States. The Compañia Minera de Peñoles, of Mapimi, Durango, which is the most important single producer in the republic outside of the Sierra Mojada district, considerably increased its output in 1898, a part of this product, which formerly was entirely sent abroad, being exported to the United States.

Servia.—The Société Francaise Miniere et Metallurgique en Serbie has purchased the silver-lead mines of Cervenibreg, near Vranja, and it is said will exploit them on a large scale.

Spain.—According to Señor Adriano Contreras, in the statistical compilations for 1898 of the *Revista Minera*, the production of lead in Spain in 1898 was 193,764 metric tons, an increase of 4,548 over 1897. Of this production 8,500 tons were consumed in Spain, 179,885 tons were exported in pigs, 979 tons in manufactured form, and 4,400 tons in ores. The total production of lead ores was 352,000 tons.

Turkey.—According to R. Helmhacker,[*] lead ores are common in European Turkey. There are occurrences in the islands of Imrus and Semidrek. In Armenia lead ores have been found and mined near Van and in several adjoining districts. Near Gulek lead mines were formerly worked, the deposits occurring as lenticular masses in Cretaceous limestones. The only lead mines now in operation in Turkey are those at Lijessi, near Karahissar, and at Gamlibel, near Enderes, both of which are operated by the Asia Minor Mining Co.

Large deposits of silver-lead ore exist in the Province of Aidin. One mine in the vicinity of Jeniköï is said to be particularly rich, but the vein occurs in very hard granite and the native miners have not yet succeeded in working it successfully. An analysis of silver-lead produced at Jeniköï showed 559 oz. silver per ton. There are lead mines in Samos which ought to be very productive, and are now about to be exploited by a Belgian company.[†]

United Kingdom.—A new discovery of silver-lead ore was made in 1898 at Douglas, in the Isle of Man.

THE LEAD MARKETS IN 1898.

New York.—The range of fluctuations in the lead market in 1898 was not as wide as in 1897, although the market was constantly moving up and down. While at no time the very high prices ruling during the fall of 1897 were reached, the average price in 1898 was higher. In 1897 the large advance was due chiefly to the increase in the duty, while during 1898 there was no change in the tariff and the market was governed solely by supply and demand. Prices reflect this condition, for while production increased consumption increased to about the same extent, and at the end of the year stocks were smaller than at the beginning.

January opened with large quantities in the hands of producers as well as consumers, and the metal was quoted at 3·70c. New York. Some parcels being pressed for sale, the market declined rapidly to 3·65c. and then to 3·60c. However, early in February, buyers' stocks having become depleted, they bought heavily, but were unable to secure large quantities on the basis of 3·55c. New York, before the market advanced again to 3·60c. Considerable quantities were still held in the East, while Western stocks had run low, in consequence of which the St. Louis market ruled comparatively

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[*] *Engineering and Mining Journal*, Nov. 26, 1898

[†] *Montan Zeitung*, Sept. 1, 1898.

higher than the Eastern. About the middle of February there was a marked improvement in the demand, and the market rose to 3·85c. New York. The higher prices, however, checked business and buyers generally held off, and the market receded to 3·75c. New York and 3·57½c. at St. Louis. During March the market moved sluggishly. Rather large quantities in the hands of smelters hung over the market, and while higher prices were asked, little business was done, manufacturers buying only from hand to mouth. The New York market during the month was about 3·70c. and St. Louis about 3·52½c.

Early in April the market was decidedly top-heavy and prices fell to 3·57½c. New York and 3·40c. St. Louis. At the end of the month, however, the low prices attracted attention of manufacturers as well as speculators, and there was a scramble to purchase May and June shipments. Prices quickly reacted to 3·70c. and 3·52½c. respectively. In May the higher prices again brought out large quantities, and had it not been for some government orders and speculative purchases prices would have suffered considerably. As it was they declined to 3·62½c., but reacted to 3·67½c., the month closing at that figure. June witnessed very heavy buying on the part of consumers, as well as for speculative account, and the market advanced quickly to 3·75c., then to 3·85c., and finally to 3·92½c., with lead selling at St. Louis at 3·82½c.

In July it became apparent that consumers had not yet covered their requirements and considerable quantities of prompt lead changed hands at 3·95@4c New York and 3·85c.@3·90c. St. Louis. At the end of the month, however, some secondhand parcels came out, which had a somewhat depressing influence on the market, and the metal was quoted 3·92½c. and 3·82½c. respectively. In August there was great activity and large buying both East and West, and prices advanced to 4·10c. New York and 4c. St. Louis, the highest prices of the year. During the first half of September the market was quiet with a flattish tendency prevailing, and while considerable business was done, somewhat lower prices were established. There was more buying about the middle of the month, but the market soon relapsed into dullness, September closing with sellers at 4c. New York and 3·85c. St. Louis.

AVERAGE MONTHLY PRICES OF LEAD IN NEW YORK.

Year.	Jan.	Feb.	Mar.	Apr.	May.	June.	July.	Aug.	Sept.	Oct.	Nov.	Dec.	Year
	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.
1893	3·87	4·22	3·96	4·08.	3·89	3·77	3·58	3·41	3·80	3·51	3·41	3·27	3·73
1894	3·19	3·31	3·37	3·43	3·39	2·31	3·50	3·41	3·17	3·12	3·14	3·10	3·29
1895	3·10	3·12	3·12	3·08	3·16	3·25	3·25	3·50	3·35	3·33	3·25	3·22	3·23
1896	3·08	3·19	3·14	3·07	3·03	3·03	2·96	2·73	2·77	2·80	2·96	3·04	2·98
1897	3·04	3·28	3·41	3·32	3·26	3·33	3·72	3·84	4·30	4·00	3·76	3·70	3·58
1898	3·65	3·71	3·72	3·63	3·64	3·82	3·95	4·00	3·99	3·78	3·70	3·76	3·78

During October there was considerable pressure to sell and very little buying. Prices declined steadily and transactions were made at 3·60c. New York and 3·50c. St. Louis. At these low figures, however, consumers again bought largely, and in November the market advanced to 3·75c. New York and 3·62½c. St. Louis, but their wants were readily satisfied and further quantities being pressed for sale, the metal declined to 3·65c. New York and 3·52½c. St. Louis.

During the first half of December, the market was quiet and weakish, but about the middle of the month a large demand suddenly sprung up, and with large transactions prices advanced to 3·90c. New York and 3·75c. St. Louis.

London.—The market opened in January with good soft foreign ruling at £12 7s. 6d. to £12 8s. 9d., English being worth £12 8s. 9d. to £12 11s. 3d., but buyers came in and the prices soon rose to £12 12s. 6d. for foreign. When consumers' wants had been supplied values receded, and the month closed at £12 7s. 6d. to £12 8s. 9d. February opened with buyers still holding aloof, and prices declined to £12 6s. 3d.; at this point things improved and more demand was noticeable, values rising to £12 8s. 9d. American sellers then raised their prices above the level of those ruling in London, and in March, when buyers were forced to cover and it was found that American producers had sold out, there was a very strong tone, and soft foreign was soon selling at £12 16s. 3d. The import duty passed in France had also a stimulating effect, but the closing quotations for foreign were £12 13s. 9d. to £12 15s. April brought the possibility of a great curtailment in supplies owing to the threatened hostilities between Spain and America, and this led to a good deal of nervousness on the part of consumers, who rushed in and bought heavily, causing prices to go up by rapid strides, and the market—after sales were made as high as £14 15s.—closed with buyers at £14 7s. 6d., sellers asking £14 12s. 6d., and but very little offering at that. May opened amid renewed excitement at £14 12s. 6d., but when war was declared, and it became apparent that shipping would not be seriously interfered with, there was more caution displayed by buyers, and values suffered through neglect until the end of the month, when £13 10s. was quoted. June opened with buyers still very reserved and only buying from hand to mouth. Trade at this time was not good, and new orders were slow in coming in; the market for pig consequently relapsed, and before the month was out sellers had accepted £13.

About the commencement of July buyers plucked up courage, and Continental users came in and made good purchases, £13 3s. 9d. being paid. Having covered their requirements for the time being, they again left the market alone, and caused values to again move downward until £12 17s. 6d. was accepted; then another spurt took place to £13 2s. 6d., followed, however, by a further shrinkage to £12 13s. 9d. This latter price was ruling at the commencement of August, but it was soon evident that consumers' stocks were seriously diminished, and when they came into the market to buy the price improved to £12 17s. 6d. There were at the time some good export orders, and this helped to keep things steady. September opened with soft foreign ruling at £12 16s. 3d. to £12 17s. 6d., and in the absence of offers from America prices rallied to £13, but dropped again before the close of the month to £12 17s. 6d. Opening in October at this figure, the market remained steady thereat until a little buying caused values to advance about 10¢, spot metal being very scarce and eagerly sought after. Early in November this fact was again accentuated, and values improved to £13 11s. 3d., but on rather free offerings of arrival parcels receded to £13 8s. 9d., and after remaining steady at that price for a day or so prices began to slip back gradually, until at the opening of December, soft foreign was worth no more than £13 3s. 9d., and with the approach of the holiday season there was the usual slackness and the article was neglected; prices, however, kept fairly steady, final rates being £13 to £13 2s. 6d. for soft foreign, and £13 1s. 3d. to £13 3s. 9d. for English lead.

UNITED KINGDOM: IMPORTS AND EXPORTS OF LEAD. (IN TONS OF 2,240 LB.)

	1898.	1897.	1896.	1895.	1894.	1893.	1892.	1891.	1890.	1889.	1888.
Imports............	194,479	167,442	167,594	162,924	161,861	191,174	182,782	169,724	158,649	145,451	182,911
Exports	98,031	40,503	41,247	41,666	47,080	48,935	58,101	48,726	55,536	52,153	42,653

WHITE LEAD, RED LEAD, LITHARGE AND ORANGE MINERAL.

The statistics of production of these substances in the United States are given in the following tables, which represent the make reduced to a basis of dry white lead, red lead, etc., and do not include the oil contents of any marketed as "ground in oil," as often reported. The domestic production of white lead, etc., is obtained chiefly by the corrosion of pig lead, but a small amount is made at Joplin, Mo., and Cañon City, Colo., by direct volatilization of ores, these products being sulphates rather than basic carbonates.

PRODUCTION OF RED LEAD, WHITE LEAD, LITHARGE AND ORANGE MINERAL.

Year.	Red Lead.		White Lead.		Litharge.		Orange Mineral.	
	Short Tons.	Value.	Short Tons.	Value.	Short Tons.	Value.	Short Tons.	Value.
1897..........	7,798	$744,709	105,804	$9,522,360	8,591	$773,190	477	$76,320
1898..........	9,160	916,000	93,172	9,391,738	7,460	710,192	541	108,200

UNITED STATES: IMPORTS OF RED LEAD, WHITE LEAD, LITHARGE AND ORANGE MINERAL.

Year.	Red Lead.		White Lead.		Litharge.		Orange Mineral.	
	Pounds.	Value.	Pounds.	Value.	Pounds.	Value.	Pounds.	Value.
1892..........	812,703	$28,443	744,838	$40,032	56,737	$1,811	1,409,601	$64,133
1893..........	854,982	27,349	686,490	34,145	42,582	1,310	1,385,828	61,360
1894..........	947,873	29,064	796,480	40,939	38,595	1,064	1,386,464	58,614
1895..........	1,764,274	53,139	1,897,892	79,887	97,667	2,812	1,689,367	66,492
1896..........	1,543,262	47,450	1,183,538	52,409	51,050	1,615	1,359,651	51,027
1897..........	1,386,070	46,992	1,101,829	48,988	60,984	1,931	1,486,042	67,549
1898..........	682,449	25,780	506,739	24,334	56,417	2,021	795,116	37,745

(By L. A. Cole): "The business in white lead in the United States in 1898 was less in volume than in 1897; sales were less and production was less. Four principal reasons may be assigned for this falling off: (1) The buyers had increased their stocks on the advancing market in pig lead during the second half of 1897, and at the end of that year the stocks of white lead in distributors' hands were large for the season. (2) In the spring of 1898, the time of the year when white lead is especially in demand, the trade was interrupted by the Spanish-American war. (3) White lead has probably suffered to some extent from the inroads of substitutes in the trade previously held.

(4) By the time the war was over, or rather by the time the end of it was in sight, the spring painting season was over. The effect of the war was felt especially on the Atlantic Coast, within range of a 13-in. gun from deep water. This made it difficult to secure tenants for a seaside cottage anywhere between Bar Harbor and Atlantic City, and not only on the immediate seashore, but also for several miles back very little house painting was done. Going inland, the falling off in business was less noticeable, and in the Mississippi Valley the trade was about the same as in 1897. Along the coast business improved somewhat in the autumn, but undoubtedly a good deal of painting that would have been done naturally in the spring of 1898 was deferred until the spring of 1899. The price of white lead in oil remained nearly constant throughout the year, the total range of fluctuation not having exceeded 0·25c. per lb. At the end of the year it closed at 5·5c. per lb. This shows an advance over 1897, made necessary by increased cost of raw material. The year closed with bright prospects for the business in 1899. At the end of 1898 stocks of white lead in the hands of dealers were comparatively small."

Processes of Making White Lead.—The processes by which white lead is now made commercially may be classified as follows: (1) The Dutch process, in which the lead arranged in suitable beds or rooms is transformed by the action of acetic acid vapor, carbon dioxide and water, successively, into acetate and basic carbonate. (2) The so-called quick processes in which the lead, either as metal or as oxide, is dissolved in acetic acid and then precipitated as basic carbonate by the passage of carbon dioxide gas through the solution. The French, or Clichy, process and the Carter and Matthews processes are types of this class. (3) Electrolytic processes, depending upon the decomposition of a lead salt by electrolysis. (4) Volatilization processes, such as those employed by the Picher Lead Co., of Joplin, Mo., and the American Zinc Lead Co., of Canyon City, Colo., the products of which are essentially sulphates.

Of the precipitation processes the Carter is the only one which is in use in the United States. In England the Matthews process has been making decided advances of late and is said to be turning out a product of very superior quality at a cost of $57·96 per 2,240 lb., or less than the cost of manufacture by the Dutch process. The chief novelty of the Matthews process consists in the use of 7·5% of glycerin in the acetic acid liquor for dissolving the litharge. This has the affect of retarding precipitation by carbon dioxide and yielding a white lead of more uniform basicity. The precipitated lead is filter-pressed and dried carefully for three days in copper pans, after which it is very soft and white. Acetic acid and glycerin are recovered from the waste liquor, with insignificant loss, while there is said to be no loss of lead in the process.

Technically the production of white lead in the United States continues the same as heretofore. Fully 90% of the total is made by the old Dutch process, and the entire output of the National Lead Co. is produced in this manner. The experience of this company has been that white lead made by the old process possesses qualities of excellence which have not been found possible to obtain with the new quick processes. With respect to electrolytic processes, they do not yet appear to have passed beyond the domain of laboratory experiment. While there have been no improvements in the chemistry of the old Dutch process, the works employing it have made numerous mechanical improvements, all tending to reduce the cost of production and improve the quality of the product, which means to increase its durability as a pigment, as well as purity of color.

At a modern American white lead works (those of the Sterling White Lead Co. at New Kensington, Pa.) employing the old Dutch process, the pig lead is cast into "buckles" 6 in. in diameter, which are put into pots containing about one pint of vinegar.* These are placed in the corroding house in 10 tiers, 600 to 1,000 pots in each, the pots being covered with boards and layers of tanbark placed between each tier. The rooms are kept closed from three to four months, during which time the heat and carbonic acid gas generated by the fermentation of the tan, together with the acid vapors, accomplish the corrosion of the lead. The corroding house is 78 × 450 ft., containing 34 rooms ("beds"), each of a capacity of 60 tons of pig lead per 100 days. Instead of the usual tedious method of charging the beds by hand, an electric crane is used. The introduction of the electric crane led to improvements in the separator, where instead of the ordinary charge of 200 to 300 lb., a charge of 2,500 lb. is fed automatically into the revolving screens. The separator has a capacity of 50 tons per diem, and there are two double-run water mills with a combined daily capacity of 30 tons. These are run in connection with 14 cedar settling tanks holding about 100 bbl. each. The drying rooms on the top floor are furnished with 17 pans with a capacity of 150 tons a week. These pans are of the filter type, which is said to be in use nowhere else in the world. This consists of an inclined bed with a false bottom of brass wire cloth covered with filtering fabric, beneath which there is a series of steam coils. Instead, therefore, of having to provide sufficient heat to evaporate all the moisture, only enough is consumed to evaporate that which cannot be drawn off by filtration. This dryer has the advantage also of making it feasible to give the pulp an extra washing. The works are equipped with a complete bark-extracting plant for the supply of tan for corrosion. Chestnut and oak barks are used. The plant is similar to those installed at first-class tanneries, having a vacuum pan for concentrating the extract. The bark is exhausted only to the most effective point, yielding a spent tan of superior quality for use in the corroding beds. The concentrated extract finds a ready sale to leather tanners for mixing with hemlock-bark extract.

Electrolytic Processes for the Manufacture of White Lead.—Sherard Cowper-Coles summarized the present status of these processes as follows:† In one process the lead is cast into thin sheets about 20 × 15 in., weighing between 2 and 3 lb. each. These sheets are then corrugated and packed in crates with plates of coke or carbon which form the negative electrodes. Strips of tin make the connection between the lead or electro-positive plate and the carbon electrode, the theory being that when subjected to warm vapors of acetic acid, carbonic acid and atmospheric air, electrical action is set up between the two, which hastens the corrosion and reduces the time required to convert the plates

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* *Drugs, Oils and Paints*, February, 1898.

† *Chemical Trade Journal*, April 30, 1898.

into lead carbonate from the 10 to 15 weeks, as required by the Dutch process, to five weeks. The corroded plates are then subjected to the same treatment as in the Dutch process.

Another electrolytic process consists in electrolyzing a 10% solution of sodium nitrate in a wooden cell, provided with a porous diaphragm, the anode being made of pig lead and the cathode of copper. In the former compartment, the lead dissolves as nitrate, while round the cathode caustic soda is produced. The two solutions being drawn off and mixed, alkaline nitrate is regenerated and the lead precipitated as hydrate. The hydrate is finally treated with sodium carbonate or bicarbonate to convert it into white lead. The caustic soda formed as a by-product in the latter operation has only to be re-carbonated to be available again.

A somewhat similar process, patented by Browne, is carried out by placing metallic lead in a suitable vessel, in electric connection with the positive pole, in the presence of a solution of an acetate or a nitrate of an alkali base. A soluble salt of lead is formed on the anode, and a solution of caustic alkali at the cathode. In consequence of the flow of the electrolyte against the cathode, these two solutions intermix, with the result that lead hydrate is precipitated, the original alkaline salt being regenerated. The solution containing the lead hydroxide in suspension is then run into a settling-tank, from which the clear solution is subsequently drawn off, to be used again as the electrolyte, while the lead hydrate is finally exposed to the air or to an atmosphere of carbonic acid gas to convert it into hydrated carbonate of lead. In reality electrolysis plays but little part in this process, as the mixture of lead carbonate and hydrate of which white lead consists is obtained purely by chemical means.

Another process, patented in the United States, consists in passing an electric current from an anode of metallic lead through an alkaline electrolyte (consisting of a solution of a salt of ammonia in combination with any acid, such as acetic, which will produce a soluble lead salt; and of a bicarbonate of an alkali) to a cathode of lead, carbon or other suitable material, thereby causing a decomposition of the salts in the electrolyte, and the formation of a soluble compound of lead, which is transformed into hydrated carbonate of lead by the simultaneous generation of free carbonic acid gas at the anode by the presence of caustic alkali (ammonia) generated at the cathode. A current of carbonic acid gas is passed through the electrolyte to regenerate the spent alkaline carbonate until the anode is completely dissolved. Both the acetate and bicarbonate of soda are continuously regenerated, the process consuming only lead, carbonic acid gas and water.

A new English process consists in decomposing acetate of ammonia by electrolysis, lead anodes being used, which are separated by porous diaphragms (made preferably of stout Willesden paper) which are insulated from the bottom to prevent irregular electrolytic action in the electrolyzer. A lead solution is produced, and caustic alkali, when on mixing the two solutions white lead is precipitated. Reference should be made to the paper on the metallurgy of zinc, elsewhere in this volume, for an account of the new Cowper-Coles process of making white lead.

RECENT IMPROVEMENTS IN LEAD SMELTING.

BY H. O. HOFMAN.

B. D. C. G., Berichte der Deutschen Chemischen Gesellschaft; *B. I. M.*, Bulletin de l'Industrie Minérale; *B. S. E. I. N.*, Bulletin de la Société d'Encouragement pour l'Industrie Nationale; *Can. Mg. Rev.*, Canadian Mining Review; *Ch. Z.*, Chemiker Zeitung; *Elect.*, The Electrician, London; *E. W.*, Electrical World; *E. & M. J.*, Engineering and Mining Journal; *G. A.*, Glück Auf; *Jl. Am. Ch. Soc.*, Journal of the American Chemical Society; *Jl. Fed., Can. M. I.*, Journal of the Federated Canadian Mining Institute; *Jl. Soc. Arts*, Journal of the Society of Arts; *Jl. Soc. Chem. Ind.*, Journal of the Society of Chemical Industry; *Met.*, The Metallographist; *M. I.*, THE MINERAL INDUSTRY; *M. & S. P.*, Mining and Scientific Press; *O. Z.*, Oesterreichische Zeitung für Berg- und Hüttenwesen; *S. M. Q.*, Columbia School of Mines Quarterly; *Th. Z.*, Thonindustrie Zeitung; *Trans. Am. I. M. E.*, Transactions of the American Institute of Mining Engineers; *Trans. Austral. I. M. E.*, Transactions of the Australasian Institute of Mining Engineers; *Trans. I. M. & M.*, Transactions of the Institution of Mining and Metallurgy; *Z. E. C.*, Zeitschrift für Elektrochemie.

INTRODUCTORY.

Analyses of Desilverized Lead.—Average samples of all the lead desilverized by the steam-Pattinson process at Pribram, Bohemia, during the years 1897 and 1898 gave upon analysis the following:[*]

Year.	Ag.	Cu.	Bi.	Sb.	Fe.	Zn.	Ni.	Pb.
1897..........	0·0016	0·0024	0·0028	0·0067	0·0015	Trace.	0·0014	99·9836
1898..........	0·0017	0·0030	0·0026	0·0045	0·0013	0·0009	0·0013	99·9848

Lead-Antimony Alloys.—Charpy,[†] in a microscopical study of metallic alloys, discusses briefly the alloys of lead and antimony, giving illustrations of alloys containing 70, 40 and 20% antimony. They show crystals of antimony surrounded by increasing amounts of eutectic alloy. Lead-antimony alloys (belonging to the group which form neither chemical compounds nor isomorphous mixtures) show a curve of fusibility composed of two branches meeting in a sharp angle at a point which corresponds to an alloy of 13% antimony—the eutectic alloy. Stead[‡] formed its melting point at 247° C. and its sp. gr. at 10·48.

Lead-Tin Alloy.—The eutectic alloy of lead and tin[§] belonging to the same class as the lead-antimony alloys, contains 35% lead and 65% tin, and solidifies at 180° C.

Lead Ores.—Ohly[‖] records the presence of native silver in galena from the Seaton mine, near Idaho Springs, Colo.

Laurium Slags.—According to Lacroix[¶] the slags made by the ancient Greeks at Laurium, which are now being resmelted, have undergone remarkable changes, especially those which had been subjected to the influence of sea water. The lead of the slags has been converted into oxide, carbonate, chloride, sulphate, etc., forming new minerals, viz.: Laurionite, $PbCl_2$, 2 PbHO; pentfieldite, 2 $PbCl_2$, PbO; fiedlerite; phosgenite, $PbCO_6$, $PbCl_2$;

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[*] *O. Z.*, 1897, p. 701; 1898, p. 189. [†] *B. S. E. I. N.*, March, 1897 (through *Met.*, I., p. 87).

[‡] *Jl. Soc. Chem. Ind.*, March and June, 1897, through *Met.*, I., p. 192.

[§] Charpy, *Op. cit.*, p. 96; Roberts-Austin, Canton Lecture, delivered March 15, 1897, published in *Jl. Soc. Arts*, Oct. 22, 1897 (through *Met.*, I., p. 144).

[‖] *Ch. Z.*, 1898, Vol. XXII., p. 60.

[¶] *G. A.*, 1897, Vol. XXXIII., p. 31; *Ch. Z.*, Repertorium, 1897, Vol. XXI., p. 31.

matlochite, $PbCl_2$, PbO; cerussite, $PbCO_3$; hydrocerussite, 3PbO, $2CO_2$,H_2O; anglesite, $PbSO_4$.

Sampling Base Bullion.—Moldenhauer,[*] assayer at Garrucha, Spain, describes his method of melting and molding the chips obtained by sampling base bullion by one of the usual methods. The chips from a lot weighing from 11 to 13 lb. are melted down in a hot plumbago crucible charged with from 2 to 2½ in. of charcoal and heated in order to reduce all oxide and to redissolve any dross that may have risen to the surface. The lead is poured into upright molds, giving sheets $3\times6\times\frac{5}{32}$ in., weighing about 500 g. In casting, it is important either to leave a 1-in. space at the top of the mold or to cut off about 1 in. from the top of the lead, as the composition at the very top differs from that below. Of course the mold must be kept cool in order to prevent any unmixing of silver and lead.

Roberts[†] discusses some experiments that he made in sampling base bullion when connected with the Proprietary mines, Broken Hill, New South Wales. The conclusions arrived at are that dip samples taken when the bullion is flowing from a furnace or is being ladled from a kettle and saw samples are the only accurate means of obtaining a true average; chip samples taken from top and bottom alone are condemned, because it is necessary to have them also from the sides, which, cooling the quickest, are the richest. He found that melting down samples quickly in a red-hot crucible was most advantageous; his experiments showed him also that the loss by volatilization in melting down the chips was *nil;* further, that in cupelling the loss by absorption with 400-oz. bullion was only 0·06 oz. per ton.

Assay Furnaces.—John Taylor & Co.,[‡] San Francisco, have brought out a portable sheet-iron muffle-assay furnace, 12 in. square, lined with 2-in. fire brick. It weighs 250 lb., takes a $6\times12\times4$ in. muffle and holds six No. 9 Battersea crucibles. By removing the muffle it can be used as a melting furnace, holding a No. 16 black lead crucible. The furnace is made also in four sections fastened together by bolts.

Koenig[§] describes a new furnace and method of making gold and silver assays. The furnace is a brick chamber, $5\times17\frac{1}{2}$ in. inside measurement, with horizontal bottom tile and inclined hinged roof tile. At the back are openings for gas or gasoline flames which supply the heat; at the front blowpipes to furnish the air required for scorifying the ore or cupelling the lead button placed in a cupel which rests on a narrow longitudinal fire-clay tile. With crucible assays this shelf is removed. The furnace holds six Battersea F crucibles or six 2¾-in. scorifiers or six cupels. Koenig finds that the results he obtains with this furnace are as good, if not better, than those from the ordinary muffle furnace; the time occupied in cupelling is from four to six times less, and the assayer is hardly exposed to heat at all. In this new desilverization, which the author compares with bessemerizing rather than with cupelling, the lead oxide is vaporized and not absorbed by the bone ash, the cupel remaining comparatively cool.

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[*] *C. Z.*, 1898, No. 6, p. 40.

[†] *Trans. Am. I. M. E.*, October, 1898.

[‡] *E. & M. J.*, May 14, 1898.

[§] *Trans. Am. I. M. E.*, February, 1898.

Muffle Lute.—Marsh* recommends as a cement for muffles a mixture of from 1 to 2 parts litharge and 10 parts bone ash, the ingredients to be mixed dry, then moistened with water and worked to a paste. He claims that this lute will stand fire and be tight, hard and strong.

Assay of Telluride Ores.—Fulton† gives the results of some of his work on the assay of two silicious telluride gold ores, sample No. 1 containing a small amount of pyrite and assaying nearly 6 oz. gold per ton, and No. 2 containing considerable pyrite and assaying over 450 oz. In the crucible assay of sample No. 1 the charge used was: Ore 1 A. T., litharge 1 A. T., silica ½ A. T., soda 1 A. T., borax glass 10 g., argol 2 g., with the usual salt cover. Four iron nails were added to assist in desulphurizing, and 20 g. silver foil to help collect the gold. With ore No. 2 the charge used was: Ore ⅕ A. T., litharge 2 A.T., soda 1 A. T., silica ½ A. T., argol 1 g.; no nails were added, but enough silver to avoid subsequent inquartation. When the buttons were brittle they were scorified before cupelling. In all charges the loss by slagging was determined; the loss by absorption, having been found to be small with No. 1, was not determined with No. 2. The subjoined table shows the data obtained from using an increasing amount of litharge:

Ore.		Litharge. A. T.	Argol. Grams.	Lead Button.		Gold Recovered.			
No.	Amount A. T.			Character.	Weight. Grams.	From Lead Button. Mg.	From Crucible Slag. Mg.	From Cupel. Mg.	Total. Mg.
1	1	1	2	Malleable.	20	5·60	0·03	0·09	5·72
1	1	2	2	Brittle.	37	5·52	0·16	0·02	5·70
1	1	2	2	Brittle.	47	5·60	0·03	0·01	5·64
1	1	4	2	Brittle.	32	5·67	0·05	0·01	5·73
1	1	4	2	Brittle.	30	5·65	0·08	0·02	5·75
1	1	6	2	Malleable.	48	5·72	0·06	0·01	5·79
1	1	6	2	Malleable.	45	5·78	0·06	0·01	5·85
2	⅕	2	1	Malleable.	20	89·10	0·22	....	90·20 (?)
2	⅕	4	1	Malleable.	26	90·50	0·08	....	90·90 (?)
2	⅕	4	1	Malleable.	48	90·26	0·10	....	90·36 (?)
2	⅕	6	1	Brittle.	28	90·30	0·09	....	90·75 (?)

The results show that the gold recovered increases with the litharge, but that above 6 A. T. there was no increase; that the loss by slagging is large and the cupel absorption small. Experiments undertaken to show the loss of gold in scorifying brittle buttons proved that the slag carried away large amounts of gold, and that the loss by volatilization was considerable. Thus with ore No. 2 the total yield of 99·55 mg. for ⅕ A. T. was reduced to 89·13 mg. The cyanide assay (ore 1 A.T., potassium cyanide 2 A. T., litharge 1 A. T., salt cover) gave with ore No. 1 5·40 mg. gold per A.T. instead of 5·67 as by the charges used first. Scorification assays gave uniformly lower results than crucible assays. While the loss in gold in the scorifier by slagging was large, the loss by volatilization was greater.

Assay of Iron in Lead Ores.—In regard to the restrictions made by Jones‡ on the titration of iron (in lead ores) with potassium permanganate in a hydrochloric acid solution, it seems in place to call attention to the Zimmermann-Reinhardt method,§ which gives accurate results even to the beginner. The

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* *E. & M. J.*, Jan. 8, 1898. † *S. M. Q.*, XIX., p. 419. ‡ *M. I.*, VI., p. 446.

§ *B. D. C. G.*, 1884, p. 779; *Ch. Z.*, 1889, XIII., p. 323; 1897, XXI., p. 894; *Jl. Am. Ch. Soc.*, 1895, XVII., p. 405.

mode of operating is as follows: Place 0·5 g. ore (containing 65–75% Fe) in a lipless beaker, add 2·5 c.c. stannous chloride (1 lb. salt in 1 lb. HCl. of sp. gr. 1·2 to which water has been added to make up 2 liters), then 10 c.c. hydrochloric acid (sp. gr. 1·1), cover and heat on iron plate to complete solution; add, while still hot, with stirring, from a burette, stannous chloride until the greenish yellow color has disappeared (with many samples slightly reoxidize the iron by adding a few drops of permanganate, keep warm and reduce finally by a drop or two of stannous chloride); wash down the sides of the beaker with water; add 5 c.c. mercuric chloride (hot saturated solution in water from which the salt has crystallized out in cooling and been filtered off), which throws down white silky mercurous chloride. Wash contents of beaker into a 500 c.c. beaker containing 400 c.c. water and 6 to 8 c.c. manganous solution (160 g. manganous sulphate dissolved in water, diluted with water to 1,750 c.c., then 330 c.c. phosphoric acid syrup of sp. gr. 1·7, and 320 c.c. sulphuric acid of sp. gr. 1·84 added), when the solution is ready to be titrated with potassium permanganate (250 g. in a carboy of about 44 liters), when with 0·5 g. ore 1 c.c. will equal about 2%.

Proof Gold and Silver.—Pack[*] describes an improved method of preparing proof gold and proof silver. For proof gold: Dissolve in nitrohydrochloric acid the gold obtained from assay buttons; allow it to stand several days and decant. Dilute the solution remaining with the silver chloride; pass through a double filter; evaporate the whole to crystallization and redissolve in water. Dilute the gold solution, and pour it into a beaker containing a piece of pure aluminum, when gold will be at once precipitated; filter, wash, dry and melt. For proof silver: Dissolve fine silver in nitric acid; dilute with much water; allow it to stand for several days to settle out any gold; decant and filter. Precipitate the silver with hydrochloric acid, stirring vigorously, wash, decompose the silver chloride with aluminum, adding some hydrochloric acid; wash the metallic silver, adding some hydrochloric acid and warm to dissolve any aluminum remaining.

Whitehead and Ulke[†] describe in detail the methods of assaying gold and silver bullion used at the United States mints, to which article the reader must be referred, as an abstract covering the necessary details would prove too long.

SMELTING OF LEAD ORES.

In the Reverberatory Furnace.—Huntington and Heberlein have patented[‡] a process for the complete oxidation of lead sulphide. They mix galena with lime, heat with access of air to 700° C., then cool to 500° C., when lead sulphide is rapidly oxidized to lead oxide and sulphur dioxide, the heat generated being sufficient to fuse the lead oxide.

In Lead Bath.—Dyson[§] describes a method of treating silver-lead ores in the reverberatory furnace on a bath of lead, as practiced at a European smelting plant. The process is a combination of soaking and roasting-reacting,

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[*] *E. & M. J.*, July 9, 1898.

[†] *Ibid.*, Feb. 12, 26, 1898.

[‡] United States Patent No. 600,347, March 8, 1898.

[§] *Trans. Austral. I. M. E.*, 1898, V., p. 71.

and aims to quickly extract the major portion of the silver and lead, the rest being recovered by treatment in the blast furnace. The ore is a mixture of sulphide and carbonate of lead. The former assays 37% lead and about 300 oz. silver per ton, and contains small quantities of pyrite; the latter, 68% lead and 14 oz. silver, and is replaced by litharge when there is a shortage of ore. The reverberatory resembles the one used for softening base bullion; *i.e.*, its oblong hearth is built with an iron pan resting on iron crossrails supported by longitudinal brick pillars. Between the dish-shaped hearth (which holds the lead bath) and the firebridge is a liquation hearth, 3 ft. 6 in. long with a slope of 25°, which serves to drain off the lead from the drosses and slags formed in the operation. The process is carried on as follows: The furnace is brought to a dark red heat, requiring 15 hours and 1,600 lb. coal, then 5 tons low-grade base bullion assaying about 30 oz. silver per ton are charged, melted down, drossed and the dross placed on the liquating hearth. Then 500 to 600 lb. sulphide ore are dropped from the hopper in the roof on to the lead, over which it spreads rapidly; the fire is urged until the ore begins to cake, then 400 lb. carbonate ore or litharge are spread over it to react upon the sulphide. The dross on the liquating hearth and the slag-like mixture on the lead bath are now worked alternately three times for periods each of 10 minutes; the slag finally receives from four to five shovels of coal dust to reduce the excess of oxidized lead, when the dross on the liquating hearth is withdrawn and the slag-like mixture taken from the bath of lead and put in its place and a fresh ore charge dropped from the hopper on to the bath of lead. A charge remains two hours on the lead bath and two hours on the liquation hearth. Twenty charges bring the silver contents of the 30-oz. bullion to 300 oz., when it is tapped and goes to the cupelling furnace. A crew of four men serves the furnace in a 12-hour shift. The consumption of coal in 24 hours is 2,800 lb. The following is a record of 112 charges treated, the lead in the furnace being replaced four times:

Material.	Charged.			Drawn.		
	Weight. Lb.	Contents.		Weight. Lb.	Contents.	
		Lead. Lb.	Silver. Oz.		Lead. Lb.	Silver. Oz.
Sulphide ore	60,450	20,771	11,268	...	...	...
Carbonate ore	37,478	25,485	262	...	...	...
Litharge	10,144	8,619	135	...	...	...
Blast furnace lead	...	51,697	1,682	...	...	...
Rich lead	...	...	...	82,293	82,293	12,047
Slag	...	...	...	68,078	20,158	554
Total	108,069	106,572	13,347	...	102,451	12,601

The table shows that of the 54,875 lb. lead in the ore charged, 55·75% is obtained in the form of bullion; 36·75% remains in the slag, and 7·5% is unaccounted for. As to the yield of silver, of the 11,665 oz. charged, 88·84% is recovered in the base bullion; 4·75% remains in the slag; 6·41% is unaccounted for. A portion of the lead and silver not accounted for is recovered in the fluedust. The slags and drosses are smelted in the blast furnace.

Combined Reverberatory and Blast Furnace Treatment.—Collins* proposes to solve the problem of how to treat the silver-bearing mixed lead and zinc sulphide concentrates of the Broken Hill Proprietary mines by combining reverberatory with blast furnace smelting in such a way as to roast part of the concentrates in a mechanical or a hand reverberatory roasting furnace, with a consumption of 12% fuel, and charging them hot with raw concentrates into a reverberatory smelting furnace (24×12 ft. inside), working continuously, with a consumption of about 18% fuel. In this furnace nearly half of the lead and most of the silver would be recovered and the resulting gray slag would be smelted in the blast furnace with about 12% coke.

In the Ore Hearth.—At Kreuth,† in the Bleiberg district of Carinthia, the Belgian reverberatory furnace, which was introduced a few years ago in the expectation of its doing better work than the American ore hearth, has proved unsatisfactory and has had to give way again to the former ore hearth. At Windisch and Kappel in the same district the old Carinthian reverberatory furnace still holds its own, being preferred on account of local conditions to the American ore hearth.

Blast Furnace Smelting at Laurium.—Georgiades‡ gives a review of the lead smelting operations carried on at Laurium (Greece) since 1865, at which time the dumps of the ancients were beginning to attract the attention of modern capital. The old dumps (*eklovades*) are of three kinds—refuse from the mines, tailings from the dressing of ores, and slags. The slags form heaps on land or are submerged in the sea, whence, after being loosened by blasting with dynamite, they are removed by dredging. Their composition is shown in the following table:

Material.	SiO_2.	Fe.	Mn.	CaO.	MgO.	Al_2O_3.	Zn.	Pb.	Resulting Base Bulli'n Oz. per Ton. Ag.
	%	%	%	%	%	%	%	%	
Land slags	31·8	9·00	....	17·6	....	5·1	3·56	9·5	146
Land slags	27·6	13·90	....	14·2	....	4·0	3·45	8·6	117
Land slags	28·4	8·40	....	20·3	....	5·3	3·00	10·0	133
Sea slags	34·9	15·37	2·5	12·2	0·25	....	3·79	11·4	132
Sea slags	32·1	13·90	2·0	13·7	2·70	....	4·31	12·4	116

The paper discusses the smelting operations carried on in the periods 1865-1876, 1876 to the beginning of the nineties, and from then on. The furnace used to-day is 5 ft. 3 in. in diameter at tuyères and 14 ft. 7½ in. high from tuyères to feed floor; it has an internal crucible, and seven tuyères, the water-cooled nozzles of which reach 4 in. into the furnace and are placed in such a way that the jets of blast entering through them shall not meet at the center. The furnace has 7 ft. 10½ in. bosh 11 ft. above the center of the tuyères. The throat is closed by bell and hopper. The furnace puts through in 24 hours 56 tons of charge consisting of 50% slags, produced from 1865 on, containing 0·75% lead and the remainder ores, briquettes and argillaceous slimes. The

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* *Trans. Austral. I. M. E.*, through *Can. Mg. Rev.*, Oct. 31, 1898.

† *O. Z.*, 1898, p. 360.

‡ *B. I. M.*, 1897, Vol. XI., p. 523.

briquettes contain SiO_2 15-16%, Fe 13·71%, CaO 18·40%, MgO 4·68%, Al_2O_3 7·60%, Zn 3·60%, Pb 7%, and are rich enough in silver to produce 695 oz. base bullion. The argillaceous slimes contain SiO_2 27·20%, Fe 11·15%, CaO 7%, Al_2O_3 7·50%, Zn 5·04%, Pb 4·65% and silver to produce 784 oz. base bullion. The ore charge averages 5·5 to 6% lead and the bullion produced assays 730 oz. silver per ton.

Smelting at Broken Hill.—Greenway* gives a review of the developments made at Broken Hill, New South Wales. He shows a longitudinal elevation and a horizontal section at tuyère level of the blast furnace used and a plan of the smelting plant with its eight 60×112 in. furnaces. The oxide ore is smelted as mined. The sulphide ore averages Pb 15-30%, Zn 10-25%, Ag 5-25 oz. Concentration in the dressing works gives five products as follows:

Product.	Pb, %	Zn, %	Ag, oz.
Concentrates, first-class.......	60-80	8-10	10-35
Concentrates, second-class....	40-60	10-15	8-25
Zinc middlings..................	8-12	10-35	3-10
Tailings.........................	4-10	5-20	2-10
Fine slimes.....................	20-30	15-25	15-25

The concentrates are smelted on the coast or shipped to Europe; middlings with over 30% zinc are shipped to Europe; the rest are stacked for later treatment by a process still to be devised. The poorest tailings are sent underground as mine filling. The smelting is discussed in Mr. Blakemore's† paper on "Some Metallurgical Methods in Australia," read before the Australasian Scientific Association. The leading points of it are as follows: At the British mine, Howell substituted low-carbon steel jackets for the brick shaft erected by Schlapp. In smelting sulphide ores, wall accretions form, beginning from 10 to 12 in. below the top of the charge, where they adhere to the brickwork and filter into it. With a water-jacket shaft only the face of the crust which is exposed to heat is hard, while back of it is loose material (the same is often the case with brick shafts); the crusts adhere only slightly to a water-jacket shaft and are readily (in six hours) removed, leaving a smooth surface, while it is difficult to remove them from brick shafts (in from 12 to 18 hours), and the walls remain rough and uneven. With a brick shaft the space directly above the jackets is easily burned out; not so when water jackets are used. The claim that a water-jacket shaft requires more fuel than a brick shaft is not so far borne out by experience. The jackets have a 6-in. water space; they are placed in four tiers, three being 9 ft. 9 in. long, and one 8 ft. having a recess for the downtake. They all receive their cooling water near the bottom and are connected by pipes so as to insure a uniform circulation. The jackets, bound by hoops, are supported by lugs resting on 8-in. I-beams carried by the four columns. The smelting zone of the furnace has 20 closed cast-iron jackets, each 20 in. wide and 57 in. high. Water-jacket tuyère nozzles reach 8 in. into the sides of the furnace. The front and back jackets have 4-in. openings for ordinary tuyère pipes. The water fed at the top of a jacket is

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* *E. & M. J.*, Aug. 27, 1898.

† *M. & S. P.*, May 21, 28; June 11, 18, 25, 1898.

conducted to the bottom by a passage cast inside the jacket and escapes through an S-shaped pipe discharging 9 in. above the top. The furnace, 50 × 132 in. at the tuyère level, is 34 in. between tuyères and 17 ft. high from bottom of crucible to feed floor. Blakemore states that Mathewson's forehearth is satisfactory only when small quantities of matte are being formed, but does not state what he means by "small."[*] He describes the blowing-in of a blast furnace, giving the usual method of warming without and with blast for from two to four days, melting down the lead to fill the crucible (15 tons) for which he advises taking from 8 to 12 hours so that the lead may be well heated and remain molten until the furnace, after having been filled and started, will make fresh red-hot lead. After filling the crucible, removing ashes, charging kindling and dry cord wood, and putting in of the breast, he gives a bed of 1,600 lb. coke, from bags into which it has been weighed, to be followed by blowing-in charges, 50 of which have been weighed out, spread on the floor and kept separate by boards. The following table shows the composition of the blowing-in charges:

No. of Charges.	Slag.	Basic Manganiferous Iron Ore.	Limestone.	Carbonate Lead Ore.	Silver-Bearing Iron Ore.	Total.	Coke.
	Lb.	Lb.	Lb.	Lb.	Lb.	Lb.	Lb.
25	750	75	75	....	....	900	125
25	500	75	75	275	....	925	125
50	250	75	200	325	150	950	125

Full ore charge, 1,000 lb.

After the 15th, the 30th and the 55th blowing-in charges 10 bars of lead are given, and for the subsequent 50 charges one bar every charge.

The blowing-in charges fill the furnace to within 6 in. from the top. The blast is turned on at first to give a pressure of from 3 to 4 oz.; in from 1¼ to 1½ hours the first slag can be tapped, which is made purposely low in silica (30%). The following table gives examples of blast furnace charges:

	Lead Carbonate Ore.	Fluedust.	Kaolin.	Silicious Iron Ore.	Crude Sulphide Ore.	Iron Ore.	Iron Stone.	Limestone.	Slag.	Coke.
1	150	....	300	150	25	....	75	300	....	125
2	150	....	175	225	25	....	125	300	....	125
3	150	....	275	175	25	....	75	300	....	125
4	150	....	250	200	25	25	50	300	....	125
5	150	....	225	200	25	25	75	300	....	125
6	125	25	300	150	25	125	....	250	....	125
7	150	....	250	175	25	125	....	275	....	125
8	175	....	250	100	25	175	....	275	....	125
9	....	....	125	325	Conc'ntr'te	....	150	225	125	125
10	....	....	125	325	....	....	250	175	125	125
11	....	....	100	....	425	11	....	225	125	125
12	....	....	125	325	....	....	150	225	125	125
13	....	....	125	....	325	....	250	175	125	125
14	....	....	125	325	....	....	250	175	125	125
15	....	....	125	325	....	....	150	275	125	125
16	....	....	100	....	425	....	100	225	125	125

The iron ore contains SiO_2 6-14%, Fe 50-60%, and the limestone SiO_2 6-11%, $MgCO_3$ 1-5%, Fe_2O_3 1·5%, and the remainder $CaCO_3$. Examples of corresponding slag analyses are given in the following table:

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[*] See Hofman's *Metallurgy of Lead*," 5th Edition, p. 237.

No.	Pb.	Ag, oz.	SiO_2.	Fe.	Mn.	CaO.	Zn.	Dip Sample Base Bullion, Ag, oz.
	%	%	%	%	%	%	%	
1	2·25	4·2	42·0	12·7	12·8	17.1	....	586
2	4·00	1·4	36·8	10·2	14·0	15·2	....	340
3	2·50	3·4	45·2	9·4	7·0	....	....	520
4	2·75	1·4	46·8	10·2	6·2	....	....	436
5	2·25	2·7	39·6	14·0	8·0	14·7	....	484
6	1·50	1·6	40·8	9·6	10·0	....	....	230
7	4·50	2·6	38·0	10·0	11·0	....	....	292
8	4·25	1·8	39·4	10·5	10·8	....	....	277
9	1·50	0·6	34·5	15·2	7·8	17·4	5·0	68
10	3·50	1·0	28·5	19·5	7·0	18·4	5·8	86
11	3·25	1·2	34·8	15·4	7·8	....	5·2	74
12	1·75	0·9	34·0	14·0	7·9	18·8	5·2	80
13	3·00	0·6	31·6	20·8	7·0	12·4	6·6	134
14	6·50	1·4	29·6	23·4	6·0	9·5	5·6	92
15	1·50	0·9	32·5	15·0	8·2	19·8	6·3	78
16	1·25	0·4	34·2	16·4	7·1	....	5·4	116

Blakemore found that alumina acted as a base with silica up to 36%, but when silica rose higher, the slags became very thick and viscous.

FIRST MATTE FROM CHARGES.

No.	9	10	11	12	13	14	15	16
Lead, %..........	36·8	25·5	....	42·0	29·2	58·4	35·4	23·7
Copper, %........	4·2	4·4	....	4·2	6·2	3·2	4·0	4·4
Silver, oz........	42·0	20·0	....	41·0	52·0	35·8	33·0	31·0

SAMPLE ASSAYS OF OTHER MATTES.

Lead, %................	39·7	33·7	31·3	32·8	34·4	43·2
Copper, %............	29·6	25·3	27·3	29·2	28·4	22·0
Silver, oz.............	195·0	244·0	138·0	168·0	172·0	182·0

THIRD MATTE ASSAYS.

Lead, %............................	29·8	32·9	37·9	37·1
Copper, %.............	40·0	42·2	35·2	32·7
Silver, oz.......	326·0	361·0	179·0	261·0

The largest number of 1,000-lb. charges smelted in the furnace was 166, the ore being a carbonate with 35·45% lead. The cost of smelting is given at 25s. a long ton. This figure includes everything from the receiving of the ore to the placing of base bullion on the cars ready for shipment. The loss of silver and lead at the Broken Hill furnaces was never carefully ascertained, as all data have been based on grab samples of the ores. The loss by slagging was great, as the slags ran high in lead (4%) and as much slag had to be made on account of the silicious character of the ore. The fluedust recovered amounted to 0·8–1·5% of the weight of the ore charge. Fluedust assays showed: Pb, 25, 37, 23, 30, 30%; Zn, 11, 13, 10·8, 10·7, 10%; Ag, 20, 24, 30, 21, 21 oz.

Smelting Progress in Utah—Terhune* discussed before the Salt Lake session of the International Mining Congress the "Recent Advances in Silver-Lead Smelting." He showed what progress has been made in Utah in the payment for values contained in the ore by the fact that in 1897 62·6% of the New York value of the metals (lead, silver, gold) was paid the miner, while in 1886 he

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* *M. & S. P.*, Aug. 6, 1898.

received but 58%. This was made possible to a considerable extent by a falling off of 37·30% of the cost of coke, iron ore and limestone during the last 18 years. Steel buildings and large plants have replaced the small wooden structures with one or two furnaces. The largest furnace in Utah is 44 × 144 in. at the tuyère level, and has an active smelting height of 20 ft. One Western plant is putting in cylinder blowers, though hitherto all works have used rotary pressure blowers only. The separating of matte from slag and the handling of the large amounts of slag have called for new devices. Matte is separated in large oblong settling pots, the waste overflow slag is brought to the dump in large bowls mounted on a railroad truck and dumped hot, or it is granulated and the granules are brought by a stream of water to an elevator which delivers them on a tram or on railroad cars. The furnace floor, neglected for many years, has received the well-merited attention of being paved, so that the handling of pots has become relatively easy and spilt rich products can be saved instead of being partly lost by becoming buried in adobe floors. Care is being also bestowed on furnishing the furnace buildings with fresh air. The blast furnace fluedust, which formerly was only imperfectly recovered, is completely saved by filtering the gases through muslin or woollen cloth, and briquetted by machinery, using lime as a bond, at the rate of 60 to 70 tons per day. With increasing depth of mines sulphide ores have become more prominent, requiring enlarged plants for roasting. Salt Lake valley has shown great preference for the Brückner cylinder, which successfully roasts ores with 32% lead or ores containing 30% zinc and 10% lead. The furnace in common use is 26½ ft. long, 8½ ft. in diameter and makes one revolution in 40 minutes; the weight of the charge varies according to the percentage of lead from 17 to 72 tons, and takes from 36 to 48 hours. The Ropp straight-line furnace was introduced at the Hanauer works by Terhune in 1896. It has the following dimensions: Width of hearth, 14 ft.; length of hearth exposed to heat, 129 ft. 6 in. It has four fireboxes, each of 20 sq. ft. grate area. There are six rabbles traveling at a speed of 80 ft. per minute. The furnace puts through from 25 to 40 tons ore in 24 hours. In 10 months there were put through 10,830 net tons of Bingham ore of the following composition: SiO_2 10%, S 39·42%, Fe 31·52%, Pb 9·20%, Zn 4·23%, the sulphur being reduced to 6·37% when 39·18 tons were roasted in 24 hours, to 5·75% when 33·75 tons and to 5·10% when 30 tons. Ore of the composition Pb 26·10%, SiO_2 18·00%, Fe 14·50%, Zn 11·50%, S 26·90%, is successfully roasted without slagging, the sulphur being reduced to 5·4%. Ore crushed through ¼-in. holes and containing SiO_2 9·20%, Zn 16·70%, Fe 16·60%, Pb 17·40%, S 29·00%, Cu 4·20%, is satisfactorily roasted at the rate of 25 tons in 24 hours, the sulphur being reduced to 6·1% with 66·1% of the zinc oxidized which existed as blende in the ore. There was collected from 400 tons ore only 1,100 lb. fluedust in a flue 107 ft. long, or close on to 1·4%, a very small amount for a mechanical roaster. Experience has taught that it is advisable to let the rabbles travel a longer distance outside of the furnace than corresponds to the length of the heated hearth. The rabbles have to be changed after having roasted 15,500 tons of ore or matte. The labor cost of a hand reverberatory furnace treating 7½ tons of ore in 24 hours being $1·26 per ton, with the Ropp furnace treating 40 tons in 24 hours it was $0·237; with 30 tons $0·316, and with 25 tons $0·379. The cost for fuel for treating 10,839 tons of ore was with the reverberatory $26·10, with the Ropp furnace $16·60.

The Keller automatic furnace is used at the Germania Lead Co.'s works and is doing satisfactory work on a limited range of ores, as it depends more on the sulphur of the ore to furnish the necessary heat than other furnaces.

Roasting Furnaces.—Some details of the work done in the Ropp straight-line furnace have been given above. Brown has patented* the combination of a mechanical roasting furnace having a stationary hearth with a cooling floor above the roasting hearth. Part of the cooling floor serves for drying the raw ore before it is fed into the furnace. The floor of the cooling hearth and roof of the roasting hearth are separated by a non-conductor of heat. The hot ore discharged from the roasting hearth is elevated to the cooling floor and slowly moved to the opposite end by mechanical rabbles. The raw ore is fed near the discharge of the cooled roasted ore and conveyed toward the mechanical feed of the furnace. It will be remembered that the Holthoff-Wethey† mechanical furnace has a cooling floor under the hearth. Godshall‡ has patented a mechanical roaster with stationary hearth which follows the principle of the Brown furnace in having on either side of the hearth passages in which trucks carrying the stirrer arms travel. While in the Brown furnaces the material forming the upper division walls is tiles built into the arch, in the Godshall furnace iron brackets protected by bricks are attached to the side walls. Hutson§ has patented a detachable sectional stirring arm with blades for a circular ore-roasting furnace, the arm being secured to the central vertical shaft by a tongue and groove joint covered by a cap piece. Roger‖ has patented a cooling chamber for ore-roasting furnaces. Its essential feature is that a number of cooling pipes pass horizontally through the chamber into which the ore is discharged from the hearth. Other patents are those of Naeff,¶ Manes,** Wehrum,†† and Calnan.‡‡

TYPES OF AMERICAN LEAD BLAST FURNACES, SHOWING INCREASE IN CAPACITY.

Name of Furnace and when Built.	Cubic Contents—Feet.				Center of Tuyere to Feed Floor. Feet.
	Crucible.	Water Jacket Space.	Shaft Proper from Jackets to Feed Floor	Total.	
Furnace, 33x84 in., Leadville, about 1880.......	24·7	72·9	402·5	500·1	14·0
Furnace, 36x80¼ in., original type at the Omaha & Grant Smelting Works, Denver............	32·4	76·1	437·3	545·8	14·9
Furnace, 36x96 in., San Juan & New York Smelting Co., about 1882	60·0	98·8	404·8	563·6	12·6
Original type of furnace, Globe Sm. & Ref. Co., Denver, 36x100 in.; about 1880................	50·0	93·4	510·4	653·8	17·0
Rio Grande Smelting Co.'s furnace, 42x120; 1888..	77·0	165·3	875·0	1117·3	17·0
Present type of furnace, Globe Smelting & Refining Co., 42x140; 1892.	72·0	171·2	1223·0	1466·2	16·0
Harrison Reduction Works furnace, 42x120 in.; 1892....................................	72·0	173·0	990·0	1235·0	18·0
Type of furnace used at Germania Lead Works, Salt Lake, 45x140; 1895......................	72·0	200·0	1565·0	1837·0	20·0

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[*] United States Patent No. 607,420, July 19, 1898.

[†] *M. I.* VI., p. 448.

[‡] United States Patent No. 598,084, Feb. 1, 1898.

[§] United States Patent No. 599,843 March 1, 1898.

[‖] United States Patent No. 600,293, March 8, 1898.

[¶] United States Patent No. 596,533, Jan. 4, 1898.

[**] United States Patent No 602,761, April 19, 1898.

[††] United States Patent No. 597,628, Jan. 18, 1898.

[‡‡] United States Patent No. 613,117, Oct. 25, 1898.

Blast Furnace Construction.—Croll* reviews from a constructive point of view the leading improvements made during the last ten years in silver-lead blast furnaces, and gives as a representative of the most modern type the furnace built at the Germania Lead Works, Salt Lake City, of which he furnishes a drawing. The tendency in blast-furnace construction has been mainly to increase the capacity. This is clearly shown by the preceding table.

The foundations are mostly of slag, which is durable, cheap and does not absorb any lead. The shaft, which was formerly supported by carrier plates or I-beams resting on the columns, is now built on a heavy brick arch sprung between skewbacks resting on the columns, the thrust being taken up by tie rods. While the inner shaft walls used to be made vertical, now they have a bosh of from 12 to 15 in. Of the different arrangements for feeding the furnace and carrying off the fumes, the most common one feeds from the sides through sliding charging doors, and the fumes are carried off from the top through a knee-shaped downtake. With a 42×100-in. furnace the downtake, beginning 30 to 35 ft. above the level of the tuyères, is 5 to 5½ ft. in diameter. The crucible is generally built on a heavy iron plate resting on I-beams or rails; the bottom is thus air-cooled and any leakage of lead is quickly discovered. To the heavy bottom plate the thick side plates are securely bolted; they have strengthening ribs and are reinforced by iron rails. The foregoing table shows a considerable increase in the water-jacketed space. This is due not only to the increase of the section at the tuyères, but also to the greater height of the water jackets. While formerly they were 3½ ft. high, at present their height reaches 4½ and 5 ft. The jacketed space in many old furnaces has been heightened by the use of water blocks. The amount of bosh in older jackets hardly exceeded 6 in., in some recent furnaces it has been increased to 10 and 12 in. (there is, however, a difference of opinion as to the advantage of this step). Cast-iron jackets have replaced most jackets of low-carbon steel. Mr. Croll states that at the Globe Smelting and Refining Co.'s works near Denver, Colo., a furnace, 40×120 in. at the tuyères, having 18 cast-iron jackets, ran steadily for 14 months without the necessity of removing a jacket. The form of the tuyère is much simplified considering the many complex tuyère boxes recently used. In most cases a cast-iron elbow is attached to the tuyère opening in the jacket in such a way that it is air-tight, but also easily removed. The elbow has a peep-hole closed with a wooden plug 1½ to 1¾ in. diameter, having a bore ⅜ to ½ in., which does not quite run through. Any slag entering the elbow ignites the plug, which, contracting, is blown out by the pressure of the blast. The open mouth of the elbow is turned and receives a sheet-iron pipe which is connected by means of a canvas bag to the thimble reaching downward from the bustle pipe. The thimble carries a butterfly valve. The canvas sacks are made heavier since the blast pressures have been raised from 12 to 40 and 48 oz. The bustle pipe, formerly of No. 24 gauge iron for 12 to 18 in. pipes, is now made heavier, No. 12 to No. 8 gauge steel plate; it is also riveted, so as to be absolutely air-tight, and has connecting flanges with male and female joint so as to prevent all leakages. The increase of blast pressure

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* *E. & M. J.*, May 28, 1898.

in smelting has caused some smelters to close the opening in the side of the furnace for drawing off the gases, and to put on a hood connected by a short downtake with the main flue.

Feeding Device.—Terhune has patented* a device for distributing the charge in the blast furnace in such a way that the fines should be fed near the walls and the coarse pieces near the center, thus counteracting the tendency of the gases to ascend near the walls and forcing them to pass more evenly through and give up their heat to the descending charge. It consists of a grate of triangular bars on either side of the furnace below the feed doors. The grates are inclined downward about 35°, and do not reach to the center; the bases of the triangular bars face the sides and the clear spaces correspondingly widen toward the center. In gliding over the grates the charge will first drop the fines, then the coarser particles and lumps will be delivered toward the center.

Water and Air Jacket.—Nesmith patented† for blast furnaces a water jacket having at its back an air jacket. The jacket is supplied with water from an elevated pipe encircling the furnace and having blow-offs for steam. The air jacket is connected with the bustle pipe which incloses the water feed pipe. The main idea of the arrangement is to save cooling water and to reduce the waste of heat.

Porous Furnace Bottom.—Reese patented‡ a blast furnace with a porous bottom for smelting lead-bearing material, especially matte. The lead is to filter through the bottom, while the matte and slag are removed by one of the usual methods continuously or at intervals, the matte being granulated for further treatment.

Influence of Foreign Matter.—All the recent improvements in the treatment of zinc-lead ores will be found under the head of zinc elsewhere in this volume.

ANALYSES OF LEAD SLAGS.

No.	Weight.		Strength of Acid.	SiO_2	Fe	Mn	Zn	SO_3	CaO	BaO	Al_2O_3	MgO
	Before Treatment	After Treatment.										
	Grams.	Grams.	%	%	%	%	%	%	%	%	%	%
Original..	....	....	....	32·0	21·2	1·0	7·0	......	17·1	2·6	5·0	2·5
1	10	8·70	1·0	31·4	18·6	0·7	6·1	3·0	16·6	1·8	4·0	2·5
2	10	7·97	2·5	29·4	15·4	0·5	6·0	8·3	15·8	1·9	3.3	1·4
3	10	7·02	5·0	23·6	7·8	0·2	4·4	21·9	17·7	2·7	1·3	0·5
4	10	7·04	10·0	20·2	3·0	0·0	2·1	35·4	18·1	2·5	0·4	0·2

No.	Weight.		Strength of Acid.	SiO_2	Fe	Mn	Zn	SO_3	CaO	BaO	Al_2O_3	MgO
	Before Treatment.	After Treatment.										
	Grams.	Grams.	%	%	%	%	%	%	%	%	%	%
Original..	....	....	....	33·6	18·6	2·8	5·4	......	14·1	2·7	6·7	4·8
1	10	6·74	1·0	35·0	16·6	2·1	5·0	1·0	14·3	2·6	6·7	3·9
2	10	5·81	2·5	32·4	9·8	0·8	3·9	8·9	12·2	3·6	4·1	2·6
3	10	3·89	5·0	25·8	3·8	0·2	2.5	20·0	14·7	4·8	1·4	1·0
4	10	3·83	10·0	18 4	1·8	0·0	1·2	28·6	17·8	4·8	0·1	0·4

Lead Slags.—Iles§ has studied the condition in which zinc is present in

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* United States Patent No. 585,297, June 29, 1897.

† United States Patent No. 612,754, Oct. 18, 1898.

‡ United States Patent No. 600,620, March 15, 1898.

§ *S. M. Q.*, XIX., p. 197.

some lead blast furnace slags. He crushed two slags to pass through respectively a 30-mesh and a 100-mesh screen, and treated them severally for 25 hours with cold sulphuric acid of different degrees of concentration and then analyzed the slags. The results are given in the foregoing table.

The results prove that the slag is more or less decomposed by the acid. The percentage of silica decreases because sulphur trioxide enters into combination with some of the bases which remain insoluble; others, however, such as those of iron, manganese, zinc, alumina and magnesia, become soluble. Now zinc oxide is readily soluble in dilute sulphuric acid; if it were present as such in the slag, it would be completely dissolved; the tables show that only part of the zinc goes into solution and that very slowly. In analyzing ores previous to smelting in the blast furnace, the zinc compounds other than silicate are decomposed by nitric acid alone or with the addition of potassium chlorate. Zinc silicate, however, requires evaporation to dryness with nitrohydrochloric acid, and sometimes fusion of residue, to get all the zinc into solution. From the results in the tables and from common experience in ore analysis, Iles feels justified in drawing the conclusion that zinc exists in slags chiefly as silicate. It is a generally accepted fact that in smelting zinc-bearing sulphide ore some zinc is present in the slag as sulphide. Iles proved this to be the case by blowing air under 8 lb. ? pressure through molten slag and filtering the dense white fumes issuing from it through both muslin and woolen cloth. An analysis of the fume gave SO_3 16·1%, total S 6·9%, Zn 7·9%, Cu trace, Pb 62·2% (fire assay) and 64·0% (wet assay), Ag 2·5 oz., Au a distinct trace. Iles believes that zinc and lead in the fume were present in the following combinations: ZnS 1·51%, ZnO 8·58% $PbSO_4$ 60·94%, PbO 24·07%, total 95·10%, and that the difference of 4·90% was made up of potash, soda, arsenic, antimony, tellurium, etc.

Fuels.—Lang* discusses briefly the influence of altitude on smelting. As increased density of the atmosphere adds to the calorific intensity of combustion, smelting operations at high elevations must require more fuel than when carried on nearer sea level. He believes that by closing the top of a blast furnace and keeping the gases under higher pressure than can be had with an open top, a saving of fuel will be effected and at the same time the loss in volatilization will be reduced.

Chemistry of Blast Furnace Smelting.—Herzfeld† discussed the chemical processes that take place in the burning of limestone, as to which reference should be made to the paper on "Lime and Lime-burning," elsewhere in this volume.

Furnace Products.—*Speise.*—H. C. Hahn‡ examined some dark-green hexagonal crystalline plates (angle 119·5°) with metallic luster that he found on a hearth accretion of a lead blast furnace. The analysis gave Cu 61·03%, Fe 0·97%, Pb 0·44%, As 35·50%, Sb 1·08%, S 0·89%—total 99·91%; which shows it to be a copper speise of the formula Cu_2As. Its sp. gr. at 10° C. was 7·976; at 20°, 8·008; at 30°, 7·575.

Fluedust.—Dust collectors have been patented by Sergeant§ and Porbeck.‖ In briquetting fine (raw and roasted) ores and fluedust, machinery has, at

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* *E. & M. J.*, Jan. 29, 1898. † *Th. Z.*, 1898, p. 374. ‡ *E. & M. J.*, April 2, 1898.

§ United States Patent No. 597,660, Jan. 18, 1898. ‖ United States Patent No. 603,285, May 3, 1898.

least with large works, replaced hand labor. The White improved mineral press* has been introduced at several works. It consists of a roller mill, with a pan 7 ft. in diameter, in which the material to be bricked is thoroughly mixed with lime, and a molding disk, 4 to 5 ft. in diameter and 2½ in. thick, having 24 openings, 4 in. in diameter, serving as molds. The machine occupies a floor space of 9½ × 11½ ft. and is 6 ft. high. It requires 25 h. p. and makes 80 cylindrical briquettes per minute.

DESILVERIZATION OF BASE BULLION BY THE PARKES PROCESS.

Blakemore† has prepared an elaborate paper on the refining of base bullion running high in gold in Australia, of which the following is an abstract: Apparatus and mode of operation are similar to those in use in the United States. The paper describes a plant treating weekly 80 tons of base bullion, having softening furnaces of 15 and desilverizing and refining kettles of 10 tons capacity. In the softening furnace the use of chrome and magnesite brick is advocated, although their cost is very high—£57 10s. per thousand, weighing 4½ tons.

Dross.—The bullion treated is very impure, making large amounts of dross. The dross is very rich in gold, as shown by the subjoined assays of base bullion and resulting dross:

Base Bullion.		Copper Dross.			
Ag, Oz. per Ton.	Au, Oz. per Ton.	Pb, %	Ag, Oz. per Ton.	Au, Oz. per Ton.	Cu, %
143·03	12·25	63·10	67·36	6·00	5·1
168·68	12·00	66·06	53·46	5·66	3·8
117·49	12·92	60·70	41·31	1·32	1·8
118·04	12·48	69·10	39·86	1·32	1·5
121·16	14·78	73·50	60·66	1·49	3·6
105·30	11·28	54·10	38·24	0·54	1·9
117·42	13·52	72·70	53·67	3·53	3·9
114·08	11·95	76·60	52·40	4 92	6·4
81·12	6·61	67·60	43·77	6·76	1·8
115·30	11·26	60·70	37·86	6·78	3·3
148·54	16·94	68·80	58·14	5·86	1·5
220·02	21·98	73·50	62·24	6·48	1·4
179·40	42·28	82·40	60·96	10·00	1·5
139·32	16·08	62·40	95·56	16·00	6·1
340·04	37·38	62·20	85·34	8·40	1·9
157·34	16·53	70·80	27·98	2·86	1·4
141·53	14·30	71·00	31·20	3·36	1·1
121·74	12·94	78·40	58·08	6·00	1·6
132·72	12·90	76·10	60·06	7·50	1·7
246·38	24·04	60·60	114·14	20·44	9·3
154·52	13·34	75·10	48·56	5·44	2·6

Gold in Antimony Skimmings.—The antimony skimmings (which assay of course much lower in silver than the dross), however, still run very high in gold, as may be seen by the following table:

Ag, Oz. per Ton.	Au, Oz. per Ton.	Pb, %	Metallics. %	Average Sample of, Tons.
15·24	1·70	48·20	1·8	2·90
15·81	2·17	47·40	1·2	20·00
13·43	1·46	47·40	1·2	34·00
12·91	1·41	51·80	1·2	30·00
15·87	1·85	51·40	2·0	40·00
16·47	2·08	48·20	1·4	15·00
18·27	1·94	48·70	1·3	18·00

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* *E. & M. J.*, Oct. 8, 1898.

† *Trans. Austral. I. M. E.*, 1898, V., p. 220.

Before desilverizing the softened bullion Blakemore lays stress on allowing the lead in the kettle to cool until a ring forms at the edge. This is to insure that as much of the copper as possible shall enter the dross. Assays of kettle dross are shown in the following table:

Ag, Oz. per Ton.	Au, Oz. per Ton.	Cu, %	Ag, Oz. per Ton.	Au, Oz. per Ton.	Cu, %
143·38	87·36	2·20	131·67	52·63	2·30
280·33	86·49	3·10	182·39	82·65	10·20
161·64	77·06	2·30	145·51	49·04	3·90
211·37	76·94	3·60	247·82	105·08	9·40

When the kettle was drossed hot and the zinc added immediately afterward, only 50% of the gold was recovered in the doré silver; while when it was allowed to cool the yield was raised to 90%.

Imperfect Extraction of Gold.—In desilverizing bullion rich in gold it is not possible to separate the gold completely from the silver, as the silver crust always retains small amounts of it. This is clearly shown by the following:

Assays of Eleven Tons Base Bullion, Oz. per Ton.											Assays of Apparently Clean First Silver Crusts Oz. per Ton.	
Before Zinking.		After Zinking.										
Au.	Ag.	Lb. Zinc.	Au.	Ag.	Lb. Zinc.	Au.	Ag.	Lb. Zinc.	Au.	Ag.	Au.	Ag.
14·78	121·16	200	0·004	108·80	30	Trace.	76·90	10	*Nil.*	67·23	0·26	927·48
12·22	124·22	211	0·05	119·59	35	0·008	53·44	20	*Nil.*	45·28	0·40	421·84
15·14	149·26	193	0·03	136·11	70	*Nil.*	44·02	....	....	....	0·50	653·80

The following assays of apparently clean first silver crusts give further evidence of the imperfect elimination of gold:

Ag, oz. per ton...........	915·13	519·49	644·76	785·38	715·77	102·28	891·35
Au, oz. per ton	0·29	0·32	0·14	Trace.	0·13	0·06	0·05
Ag, oz. per ton	785·38	715·77	317·48	891·35	583·94	145·80	534·61
Au, oz. per ton...........	Trace.	0·13	Trace.	0·05	0·06	*Nil.*	0·21
Ag, oz. per ton...........	792·29	510·86	133·18	573·42	156·59	....	....
Au, oz. per ton	0·02	0·04	Trace.	Trace.	*Nil.*	....	....

The work that zinc will do in removing the gold with as little silver as possible, in desilverizing with the aim to extract all the gold and as much silver as possible with it, and in desilverizing lead free from gold, is shown in the following tables, in the order referred to, the corresponding crusts obtained being shown in the next three tables:

Tons of Lead in Kettle.	Total Gold in Kettle, Oz.	One Lb. Zinc will take out	
		Oz. Gold.	Oz. Silver.
11	300	1·30	1 to 3 with it.
11	200	1·00	"
11	150	0·79	"
11	100	0·59	"
11	60	0·45	"

Tons of Lead in Kettle.	Total Gold and Silver in Kettle.		One Lb. Zinc takes out		Assay of Kettle after First Zinking.	
	Au.	Ag.	Au.	Ag.	Au.	Ag.
11	494	3,110	0·59	3·60	*Nil.*	10·04
11	194	1,663	0·34	2·83	*Nil.*	3·88
11	296	1.934	0·45	2·95	Trace.	17·46
11	443	1,883	0·64	2·80	Trace.	4·10
11	330	2,417	0·45	3·34	*Nil.*	8·21
11	204	1,638	0·36	2·86	Trace.	5·22
11	143	1,330	0·28	2·65	Trace.	4·04
11	123	1,320	0·23	2·54	Trace.	20·14

Tons of Lead in Kettle.	Total Silver in Kettle, Oz.	One Lb. Zinc will remove Oz. Silver.	Lead Assay after Zinking, Oz. Silver per Ton.
11	1,380	5·3	2·08
11	1,450	5·6	3·44
11	1,200	4·1	4·78
11	1,110	3·9	0·70
11	775	3·5	0·69
11	930	3·8	1·59
11	616	3·4	0·54
11	560	3·2	1·70
11	460	2·6	0·54
11	132	1·1	0·40
11	55	0·55	*Nil.*

Tons Lead in Kettle.	Assay Value, Oz. per Ton.		Zinc Added Lb.	Crust Skimmed. Lb.	Crust, Per Cent, on Lead in Kettle.
	Au.	Ag.			
11·48	21·02	156·54	225	2,454	9·67
11·259	15·88	179·60	205	2,792	11·16
11·90	22·80	145·30	230	3,682	13·80
10·17	17·52	121·08	208	2,049	8·90
10·308	19·86	159 82	215	2,442	10·54
10·279	20·58	146·26	212	2.101	9·11
10·02	14·26	136·53	188	2,340	10·18
11·608	14·54	129·92	203	2,675	10·29
10·05	12·42	132·68	176	2,166	9·60

Tons Lead in Kettle.	Assay Value. Oz. per Ton.		Zinc Added. Lb.	Crust Skimmed. Lb.	Crust Per Cent. on Lead in Kettle.
	Au.	Ag.			
11·89	16·52	195·48	638	5,056	20·50
11·85	15·98	151·71	534	4,454	19·58
11·65	13·12	118·88	502	4,625	17·90
10·01	14·00	159·90	557	5,470	24·95
10·41	15·00	141·20	540	4,186	16·64
10·87	12·30	132·02	444	4,607	20·72
11·29	12·25	148·03	495	3,894	14·47
10·63	24·04	246·88	667	6.154	24·85
11·62	13·34	154·52	480	4,016	17·54

Tons Lead in Kettle.	Assay Value. Oz. per Ton.		Zinc Added. Lb.	Crust Skimmed. Lb.	Crust, Per Cent. on Lead in Kettle.
	Au.	Ag.			
11·89	*Nil.*	67·23	220	2,285	8·58
11·85	*Nil.*	64·21	217	2,424	9.02
11·65	*Nil.*	65·02	228	3.019	11·56
10·01	*Nil.*	56·24	175	1,933	8·61
10·41	*Nil.*	61·22	200	2,210	9·47
10·87	*Nil.*	109·28	284	2,656	10·96
11·29	*Nil.*	34·64	165	2,584	10·21
10·63	*Nil.*	75·64	231	2,085	8·55
11·62	*Nil.*	63·42	223	1,690	6·49

In desilverizing 1,775 tons of unsoftened base bullion averaging 14 oz. gold and 131 oz. silver per ton in kettle charges of about 10 tons, 20·05 tons spelter and 8 09 retort zinc (or 1·59% zinc) were required.

Reverberatory for Liquating Crusts.—For liquating the zinc crusts Blakemore advocates the use of a reverberatory furnace. He obtained from charges weighing 1,200 lb. the following weights of liquated gold crust: 510, 450, 520, 542, 633, 666, 707, 646, 661, 595, 672, 690 lb.; and of liquated silver crusts: 487, 484, 560, 560, 480, 422, 348, 504, 514, 534, 404, 448, 323 lb. He cautions against overheating the crust, as some of the values are liable to re-enter the liquated lead, as seen by the two following tables:

I. LIQUATION OF GOLD CRUST.
(Ounces per Ton.)

Liquated Gold Crust.		Liquated Lead.	
Ag.	Au.	Ag.	Au.
1047·66	209·45	6·92	Trace.
1195·66	329·12	(*a*) 17·07	(*a*) 4·29
1302·58	279 61	5·78	Trace.
1579·93	311·59	6·10	Trace.
1886·35	336·40	10·87	0·06
2083·65	322·75	(*a*) 17·90	(*a*) 0·20
2035·80	384·20	6·42	0·02
1975·40	337·30	5·04	Trace.
1470·00	211·70	22·03	0·02
1513·64	298·15	21·68	0·01
1964·10	307·25	7·52	0·04
1439·96	149·10	7·04	Trace.
2014·85	164·55	15·53	Trace.

II. LIQUATION OF SILVER CRUST.
(Ounces per Ton.)

Liquated Silver Crust.		Liquated Lead.	
Ag.	Au.	Ag.	Au.
2077·61	0·25	3·98	Trace.
2010·41	0·27		
1765·64	0·10	11·94	Trace.
1949·12	0·18		
1351·90	0·18	1·94	Trace.
2061·37	0·25		
1505·90	0·08	7·10	Trace.
701·04	0·10	2·19	Trace.
1977·79	race.	(*a*) 25·10	Trace.
2005·52	0·06	(*a*) 41·89	Trace.
2111·10	0·10	11·46	Trace.
1614·00	0·46	(*a*) 15·89	Trace.

(*a*) Crust overheated.

Steaming Kettles.—Following the desilverized lead to be refined, the method of steaming appears to be common. The kettles are of the same size as those for desilverizing. Blakemore advocates the use of steel kettles, which, with a capacity of 14 tons, stand 60 charges; he lays stress on heating up the lead to a cherry red in not less than four hours, as quicker heating shortens the life of the kettle. On account of the uncertain character of the castings he recommends in purchasing kettles contracting for them according to the number of charges that can be refined in them, as then it becomes the interest of the foundry to furnish only first-class kettles.

Fuel.—The fuel used in the three operations of softening, desilverizing and refining varies with the quality of the bullion and the size of the charge. With very impure bullion and a softening furnace of 15 tons capacity the coal consumed per ton of unsoftened bullion was 680 lb. for softening, 60 lb. for desilverizing and 40 lb. for refining (steaming).

Retorting.—The retorts used are those of the class 409 of the Battersea Crucible Co.; they hold 560 lb. of crust and are good for 24½ charges. In retorting 475 charges, the following weights of products were obtained per average charge of 552 lb. crust: 423 lb. retort bullion, 48·09 lb. zinc, 61·40 lb. retort dross, 21·20 lb. blue powder, and the coke consumed was 506 lb. Assays of products are subjoined:

ASSAY OF PRODUCTS FROM RETORTING GOLD CRUST.
(Ounces per Ton.)

Retort Bullion.		Retort Dross.		Blue Powder.	
Ag.	Au.	Ag.	Au.	Ag.	Au.
1860 9	243·6	1923·40	141·30	7·32	0·04
2533·2	304·0	1866·40	168·80	20·52	1·84
2621·0	155·7	2343·90	416·20	15·32	1·02
2418·9	182·5	1875·66	183·60	1·18	Trace.
2592·0	96·0	1044·10	295·20	20·78	2·74
2866·7	371·1	1367·90	141·70	2·18	Trace.
2933·8	182·1	1601·10	241·85	5·47	Trace.
2702·2	450·0	2074·25	347·85	332·21 (*a*)	52·24 (*a*)
3119·9	483·3	1579·15	236·05	6·89	0·59

(*a*) Some rich bullion must have run into the condenser.

ASSAYS OF LIQUATED SILVER CRUST WITH RETORT BULLION AND DROSS.
(Ounces per Ton).

Liquated Silver Crust.		Retort Bullion.		Retort Dross.	
Ag.	Au.	Ag.	Au.	Ag.	Au.
1842·56	0·11	2187·1	0·1	1583·55	0·45
2133·02	3·63	2659·4	15·2	1665·50	8·60
2049·35	0·71	2417·0	2·0	1708·40	24·60
2191·41	0·35	2377·5	1·5	1786·50	1·00
1097·84	1·30	2567·1	4·8	1811·60	2·90
1727·34	0·40	2496·5	1·1	2098·45	0·45
1912·04	0·62	2570·1	2·1	2552·95	1·65
2219·16	0·84	2619·2	4·8	1457·20	1·35
2174·27	0·94	2540·3	5·1	2183·05	0·90
2392·12	0·76	2949·6	3·6	1863·60	1·30
1868·61	0·39	2136·8	0·6	1758·27	1·28

These tables exemplify how a small amount of gold in the silver crust as drawn from the kettle becomes concentrated in the subsequent operations.

Effect of Copper.—The high percentage of gold in the dross is mainly due to the presence of copper. Blakemore believes that an alloy of copper-zinc-silver-gold is formed; he is led to this by having repeatedly a matte-like brittle substance on the retort bullion after it had been poured and cooled. An analysis showed: Cu 29·00%, Zn 55·80%, Pb 3·00%, Au 2·639%, Ag 9·847%—total 100·286%.

Cupelling.—In cupelling in the English furnace, a water-jacketed concentrating test holding about 1,500 lb. lead, and a finishing test holding about 1,000 lb. lead are used. The hearth material is made up of the following mixtures: Limestone, 7, 4, 8 parts; magnesite 0, 4, 0, parts; cement, 2, 1, 1 parts; fire clay, 1, 1, 1 part. They work about equally well. The following table gives some assays of retort bullion and resulting litharge in ounces per ton:

Retort Bullion.		Litharge.	
Ag.	Au.	Ag.	Au.
10089·2	942·5	22·96	1·12
14278·9	1137·8	78·90	5·14
17217·8	1547·6	28·22	0·36
15743·0	1235·6	61·16	3·38
13804·8	1198·4	225·78	1·04
4242·9	481·2	10·64	2·11
6689·3	523·9	18·39	1·26
13814·1	1794·5	11·48	2·45
16775·2	2128·8	17·07	4·29
12341·8	1634·4	23·16	1·60

Working of Dross.—The dross obtained from melting down base bullion in the softening furnace of a Parkes desilverizing plant is usually added to the blast furnace charge after it has undergone a preliminary liquation. The question has sometimes arisen, What is the loss in lead in smelting this liquated dross with sulphur-bearing materials ? It is believed that it sometimes reaches 3%, but usually is nearer 2%. Liquated dross of the composition Cu 32%, Pb 47·1%, Ag 293·1 oz. and Au 10·5 oz., when smelted with sulphur-bearing material in a reverberatory furnace, gave base bullion: Pb 90·6%, Cu 5·4%, Ag 355·3 oz., Au 8·7 oz.; speiss: Pb 26·6%, Cu 47·0%, Ag 422 oz., Au 25·8 oz.; matte, Pb 15·8%, Cu 62·3%, Ag 164·3 oz., Au 0·56 oz.; slag: Pb 19·8%, Cu 7·8%, Ag 14·0 oz., Au 0·04 oz. With the large amounts of arsenical and coppery material that the lead works located at smelting centers have to treat, the affinity of arsenic for copper becomes apparent in the production of lead-bearing copper speiss. Such a speiss contained:[*] Pb 29·1%, Cu 31·0%, Fe 11·4%, SiO_2 0·7%, Ag 310 oz., Au 7·8 oz. It pays to liquate such a speiss before smelting for matte and slag in the reverberatory furnace.

PARTING DORE SILVER.

By Sulphuric Acid.—Blakemore[†] describes the Gutzkow process of parting doré silver as carried out in Australia. His description is based on a plant treating 200 lb. doré silver in 24 hours. The doré silver coming from a zinc desilverizing plant has the following values: Ag 892·00, 938·00, 941·40, 921·50 and 924·00; Au 10·39, 56·78, 53·82, 67·79 and 74·18. It is cast into plates 12×10×½ in. and is charged into the solution kettles, which are heated from below and have a capacity of 1,000 lb. sulphuric acid, 66° B. A charge consists of 200 lb. alloy and 250 to 300 lb. sulphuric acid of 66° B. The silver is dissolved after three or four hours' boiling, in which time some mother liquor from a previous charge is added to prevent the kettle from boiling over. When the silver is all dissolved, enough cold mother liquor is added to have about 4 lb. acid to 1 lb. of silver. The diluting and cooling of the silver sulphate causes some of the salt to fall out, which carries down the float gold. The powdery gold is now bailed out with an iron ladle; the silver solution is allowed to clear for half an hour and is then siphoned off by means of a ¾-in. iron pipe into one of two flat cast-iron cooling pans 6×3×1 ft., which may be water jacketed or not. Steam is turned into the pan until a sample of the solution dropped on a cold plate shows that the monosulphate falls out in the form of yellow crystals surrounded by mother liquor. At the beginning the bisulphate forms, which is indicated by a solid white lump practically free from mother liquor. An hydrometer test will show 135° Tw. (70° B.). In starting, it takes 10 hours' steaming to bring the solution to the right point. When mother liquor, however, has been added, steam need be introduced only for three hours. The solution is now allowed to stand until the next day, when nearly all the silver sulphate will have crystallized out. The mother liquor is siphoned off and the silver sulphate crystals are shoveled into a lead-lined box where most of the copper and iron sulphates are removed by washing with water. The silver

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[*] Private communication.

[†] *Trans. Austral. I. M. E.*, 1898, V., p. 213.

sulphate crystals are transferred to a box on wheels having a false bottom and cotton filter, to be washed four times with hot water. They are now removed to a drying pan, dried in about two hours and then decomposed by melting down in a graphite retort (capacity 560 lb. silver crust) in charges of 260 lb. in about four hours. From 4 to 5% charcoal or coke are added to the charge, as well as about 5 lb. clay, which retards the corrosion of the retort at the level of the molten charge. Besides a smell of sulphur no troublesome fumes are evolved. (In one refinery in the United States this method of decomposing silver sulphate was tried, but given up on account of the great loss in silver.)

The silver, ·985 to ·995 fine, is cast into bars, which are melted down in a cupelling furnace and the silver brought to a fineness of ·998. All the silver washings are run through a box holding metallic copper and thence through a second one holding scrap iron. The precipitate in the copper box is removed monthly; a sample showed Ag 53·15%, Cu 10·40%, the rest being iron, dirt, etc. Any richer parts resembling silver sponge are separated and treated with silver sulphate in a barrel filled with hot water. The gold from a week's run is put back into the solution kettle and boiled with concentrated sulphuric acid (1 lb. acid to 2 oz. gold) for three hours, then ladled out (while the acid is simmering) into a lead-lined box and washed with hot water for two to three hours. It is then dried and melted down in a crucible, giving gold ·980 to ·995 fine.

By Electrolysis.—Butler* gives a brief illustrated description of the Moebius process for parting doré silver as carried out at the works of the Guggenheim Smelting Co., Perth Amboy, N. J. Wohlwill† discusses from an electrochemical point of view the precipitation of gold from a gold chloride solution. An outline of the mode of operating at the Hamburg (Germany) refining works was given in THE MINERAL INDUSTRY, Vol. VI., p. 461.

The Gutzkow Process of Parting Gold and Silver.—The improved Gutzkow process which was described in THE MINERAL INDUSTRY, Vol. VI., pp. 459–460, has been criticized by B. Edelmann, who claims that there was no novelty in the original Gutzkow process, which has been in use since 1870 at the works of the Deutschen Gold und Silberscheideanstalt at Frankfort-on-the-Main, Prussia. In a communication to the *Berg-und Hüttenmännische Zeitung*, of Feb. 17, 1899, Gutzkow points out that the process was patented by him in the United States, Dec. 18, 1868. No patents were taken out in Germany, and the process was described by the inventor to Dr. H. Roessler, then the manager of the Frankfort works, during a visit in 1870. The process is known in Germany as the "Roessler Process," and as such was introduced at Lautenthal and in German metallurgical literature. It was described by Dr. Roessler in *Annalen der Chemie*, 1896, without mention of Gutzkow's name. In the "Roessler Process" metallic iron is used instead of iron sulphate to decompose the silver sulphate, but according to Gutzkow this is not a material difference. The dates given by Gutzkow seem to establish the priority of his invention.

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* *Jl. Fed. Can. M. I.*, III., p. 120.

† *Z. E. C.*, 1897-98, pp. 379, 402, 421.

PRESENT PRACTICE IN SILVER-LEAD SMELTING AT DENVER AND PUEBLO.

BY L. S. AUSTIN.

THE so-called "Valley Smelters" of Colorado are the Globe Smelting and Refining Co. and Omaha & Grant Smelting and Refining Co., of Denver; the Boston & Colorado Smelting Co., of Argo, near Denver; the Pueblo Smelting and Refining Co., the Colorado Smelting Co., and the Philadelphia Smelting and Refining Co., of Pueblo; and the American Zinc-Lead Co. at Canyon City. The Globe and the Pueblo Smelting and Refining Co. refine their own bullion, and the former parts its doré bars, selling its gold to the local United States mint and shipping its silver to the Orient via San Francisco. The Boston & Colorado also works up its precious metals to the same point, and ships away its copper as copper oxide or as black copper. In this case, however, the copper still carries 10 oz. silver to the ton, which is too little as yet to justify electrolytic work in refining it. The improvement in electrolytic copper refining raises the question whether it would not be better to bring the copper to the state of blister, and complete the work by electrolysis at some Eastern point. The Philadelphia smelter is the only one of the lead smelters which brings its copper to the condition of blister. At other works the copper matte is concentrated to 40 to 45% copper and 10% lead and then shipped East, or it may be concentrated to white metal. In either case the endeavor is to keep the tenor in precious metals and lead as low as possible, since they are not paid for at their full value. The American Zinc-Lead Works treat zinc-bearing ores, which are avoided by the other smelters, or which, when handled by them, carry a zinc penalty of 50c. per unit for all over 10% zinc contents.

Ores.—The valley smelters draw their supplies from Leadville, Cripple Creek, Aspen and other camps of Colorado. They buy a large amount of Cœur d'Alêne lead ore, and get a good deal of silicious ore from Utah, which carries more or less lead. The Argo works, while getting a large proportion of its copper ores from the immediate vicinity, draws its supply from a large territory. Its uses no lead ores and those lots which perchance go high in lead are resold to the lead smelters. A considerable quantity of the iron ore used for fluxing comes from Leadville, and carries silver in quantity sufficient to justify freighting it out. The rate on ores from this point to Pueblo and Denver is $3·30 per ton. Local smelting works, which smelt ores to a copper matte, ship that product to one of the valley smelters.

Sampling.—The Cripple Creek ores average 60% silica, and carry their gold contents very irregularly distributed. They are regarded as the most difficult to sample of the ores treated by the smelters. To insure this the entire lot is coarsely crushed to pass 1 in. mesh and the assay sample is ground to pass a 100 or 120-mesh screen. The result of such crushing is to produce a good deal of fines and dust which carries twice the value of the average of the ore, which is one reason for a thorough system of dust collection. It has been held by some that the smelting of Cripple Creek ore results in a loss of gold because it there occurs united with the volatile element tellurium. Dr. F. L. Bartlett, however, denies this, saying that where the bag-house system is used the fines already mentioned are caught, and this saving will account for the loss.

The public sampling mills of the State and the chlorination and cyanide works have freely adopted automatic sampling methods, but the smelters are much more conservative. They are willing to adopt it to a limited extent as a starting method, but they trust to hand sampling for the finish. The sampling mills naturally wish to sample cheaply and the chlorination and cyanide mills handle lower grade ores, hence their preference for automatic sampling. I am not to be understood as saying that automatic sampling is less exact, for the contrary may be true, since it removes the influence of the laborer from the problem; but it must not be forgotten that the miner is conservative and suspicious, and it is well to follow a method which is satisfactory to him. The plan of resampling the ore and thus checking the work is an excellent one, though in the case of hand sampling it adds much to the expense. With automatic sampling this duplication of samples may be readily and cheaply effected. Duplicate samples of Cripple Creek ore, as at present made, have been known to vary 0·5 oz. or more.

Taking account of the fact that the richer pieces or particles of the ore increase in value as they become smaller, I have calculated the following proportions based on probable specific gravities and values: The ore crushed to 1 in. may be quartered down to 6,750 lb. It is recrushed to pass a 4-mesh screen and may then be cut down to 140 lb. After recrushing to pass a 16 or 20-mesh screen it may be again cut down to 1 to 1·5 lb., all of which is to be ground to 100 to 120 mesh, and reserved for the assay sample. It should be thoroughly mixed in a Wedgewood mortar, as it has been found that mixing on a cloth or by a spatula is imperfect. After each grinding a thorough mixing is essential.

Assaying.—In Colorado the assay work used to be done largely by scorification, now the crucible method is much used since it permits larger amounts to be taken. For ores containing considerable gold, such as those of Cripple Creek, two assays of 0·5 assay ton are regarded as sufficient, but for ores low in gold as many as three of 1 assay ton each are taken. Where formerly variations in results between buyer and seller of 0·05 or 0·06 oz. gold per ton were allowed on low-grade gold ores, now these differences must not exceed 0·03 oz. and on high-grade ores 0·05 oz. With greater differences an umpire is chosen, whose results are accepted as final, and the expense of his work is borne by the party who differs most from him. To meet these special requirements local makers have entered on the manufacture of balances especially suited to close, accurate and quick work. These have short beams 4 to 5 in. long, and are provided with magnifying glasses for accurate reading of the scale. They have little or no "kick," *i. e.*, when the beam is lifted it does not sway irregularly, but the first movement of the pointer at once indicates whether an addition or subtraction of weight is needed. An average day's work of an assayer at one of the large smelting works may be counted at 20 samples. Upon these samples there would be about 30 crucible assays, 30 scorifications and 20 crucible assays for lead, or 80 determinations in all. This can be done in about eight hours with the assistance of a helper for half a day.

Bricking or Lumping the Ores.—Increasing use is being made of the brick press, not only for the fluedust, but also for a good deal of the finely ground ore, both oxidized and roasted. In other words, it is thought best not only to carefully save the fluedust, but also to make as little as possible. Following a custom which I first observed at the El Paso smeltery, Texas, a mixture is made of certain silicious ores or oxidized tailings, fine roasted ore, and fluedust, all held together with lime in the form of milk of lime, which is added to the extent of from 3 to 8%. The briquettes are pressed quite dry, and are piled up and allowed to harden. In winter they may quickly freeze and are then in excellent condition to feed into the furnace. In summer in this dry climate they become hard in a few days by desiccation, and by the usual reaction of lime when in presence of silica. Another excellent method is to mix the materials in an edge mill, and to shovel out and wheel away the mixture to an ore bed, where it is distributed shovelful by shovelful over the surface. Here it has time to dry and harden before it comes to be used. An improvement upon this would be to use a mill with an automatic discharge, much like those used in the fertilizer business.

Roasting.—The practice in roasting is in a transition state, and it is hard to predict its future. The various types of automatic roasters are doing excellent work in roasting dry, *i.e.*, non-lead-bearing ores, but for the peculiar conditions of the lead smelters they have not come up to expectations. The various smelting works which put them in, found out their shortcomings, and a reaction set in in favor of the long-bedded hand roaster, which will put through per 24 hours 12 to 15 tons of the crude ore crushed to 3-mesh or 4-mesh size, and will roast down to 3% sulphur. In the automatic furnaces the roast may go down to 5%, but if it be attempted to go lower the ore softens, agglomerates, sticks to the rakes and to the hearth until it becomes necessary to shut down. The furnace then cools off, and will not do good work until thoroughly heated up again, which may take one or two days. The rakes in ordinary operation also drag forward a little of the ore in advance of its proper position, raising the grade of the final product in sulphur. This is exemplified in hand roasting, where the charges are kept entirely separated by the careful furnaceman who wishes to insure a low sulphur in his roast. Of the better-known automatic roasters the Pearce turret furnace seems best arranged for permanence and facility of repairs. Its capacity, however, is less than that of the roasters of large hearth area, whose drawbacks in other respects have already been mentioned. Dr. Pearce is having excellent success with it at the Argo works, where he uses a step grate, hopper feed and a closed ash pit with forced draft.

The success of the Brückner cylinder in Utah has brought this roaster to the attention of the lead smelters of Colorado, and the Globe is putting in 12 and the Pueblo Smelting and Refining Co. five. They are 8·5 ft. in diameter by 26 and 28 ft. length and have a capacity of 24 tons. If they can be so driven as to roast a charge in 48 hours this will be equal to 12 tons per day. These roasters differ from all others in that they do not carry a uniform fire. Upon dropping the charge the firing is urged until the ore is "caught," when the fire is withdrawn, and the ore burns by its own heat. The intensity and consequently the duration of this stage depends upon the height above the sea.* At the close of this stage the fire is gradually increased until the finish. Now, unless the firing is properly managed, the charge, especially when containing much lead, may hang up, agglomerate, and because of this, imperfectly roast. Even with careful firing this cannot altogether be avoided. There is the satisfaction of knowing, however, that when the cylinder begins to be thus crusted, the ore may be easily sliced off, and if the agglomeration is incipient, the charge will crumble again. To the objection that the cylinder makes much dust it can be replied that we do not know yet how slowly the cylinder can be revolved and still do satisfactory work. It is quite possible that by slow movement the fluedust loss can be reduced to as little as in other automatic roasters. A great merit of the Brückner cylinder is its small cost for repairs, its durability, and its certainty of action. It appears to be capable of much development in the future by using oil residuum for heating, by the judicious introduction of air, and by the application of the principles of open-hearth practice.

The Blast Furnace.—The blast furnace for lead smelting is being increased in size and in height with a view of obtaining larger tonnage from the single unit. Increased height permits higher blast pressure without making more fluedust, especially if the bosh of the furnace is also increased, from which results a larger area through which the blast may escape. The large tonnage permits the use of matte-separating appliances which might be troublesome to apply to smaller and slower furnaces. The higher pressures (30 to 40 oz.) require heavier and better-built machinery, both engines and blowers. The rotary blower of the Root type, so generally used, is very carefully made so as to diminish slip, and may be direct and flexibly connected to its engine. Such a connection is well exemplified on some of the direct connected generators of the Westinghouse Electric Co. In these a slight defect in alignment in no way affects the smooth operation of the apparatus. Each furnace should be served by its own blower. I am still of the opinion that in the future the lighter types of cylinder blowing engines will be used, together with tight iron connections to the furnace, so that the metallurgist will count the revolutions of his engine rather than observe his blast pressure. Still on the present lines excellent work is done and the proposed improvement seems to be by no means vital.

Mr. Livingstone, of the Omaha & Grant works in Denver, with a view to increasing the height of the smelting column, has lately done away with the down-take beneath the feed floor, and added a sheet-iron hood, such as is used at the Arkansas Valley smeltery at Leadville. This sensible and inexpensive improvement enables him either to increase the tonnage of the furnace or, as he seems to prefer, to give it a cooler top, less fluedust and slightly less fuel. The hood can be readily removed and the accretions cut off.

A new style of water-jacketed furnace, built by the Colorado Iron Works Co., has been put in use and found to be eminently successful. The jackets require but little attention and are economical of water, since the water is permitted to boil away. The amount used may be as little as one-tenth of that

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* Hofman, *Metallurgy of Lead*, 1899, p. 202.

required under the ordinary system, which marks the furnace as specially adaptable to use in a dry country.

The Pueblo Smelting and Refining Co. continues to use the automatic system of charging. In its case there is a saving of 20 men, but simple as it seems it is a question whether it would be successful elsewhere, since so much depends upon experience in its use and upon the personal equation. I have elsewhere expressed myself as to this subject.[*]

Slags.—The type of slag chiefly used at the "valley" works is the so-called three-quarter slag (SiO_2, 33%; $Fe(Mn)O$, 33%; $Ca(Ba,Mg)O$, 24%). A good deal of zinc is used on the charge replacing lime. This decrease in lime calls for a decrease in silica, in accordance with the dictum "a high silica requires a high lime;" and thus the new proportions become: SiO_2, 31%; $Fe(Mn)O$, 33%; $Ca(Ba,Mg)O$, 18%. Subjoined are analyses of slags at the Globe and the Omaha & Grant Works, Denver:

(1) Globe: SiO_2, 31·5%; $Fe(Mn)O$, 33·5%; $Ca(Ba,Mg)O$, 20·0%, ZnO, 5·0%—total, 90·0%.

(2) Omaha & Grant: SiO_2, 30%; $Fe(Mn)O$, 33%; $Ca(Ba,Mg)O$, 16%; ZnO, 9%—total, 88%.

It will be noticed that the latter (Omaha & Grant) varies largely from the usual proportions because of the larger amount of zinc carried upon the charge, but it is in the direction the conditions call for.

Separating Appliances.—The ordinary settling pot still continues to be used, or preferably two of them, the second one being a guard to the first. The separation while as good as ever does not compare with that effected in the large cast-iron box or forehearth. This, when changed before it fills up to any extent, does quite satisfactory work. The matte is tapped off near its bottom and as it accumulates. The reverberatory settling furnace is used at two of the larger works and they are satisfied with the thoroughness of its action. It costs 10 to 14c. per ton to operate, and saves 40c. per ton more than by any other method as judged by the cleanness of the slag. There is some difference of opinion as to whether this represents the actual saving and whether there is not a volatilization loss unaccounted for. The process has been somewhat modified so that most of the matte is separated in front of the furnace by a settling pot and consequently there is but little to be removed from the separating furnace.

At two of the lead smelting works the problem of slag separation has been worked out on different lines. The idea is that the slag is to be run or poured into a large slag pot holding 3 or 4 tons. After the matte has been allowed to settle out by gravity the supernatant slag is tapped or poured off, and the remaining matte recovered. The shell is also saved and returned to the furnace. At the Pueblo Smelting and Refining Co.'s works this idea is thus carried out: A Mathewson matte tap effects the separation of most of the matte, which is tapped off as it accumulates. The slag flows into a 4-ton slag pot until the pot is full. The pot mounted on wheels is dragged out to the dump by horses and the matte is allowed to settle. The contents of the pot are carefully

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[*] *Transactions of American Institute of Mining Engineers*, Vol. XXV., pp. 396-397.

poured off. A lip is arranged at the pouring edge of the pot which holds back any matte which has settled out. The shell with this matte is returned to the furnace. The success of the method depends on having a furnace of 150 or 200 tons capacity per day, so that the slag will not chill before the pot becomes full.

At the Omaha & Grant works a portion of the slag is sold as ballast. This is made by pouring from the slag cars into large slag pots placed alongside the track. The slag is allowed to cool, dumped out, and broken up for rip-rap or for ballast.

Fluedust Collection.—For the roasting furnaces the only practicable way is to permit the dust to collect and the fumes to cool and condense in long flues, since the free sulphuric acid necessarily present would soon demolish bags. These flues should be of large area, for the slower and gentler the current the better the settling. The bag house is bound to be extensively used in the future. This appears to be particularly suited to closed top, hand-fed lead blast furnaces, where the furnace can be watched and overfire checked or prevented. Patents upon the system, which appeared to be prohibitive, have lately expired, competent expert advice can be obtained in putting them in, the initial cost of the building and bags is much less than has been heretofore thought necessary, precautions to insure durability and proper ventilation are understood, and in the case of a new plant an expensive stack and long flues can be saved. On the other hand the bags, especially of the cheaper cotton, do not last long, it is necessary to provide a hundred horse power, more or less, to operate the fans, and the fluedust has to be removed (an unhealthy occupation) since the fumes contain the arsenic volatilized from the charge. The temperature of the escaping gases from the blast furnace ranges between 200 and 300° F. In addition a large amount of cold air is drawn through the charge doors, so that the gases need but little cooling to bring them down to the 180° needed where cotton bags are used. With woolen bags a suitable temperature is 250° F. Dr. Bartlett prefers small bags to large ones. The large surface exposed to pressure in the Brown & De Camp bag house is a serious defect, and may have been a reason for not extending the system beyond Omaha. Curiously enough the use of bag houses has extended slowly. They have covered a multitude of sins in smelting, enabling the recovery of volatilized lead from overfire, from feeding scrap lead, from matte concentration, etc.

Further Treatment of the Base Bullion.—At the Philadelphia works particular pains are taken with the base bullion. The roughly dipped bars from the blast furnaces, say 60 to 70 tons per day, are remelted in a refinery market kettle; the drosses are skimmed off, and the clean metal molded into bars as at the refinery. The drosses are liquated on an inclined hearth reverberatory, and the well-drained dross returned to the copper blast furnace.

Copper Smelting at Pueblo.—At so large a works as the Philadephia the amount of copper produced from the lead ores, and purchased besides, justifies the operation of a special plant of which one or two blast furnaces are run always for copper matte. This matte is roasted and sent for treatment to a reverberatory which raises it to white metal of 68 to 72% copper. The white metal then goes to a concentrating furnace resembling an English cupelling furnace, being charged with some silicious ore and slag, and is blown on by tuyères, which oxidize the matte and work it up to blister, in which form it is shipped to Perth Amboy. The Monier flue system has proved very effective in cooling the fumes from the copper furnace; in fact it has cooled them so effectually as to impair the draft to the stack to which they are connected, and it has been thrown out of use for the present. The thin walls of this flue when cracked by unequal expansion due to variations of temperature also injures the draft.

The Bartlett low blast copper matting furnace at Cañon City presents itself to our consideration as a most remarkable variation from ordinary copper smelting work. It is a furnace working upon ores which would be the despair of an ordinary furnaceman. Its low height enables accretions to be barred off at a moment's notice without interfering with its action in any way. The tuyères are punched, opened or closed with facility, and as the furnace requires, and the furnaceman appears to have everything well under command. A change in the charge comes down in less than an hour, and at any point where extra fuel is needed it can be readily supplied, and is effective. It must not be forgotten, however, that a furnace which can be brought out of trouble so readily can get into trouble just as easily and must be closely watched. The furnace can be very quickly cleaned out, and is an inexpensive one to build. It is now used at the Globe works at Denver for concentrating the leady copper matte. Its intense oxidizing action enables it to concentrate a charge composed of raw matte, slag and silicious ore. The lead is of course driven off, but is again caught at the bag house. The lead-free matte is a desirable product to the refiner.

Labor.—The question of labor agitates the employer. It is becoming difficult to obtain it, and he is consequently driven increasingly to the use of automatic machinery, which demands higher-priced men. Such men are, however, more easily retained. It is not to be lost sight of, however, that the automatic handling of the ore to the furnace and the slag away from it may be carried to such a point as to interfere with the efficient operation of the furnace.

LIMESTONE.

Limestone is used industrially under various names and for numerous purposes. Massive limestone and dolomite are important structural materials and are employed extensively for road metal. The oölitic limestone of Bedford, Ind., is a highly valuable building stone. Limestone free, or nearly free, from magnesia is used in immense quantities as a flux in smelting iron, lead and copper ores. Another great use of non-magnesian limestone is in the production of lime and cement. Crystalline limestone or marble is also employed in lime burning, and as a building material it is more valuable than ordinary amorphous limestone. Certain special grades of marble, especially black marble and some variegated varieties, are highly prized, their value sometimes approaching that of onyx marble, which is a limestone deposited from solution in the water of springs in such a way that layers of different colors appear, producing an exceedingly beautiful stone for ornamental purposes. Another very valuable form of limestone is the kind known as lithographic stone, which for many years has been indispensable in lithography; recently sheets of aluminum have been found to be efficient substitutes. There are no accurate statistics showing the total production of limestone of all kinds in the United States, and probably it approaches the impossible to collect anything which would completely represent this industry, a part of the production being employed for purposes as to which no accounts are preserved.

Limestone for Flux.—According to the statistics collected by the American Iron and Steel Association the limestone consumed for fluxing purposes by the iron blast furnaces in the United States in 1898 was 5,275,819 long tons, an average of 0·448 per ton of pig iron, against 4,247,688, an average of 0·440 in 1897. Besides this production there is a large amount of limestone consumed by the silver-lead and silver-copper smelters of the United States.

Lithographic Stone.—There was a production of 112 tons of lithographic limestone in Utah in 1898. Some of this stone was sent to Chicago in the summer as an experiment, and later orders were received for regular shipments. It is believed that there is a future for this new industry. A deposit of lithographic stone, which is supposed to be extensive, was opened near Brandenburg, Ky. At this place there are three strata; one 34 in. thick and the others 12 in. thick each. The quality of the stone is said to be good. Arrangements were made for opening a quarry, and some production is expected from this source in 1899. There was no production in 1898 from the deposits of lithographic stone in South Dakota and Arizona.

The world's supply of lithographic stone is derived chiefly from Solenhofen in Bavaria, although there are two quarries in France. The stones from the latter, however, are said to be useless for high-class work. The stones from the Salsabella quarry in Bavaria are of the finest quality, being principally of the well-known gray color, and capable of production in large dimensions. Large stones are worth from $100 to $150 apiece.

A new discovery of lithographic stone was made in 1898 at Negoshevo, in the Balkans (Bulgaria), not far from Sofia. The deposits are owned by Popoff & Co. The Mikhalitch Lithographic Stone Co. was organized to operate quarries in Asia Minor. A discovery of lithographic stone was also reported on the Island of Mato, New Caledonia.

Onyx Marble.—This stone was produced in the United States in 1898 in Virginia, Colorado, Arizona and Kentucky. A quarry was opened near Pringle, Custer County, S. Dak., but no shipments were reported from that place. The Virginia Onyx Co. resumed work in its quarries at Rapp's Mill, on the headwaters of Buffalo Creek, and erected sawing and dressing machinery; shipments are made from Buchanan, 10 miles distant. The Colorado Onyx Co., of Steamboat Springs, Colo., which began work in 1896, made its first shipments in 1898, one carload going to New York and one to Chicago. These were largely in the nature of samples, the company being able to produce a good deal if a market can be found. Freight rates have been obtained which enable the product to be shipped to the Atlantic coast. A large deposit of stone has been uncovered, from which blocks of large size can be obtained. The stone is generally white, but is very beautiful. Shipments are made from Wolcott Station on the Denver & Rio Grande Railway.

The Onyx Deposits of Kentucky.—(By S. S. Gorby): New deposits of onyx marble have been discovered recently in this State in a zone extending from the central part of Green County through the vicinity of Knoblick in Metcalfe County, extending further west through Hart and Barren counties to the vicinity of Mammoth Cave in Edmondson County, the zone having a general southwesterly trend. Later researches resulted in the discovery of similar deposits in Warren County. The field also extends northward into La Rue County and eastward into Taylor, Adair and Russell counties, and southward into Wayne, Clinton, Cumberland and Monroe counties. Throughout this area the stone occurs in groups of "pockets," the "pockets" varying in size from a few square rods to many acres. The largest group so far known is situated in the northeast corner of Barren County, and embraces 25 or 30 sq. miles of territory, more than 150 distinct deposits being known in this group. These deposits sometimes occur over an area of several hundred acres, and sometimes are much smaller, there being one tract of 150 acres on which there are more than 20 distinct deposits.

The onyx deposits of Kentucky occur in the St. Louis limestone, at this place 700 to 800 ft. thick, which is one of the series of limetones near the top of the Subcarboniferous formation. The deposits of Barren County have a vertical range of about 200 ft. In texture the onyx of this county, and in fact the entire Kentucky region, is very fine, the stone having a satin-like gloss that is not equalled by the stone of any other locality. Except the green varieties formerly found in limited areas in Mexico nearly all colors may be obtained. A large proportion of the stone shows these colors alternating in unique lines and stripes, curving and zigzagging in every direction, and presenting a very beautiful appearance. A noteworthy feature of the field is the great size of the blocks that may be obtained, blocks as large 10×10×10 ft. being available. Instances are known near Cave City, Barren County, where slabs as large as 20×40 ft. area might be had if it were possible to quarry them of such large dimensions. So far the work in the field has been chiefly development, but at least 1,200 cu. ft. were shipped in 1898, the average value of the product being probably about $6 per cu. ft. at the quarry. One block, 6·5×4·5×3 ft., sold for $400 at the quarry, or about $4·50 per cu. ft.

THE BEDFORD OOLITIC LIMESTONE.

BY C. E. SIEBENTHAL.

THE Bedford oölitic limestone is one of the best-known building stones in the United States and enjoys a well-deserved popularity, due to its intrinsic worth as a handsome, easily-worked freestone of great durability.

Occurrence.—It occurs as a massive stratum, varying from a few feet to nearly a hundred feet in thickness, intercalated near the middle of the Subcarboniferous limestone of Indiana. Its labyrinthine outcrop has been traced in detail for a distance in a direct line of 60 miles through Owen, Monroe and Lawrence counties. To the south of this area extensive deposits of good stone are known to occur in Washington, Harrison and Crawford counties, and though the exact limits of these deposits have not been traced, the Bedford limestone undoubtedly continues from Lawrence County to the Ohio River.

Whatever be the thickness of the oölitic stratum it is with rare exceptions massive, the size of the blocks which may be quarried being limited only by the capacity of the quarry machinery and facility of transportation.

Texture.—The Bedford limestone is essentially a freestone, inasmuch as it works nearly equally well in all directions. This quality is due to its massive and semi-clastic character. In places the true oölitic structure discloses its organic origin, while in other places the absence of oölite and the presence of local cross-bedding betrays its clastic origin. Occurrences of oölite, however, are very local, and by far the greater part of the stone is oölitic, but not oölite. The size of the grains varies in different parts of the area and in different horizons at any one locality. The preference of the markets at present is for the finer grained, and the first quality must be a very fine homogeneous stone. The coarser grained, though fully as handsome and working just as well, is neglected in the best markets.

Color.—The original color of the Bedford stone in all cases varies from a light to a rather deep blue. On the outcrops and along the vertical clay seams where the stone has been exposed to the leaching action of terrestrial waters impregnated with the acids of organic decomposition, the color has been changed to a buff, and the bluer the stone was originally, the more pronounced the buff color of the altered part. The buff stone has the preference in the markets now, although a few years ago the reverse was true, and in a few years is likely to be so again. While the buff stone is necessarily restricted to that zone which has been subjected to leaching and is limited in quantity, the blue stone, on the contrary, when once reached, extends back as far as the ledge reaches, and is thus far greater in amount. It is to be presumed that the blue stone, not having been subject to leaching and weathering, is the stronger and more durable stone, though it is possible that the leaching process may result in a firmer cementation of the stone and thus have a contrary effect. No experiments bearing on this point have been made.

Crushing Strength.—Many compression tests have been made upon the Bedford stone in both the green and seasoned states, and with sawed and tool-dressed specimens. The results vary widely, as might be expected. A series of tests recently made under uniform conditions gave as an average 7,000 lb., with a maximum of 13,500 lb. per sq. in. for 2-in. sawed cubes. Formerly great crushing strength was the first desideratum in a building stone, but the universal use of the steel skeleton construction in the modern high buildings has so reduced the crushing strength requirement that it becomes a minor consideration, and greater stress is laid on homogeneity of texture, pleasing color and ease in working. Suffice to say that tests show the stone to be possessed of a very comfortable margin of strength over what is required in the lower masonry courses in the tallest buildings.

Specific Gravity.—The sp. gr. of the Bedford stone varies from 2·46 to 2·51. The average weight per cubic foot is very near 150 lb. for the seasoned stone. The green stone is somewhat heavier, and ranges from 175 to 180 lb.

Refractoriness.—A series of experiments was made on 1-in. cubes to determine the refractoriness of the Bedford limestone. Heating to 1000° F. and plunging in cold water failed to affect the cube. Heated to 1200° F. and plunged in cold water, the cube crumbled slightly on the lower edges. Heated to 1500° F. and cooled in the air, the cubes retained their form intact, but were calcined to a marked degree. This conclusively shows that the stone is uninjured by heat until calcination takes place. Instances are on record where buildings of the oölitic stone have been destroyed by fire and the stone of which they were built, after being scoured up, has been relaid in the walls.

Chemical Composition.—The Bedford stone is a nearly pure limestone of remarkably regular composition. Lime carbonate constitutes on the average 97% of the whole, varying in different analyses from 95 to 98%. It thus equals in point of purity the French Caen limestone and surpasses the English Portland oölitic limestone, the former of which contains 97·60% $CaCO_3$ and the latter 95·16. It is excelled in purity only by the purest marbles.

Durability.—The durability of the Bedford stone must be estimated from its purity and from its ability to stand sudden extremes of temperature, as shown by the experiments noted above, there being no structure built of this stone, so far as I know, which has an age greater than 60 or 70 years. As exhibited in buildings of that age, the effects of weathering and exposure are inconsiderable.

Method of Quarrying.—After the quarry site has been cleared by "stripping" off the earth, bastard limestone and irregularities of oölitic limestone and brought to a level surface, channels are cut with Wardwell, Ingersoll or Sullivan channelers to the depth desired, usually 6½ or 8½ ft. and of the distance apart desired. Having channeled the quarry one way, at some point convenient to the derrick two cross channels are cut from one channel to its neighbor and this "keyblock" is wedged loose from its bottom and lifted out with the derrick. Other blocks on either side of the "keyblock" are similarly removed until one "ledge-block" has been taken away from one side of the quarry to the other. The removal of the other "ledge-blocks" then becomes a matter of ease. By means of the steam drill holes are bored under the "ledge-block" at its base at distances of 8 or 12 in. from each other and to a depth of 3 or 4 in. Plugs and feathers are inserted in these and serially tightened until the ledge is freed on the bottom, when by means of the derrick it is pulled over on its side. Then it may be broken into the lengths desired by means of the steam drill and plugs and feathers. The depth on one ledge-block (6½ to 8½ ft.) is called a "channel cut," and the level bottom from which a tier of ledge-blocks has been removed is a "floor." The process is continued, channel cut by channel cut, or floor by floor, until the bottom of the ledge of stone is reached.

These rough blocks if they are to be sent to distant markets are "scabbled," that is, trued up and shaped to the proper dimensions by a sharp-pointed scabbling hammer. But if they are to be sawed and dressed at the quarry mill they are loaded on flat cars on the temporary railway switch which follows the active quarrying and taken to the mill. Along the whole front of the more perfectly appointed mills runs a traveling crane, beneath some part of which runs the railway switch. The blocks are carried by the traveler to the car of the saw-gang. Having been fixed on this car and the saw-blades adjusted, the cars are moved under the gang and the block ripped into ashlar of the required thickness. The slabs are then carried by the traveler to the planers and headers and there edged and jointed, and then carried by the traveler to the waiting flat car.

The freshly quarried or "green" stone contains a considerable quantity of sap or quarry water, and this in severe weather freezes and bursts the stone. This necessitates a closure of the quarries during the midwinter months, though some quarries operate continuously and protect the green stone from frost by dumping it into the water which generally fills the worked-out parts of the quarry.

Prices.—At the beginning of the industrial depression in 1893 the price of Bedford stone was 20c. per cu. ft. f. o. b. cars at the quarry. This price was maintained by general agreement until the autumn of 1895, since which time prices have fallen nearly one-half. The price of mill blocks to-day runs from 10c. to 20c. per cu. ft., with the larger amount of sales much nearer the former than the latter figure. The price for the last two seasons seems to have recovered to a certain extent, as an inspection of the table below will show that while the production for 1897 fell off slightly from 1896, the value of the product showed a gain over that year. This is in part, however, to be attributed to the greater amount of stone sawed at the home mills during 1897 and 1898. Almost one-half of the entire output for those years was worked up within the State, which is a much better showing than in any previous year.

Recent Developments.—One noteworthy feature of the development of the Bedford oölitic stone industry during the last two or three years has been the improvement, in point of number as well as in capacity and equipment, of the stone sawmills of the district. Almost all the mills have been enlarged in capacity and many new ones have been built. Among them is the largest mill in the district, which has just been erected by the Bedford Quarries Co., at Oölitic, Lawrence County. This mill is equipped with 12 saw-gangs, four planers, two headers, and twenty Knobel wire saws, the latter a new invention for scabbling and ripping large blocks.

The Indiana Stone Railway is the name of a short line of railway now under construction in Monroe County, which is intended to develop one of the most promising fields in the whole oölitic belt. Leaving the Chicago, Indianapolis & Louisville Railway at Clear Creek station, it follows the tortuous valley of Clear Creek, dodging from side to side to escape the bluffs, crossing it a dozen times and rejoining the main line of railway at Harrodsburg, some 9 miles from the point of departure. The quarry ledge outcrops back in the hills on each side of the main valley and of each valley branching off from it. The road will thus open up a linear extent of 50 miles of the ledge which has been hitherto inaccessible. Over this entire length the ledge varies between 40 and 45 ft. in thickness and is of first quality in point of texture and color. This remarkable uniformity in thickness demonstrates the persistence of the conditions of its deposition throughout the region and argues for a like uniformity and homogeneity of structure, texture and composition over the area. With proper exploitation the region is destined to become a very productive one.

PRODUCTION OF BEDFORD OOLITIC LIMESTONE.

	Monroe and Owen Counties. (*a*)		Lawrence County.		Total.	
	Cubic Feet.	Value.	Cubic Feet.	Value.	Cubic Feet.	Value.
1894...	2,176,246	$576,962	2,404,172	$577,284	4,580,418	$1,154,246
1895...	2,337,716	751,792	3,030,591	771,468	5,368,307	1,523,260
1896...	2,016,926	483,749	3,438,656	725,883	5,455,582	1,209,632
1897...	1,792,586	502,748	3,590,308	841,410	5,382,589	1,344,158
1898...	1,660,170	480,990	3,969,876	908,214	5,630,046	1,389,204

(*a*) Including the production of three mills in Indianapolis.

The total amount of capital invested in the industry is $2,463,400. There are 42 quarries, of which 20 were in operation in 1898, and 34 mills, of which 26 were in operation. The average working time in 1898 was nine months, and 1,614 men were employed.

LIME AND LIME BURNING.

BY HANS A. FRASCH.

CALCIUM oxide, or lime, which does not exist as such in nature, is produced mostly from the carbonate. It is obtained chemically pure by heating calcium nitrate to dark redness. Technically it is manufactured from limestone, calc-spar, marble, marl or sea shells, all more or less impure forms of calcium carbonate, by exposure to a temperature of 800 to 900° C. in a current of air, whereby the carbonic acid is expelled and caustic lime remains. The normal temperature for successful lime burning is said to be 850° C., but besides the temperature, proper conditions of draught have to be observed in order to obtain good results. Calcium carbonate, heated intensely, without admission of air, does not give up its carbonic acid, but fuses, and upon cooling congeals to a crystalline mass resembling sugar. Limestone with 43·4% carbonic acid, after exposure for four hours to a temperature of 615° to 675° C., has been shown experimentally to contain 43·4%; after three hours at 860° to 880° C., 40·6% was found; while after two hours at 850° C. in a current of air the lime was entirely free from carbonic acid.

Requisites of Limestone for Lime Burning.—Limestone, according to its purity, produces lime of different properties. White lime should not contain over 10% foreign matter, and should slake well with twice its volume of water, forming a fat paste, which will not harden under water, and in the air only in course of time. Gray, or short lime, slakes slowly and forms a short and often gritty paste. The presence of more than 10% of magnesia in the stone causes the lime to be short, and 25 to 30% renders the stone unfit for burning. Silica, free or in combination, has the same effect, the presence of 10% giving the lime hydraulic properties, *i.e.*, causing it to approach the nature of a cement.

Pure limestone, properly burnt, retains the shape of the pieces, loses 44% of its weight and increases in volume from one-fifth to one-tenth; it does not clinker, and slakes completely. The quicker such lime is burnt at the highest possible temperature the more readily it slakes, and consequently a slow-burning process is disadvantageous. However, limestone containing silica and alumina should not be burnt at too high a temperature, because the pieces will sinter on the outside, thereby preventing the escape of the carbonic acid, and lime so burnt is called "dead burnt lime," and does not slake completely. Dead burnt lime is formed especially if the impurities are evenly diffused through the stone. The ashes of the fuel and also the presence of alkali in the raw material (this is especially the case with shells) are likely to cause dead burning; even the best limestone may yield dead lime if heated too quickly and not long enough. In this case a basic calcium carbonate ($2\,CaO,CO_2$ or Ca_2CO_4) is produced, which with water forms a mixture of calcium carbonate and hydrate and hardens.

The harder and more compact the limestone the longer is the time required for burning properly, but the better the product. This is the reason why marble yields the best lime. The oldest formations contain the best limestone, yielding as much as 56% lime. For making mortar a lime with 28 to 30% of CaO is the lowest grade which is acceptable.

Principles of Lime Burning.—The burning of the limestone should be done under good draught. The time required for the burning is dependent upon the size and compactness of the stone and also its tenor of moisture. The latter helps greatly to carry off the carbonic acid and hastens the burning. It is claimed that in the presence of steam limestone can be burnt in one-eighth of the time required in a current of dry air and gases of combustion. This accounts for the fact that stone fresh from the quarry can be burnt much faster than stone which has lain in the air for a considerable length of time. Although it is to be assumed that the greater part of the moisture is expelled high in the kiln and before the carbonic acid commences to be driven off, nevertheless it seems to leave the stone in a more porous condition, allowing the carbonic acid to escape more freely from the interior of the lumps. Periodical injection of sprays of water or steam into the kiln is recommended. In kilns where the fuel is not charged with the rock, being heated by grate fires (flame kilns), it suffices to keep water in the ash pit.

The theoretical amount of lime obtainable from pure calcium carbonate is 56%. Limestone of 95% purity yields 59% of burnt lime, containing 90% CaO; 90% stone gives 62% of burnt lime with 80% CaO; 85% stone gives 65·5% with 72% CaO; and 80% stone gives 70% with 64% CaO.

For a large production modern apparatus for quarrying, crushing and conveying the limestone should be employed. The kilns should be situated at the quarry, and if possible they should be placed at a lower level so as to take advantage of gravity. The stone should be burnt soon after quarrying, since otherwise the extra time required to burn it will increase the expense both as to fuel and capacity.

The construction and manipulation of the kiln are the chief factors of cost of production, since they determine the quantity of fuel required. The latter depends upon whether during combustion the carbon is converted into carbon monoxide or carbonic acid, and the extent to which the heat developed is utilized. As will be shown subsequently, a large amount of heat may be wasted by imperfect draught or faulty firing. In kilns where the fuel is mixed with the charge care should be taken to distribute the fuel uniformly, and the individual charges should not be too large, so as to insure perfect combustion at all times. Uneven and too large charges are likely to cause imperfect burning of the limestone, and in continuous draught kilns "hang up" the kiln. This is to be avoided especially since it impairs the draught and causes uneven burning, which again results in "hanging up," and it may take days to recover normal conditions in the kiln.

*Heat Requirements.**—One kilogram of calcium carbonate requires 373·5 calories of heat for decomposition into lime and carbonic acid. The combustion of carbon to carbon dioxide (carbonic acid) evolves 8,080 calories per kg; hence if the combustion product is carbonic acid 1,000 kg. of limestone require 373,500÷8,080, *i.e.*, 46 kg. carbon. If, however, the combustion product is

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* Gruner, *Polytechnical Journal*, CCIV., p. 39.

carbon monoxide (as in the blast furnace) 1 kg. carbon evolves only 2,473 calories, so that 1,000 kg. of limestone require 373,500÷2,473, *i.e.*, 151 kg. carbon. Considering the heat which in both cases escapes with the combustion products, these having a temperature of 300° C., the carbonic acid contained in 1,000 kg. of calcium carbonate being 440 kg., it will carry off $300\times440\times0{\cdot}22$ (specific heat of carbonic acid), *i.e.*, 29,040 calories, to produce which will require 29,040÷8,080, or 3·60 kg. carbon, if the latter is converted into carbonic acid; and 29,040÷2,473, or 11,70 kg., if it is converted into carbon monoxide. Hence 1,000 kg. calcium carbonate require for decomposition, if the combustion product is carbonic acid, $46+3{\cdot}6=49{\cdot}6$ kg. carbon, and $151+11{\cdot}7=162{\cdot}7$ kg. if it is carbon monoxide. At least two-thirds of the heat effect are lost for every equivalent of carbon which escapes in a lime kiln as monoxide. In the application of limestone in the blast furnace this loss of course is unavoidable.

Gruner states that in the Hoffmann ring kiln 1,000 kg. of limestone required 70 kg. pure carbon, while an inferior coal was being used. Since 100 parts calcium carbonate require theoretically about five parts of carbon, this would show an efficiency of 71·4% of the fuel used. According to Thomson calcium carbonate requires technically in the kiln process about 12% of its weight of pure carbon.

Types of Kiln.—The style of kiln best to be adopted is governed chiefly by the kind of fuel at disposal and also the quantity of lime to be produced. Intermittent kilns are used when the consumption is irregular, while continuous kilns (draw kilns) are better adapted for constant and larger supply. The burning may be done in piles, ditches and horizontal or vertical kilns; the last may be constructed for intermittent or continuous operation, and for coal, oil, gas or wood firing.

Lime Burning in Heaps.—Lime burning in heaps or ditches is almost extinct, and is performed nowadays only in rural districts for local consumption. The heaps are built from 1,300 to 1,400 cu. ft. in size, being from 10 to 14 ft. high, and producing about 1,000 cu. ft. of lime. The stone is piled upon a grate of wood, the larger pieces being placed toward the center and for the laying out of draught channels, while toward the side and top pieces of smaller size are used. The fuel (coal or peat) is mixed with the stone and the whole is covered with fines or clay and provided with the necessary draught holes for the regulation of combustion, just as in charcoal burning. The amount of fuel required for this method is about 1·25 measure by volume of coal slack for 6 measures of limestone. This method was in use at a cement works at Argentine, Kan., as late as 1890, though, as might have been expected, the concern did not prosper.

Kilns, Horizontal and Vertical.—The horizontal kiln is constructed and operated in the same manner as an ordinary brick kiln. Vertical kilns are the most generally used. Next to the circular, or ring kiln, they get the highest efficiency out of the fuel; they are easily operated, and permit control of combustion and output.

"Dug-out Kiln."—The most primitive form of a vertical kiln is the old "dug-

out" kiln, which is built into the side of a hill or bluff, and constructed of ordinary stone and brick. It is shown in Figs. 1 and 2, in which *a* is the fireplace, *b* is an arch of large pieces of stone, supporting the charge, *d* is the front wall, and *c* are pieces of timber which after combustion form draught channels. This kiln is fired with wood, being first "smoked" or slowly heated to avoid cracking the supporting arch. The firing is then increased and continued until the stone appears white-hot and a bright flame, free of smoke, appears on top, when the fire is drawn and the kiln permitted to

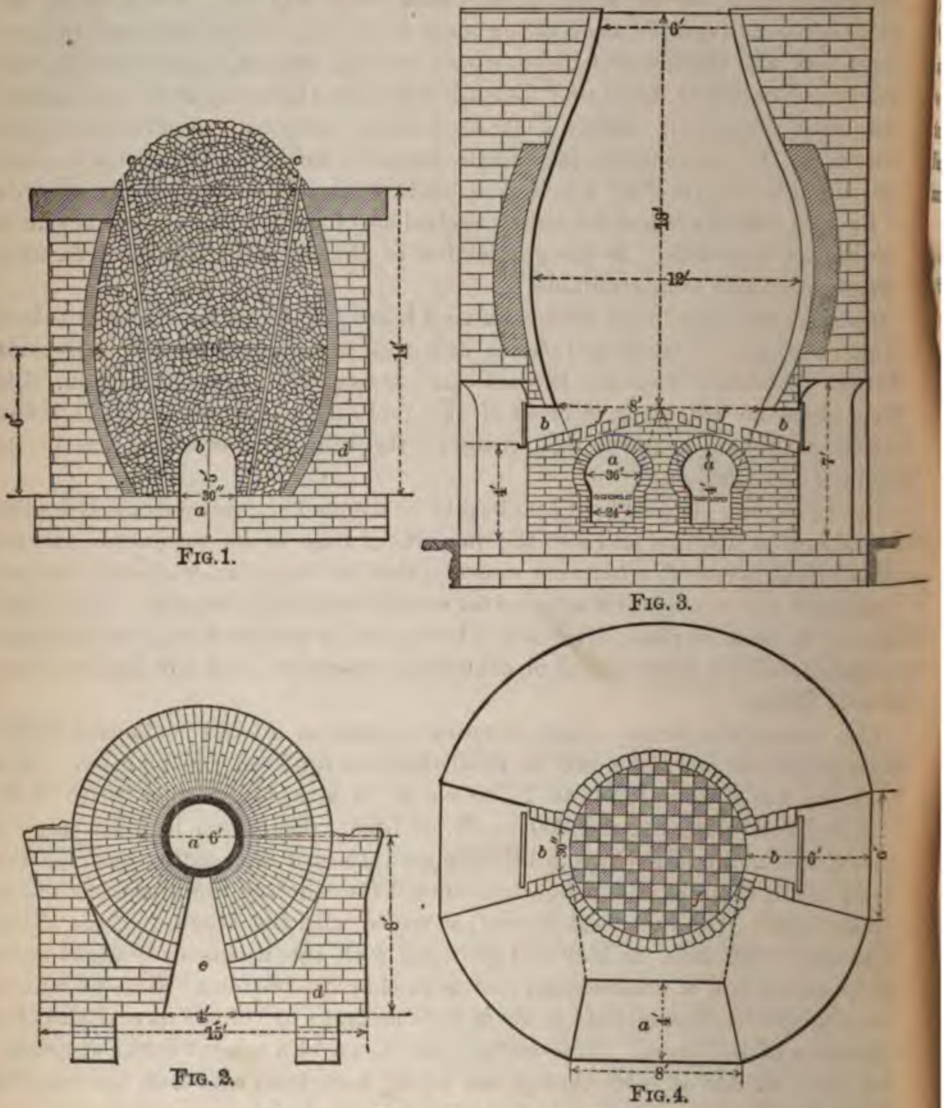

FIG. 1.

FIG. 2.

FIG. 3.

FIG. 4.

TYPES OF WOOD FIRED DRAW KILNS.

Figs. 1 and 2.—Vertical Section and Plan of a "Dug-out" Intermittent Kiln. Figs. 3 and 4.—Vertical Section and Plan of a Continuous Kiln.

cool for about 12 hours. The period of firing is about 72 hours. It requires for each volume of lime from two to three volumes of hard wood. For coal firing this kiln may be built with a grate.

Continuous Kiln for Wood Firing.—A kiln for continuous burning with long flame is shown in Figs. 3 and 4. This is of large capacity and useful in districts where wood can be obtained to better advantage than coal. It consumes about 50 lb. of pine wood per cubic foot of lime, and can be drawn four to six times per 24 hours. When freshly charged the first draw is made after three days' firing. The charges are from 400 to 500 cu. ft. of limestone. In Figs. 3 and 4 *a a* are fireplaces with perforated arch *f*; *b b* are the draw openings. These kilns are built 18 to 26 ft. in height and 10 to 12 ft. in width.

Kilns with the draw opening below the fire line are preferable to those above described. The firing can be done with the least possible amount of air, while the air passing through the hot lime cools that and is itself preheated and supports combustion better. Higher temperatures can be maintained with a smaller amount of fuel in this style of kiln than in those previously described. In Fig. 5 *a a* are fireplaces which may be arranged for wood or coal firing; *b* is the ash-pit provided with a water basin; *c* is a draw opening; *d* a chimney with draught regulator *e*; while *f* is the charge door. The chimney appears a superfluous appendage unless it is desired to collect and utilize the carbonic acid. A modification of such a kiln, which is in use near Austin, Texas, is shown in Fig. 6. It gives good results with 36 to 40 lb. of hard wood per cubic foot of lime.

Kilns without fireplaces, in which the fuel is charged together with the limestone, are the most popular. They are cheaper to construct, and give with a smaller consumption of fuel a larger output in less time. Though the lime is inferior in quality and color, it will answer for all ordinary purposes. For strictly white lime of best quality, however, they should not be used. For these kilns the rock should be of comparatively small size (6 to 8 in.), since the burning is done much quicker.

Rock of too large size will lead to imperfectly burnt lime or loss of time. In fact it should always be an object to have the pieces of rock of uniform size, and it is best to sort the rock and run for a time on one size and then on another. A certain proportion of small pieces can of course be used to fill out at all times, but if the disparity is too great the time consumed to burn the larger pieces is lost for whatever percentage of small rock is present, and the cost of lime burning may be greatly increased by the waste of time and fuel caused thereby.

Kind of Fuel Required.—In kilns where the rock comes in contact with the fuel, the lime is more or less affected by the ashes; if the latter are very alkaline and the limestone contains much silica it causes clinkers and results in short and perhaps dead burnt lime. The fuel should be rich in carbon and the less ashes it produces the better it is for the purpose. It should be compact enough to stand the weight of the limestone and of a form which will permit it to be spread easily. The best fuel for this class of kilns is hard coke, charcoal and anthracite; gas coke, lignite and soft coal are also used, but they are not so desirable. Rich bituminous coal is likely to affect the quality of the lime and obstruct the draft. Coal with much sulphur is detrimental. Lignite, if hard and compact enough, makes a fair fuel. Gas retort coke as a rule is too soft; otherwise it can be used advantageously, especially where it is desired to utilize the carbonic acid, as in the sugar and ammonia-soda industries.

Charging and Discharging.—The quantity of fuel to be charged in proportion to limestone is of course dependent upon the amount of carbon it contains. About one volume of soft coal, one and a half of coke, and three to four of lignite are required for three volumes of limestone. The kiln is charged with layers of stone 12 to 18 in. deep, according to the size of the stone and dimensions of kiln. A kiln of 1,200 to 1,400 cu. ft. available capacity will turn out from 700 to 900 cu. ft. of lime in 24 hours. Ten to 12 charges are drawn in a day. It is considered most advantageous to draw more frequently and operate on

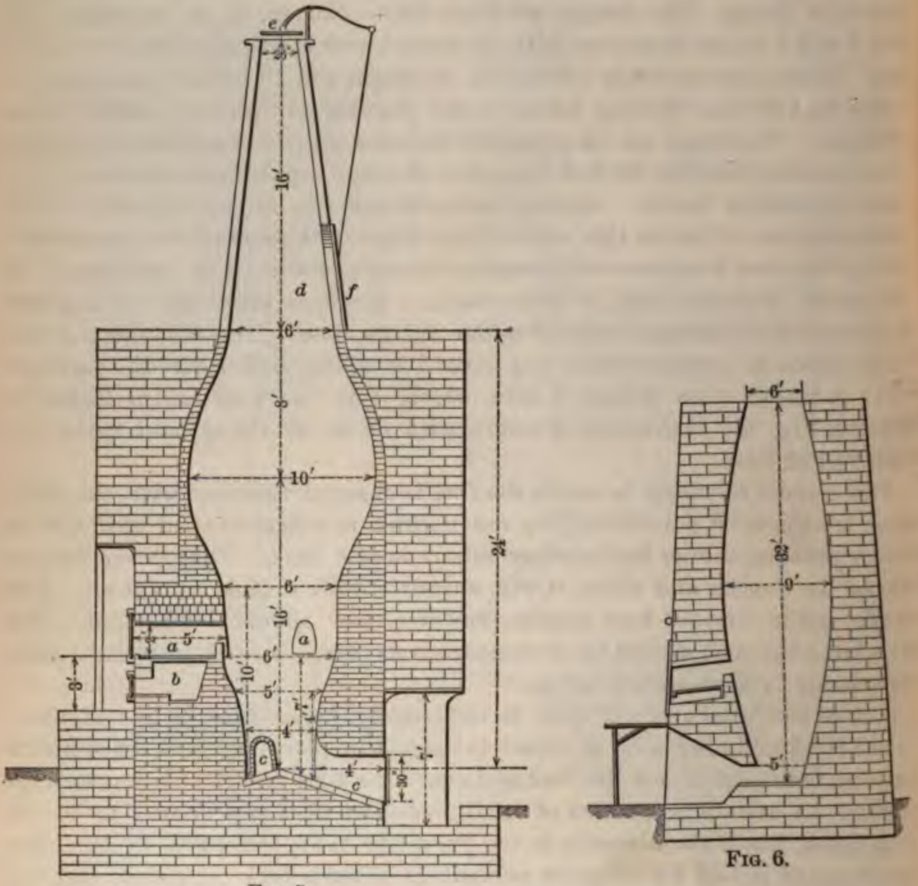

FIG. 6.

FIG. 5.

CONTINUOUS KILNS FOR WOOD FIRING.

Fig. 5.—Kiln with Draw-opening below the Fire Line, Chimney and Regulator. Fig. 6.—Modification of the type shown in Fig. 5.

smaller charges. A kiln of 800 cu. ft. capacity, 6 ft. in diameter at the top, and 10 to 12 ft. in the body, is charged with 3 ft. of limestone and 1 ft. of coal at a time.

The drawing of the lime at the proper time is of great importance. If it be drawn too slowly the fire advances too far toward the top and the kiln cools off, while if it be drawn too rapidly the lime is imperfectly burnt. Regularity in charging and drawing is the key to the successful operation of a kiln.

A very good way to observe the condition of a kiln and keep control of the burning is to test the exit gases for carbonic acid, which can be done very easily by tapping the kiln near the top and inserting a 0·5-in. pipe, through which the gas can be drawn by means of an aspirator or hand pump. The testing is generally done by means of the Winkler apparatus, which is very simple and easy to handle. Any ordinarily intelligent workman can manage it.

The percentage of carbonic acid contained in the gases of a lime kiln is very regular as long as the kiln itself is in good working order. Any change in the working of the kiln will at once show itself in the character of the gas. If the kiln is burned out too far the carbonic acid becomes less, as it does also if too much air is admitted. Immediately after a draw has been made, the amount of carbonic acid in the gases increases, until after a time it again becomes stationary and then commences to decline. When the decline of the carbonic acid is observed the kiln may be drawn again. Imperfect draught is generally observed by the smoking of the kiln. The percentage of carbonic acid in the gas of a lime kiln in good working order should range from 18 to 24.

Putting a Kiln in Operation.—To start the kiln it is filled with stone to about two-thirds of its height, when a layer of shavings and wood is introduced (about 3 ft.) and ignited; coal is then added and upon this a thin layer of limestone is placed. When the coal is burning well a larger charge is put on above the limestone, and the alternate charging with coal and stone is continued until the kiln is full. As soon as flame shows near the top the drawing is commenced, and the amount of coal per charge is reduced to the regulation quantity. By this method the first lime is obtained in about a week.

Another way to start one of these kilns is to fill it only about one-quarter (generally to the top of the bosh) with stone, having first placed a few pieces of fat pine wood perpendicularly in the center, around which the limestone is piled, for the purpose of communicating the fire from the draw-opening to the body of wood; then adding kindling wood and coal, and filling the kiln completely with alternate charges of stone and coal, the latter in somewhat heavier charges than ordinarily required, before igniting the pine wood at the draw-opening. In place of the pine wood, excelsior and tar rope may be used for ignition of the kindling. By this method lime is obtained within two or three days after the fire has been started. There is, however, the possibility that the kiln will miss fire if the conveying combustibles have not been placed properly and fail to lead the fire from the initial point (the draw-opening) to the body of wood. However, with proper care in the building of the bottom charge this will hardly ever happen.

To interrupt the burning process, once it has been started, the draw-opening is sealed by masonry and the top of the kiln is covered with fines. In doing this the draw-opening should first be closed before covering the top.

Design of Kilns.—The form given to these kilns is either conical or bottle shaped. The cone or funnel kiln shown in Figs. 7 and 8 is cheaply constructed and turns out about 250 cu. ft. of lime in 24 hours, consuming 20 to 25 lb. coal per cubic foot of lime. The larger size has generally three and the smaller two draw-openings. The kilns shown in Figs. 5 and 6 may also be used for draw kilns by omitting the fireplaces.

Iron-shell Draw Kilns.—Within late years a kiln with iron shell has come into use, and has given so much satisfaction that it is gradually displacing all other draw-kilns. It was first used in the United States in the ammonia-soda industry, and for this a good many of them are in operation near Detroit, Mich., and at Cleveland, Ohio. In cost of construction they are the cheapest, while they are easily repaired, are of long durability, and are operated with small cost. One of these kilns is shown in Fig. 9, in which *a* is a shell of heavy tank iron, which rests on brackets *b* and pillars *c*. The shell is lined

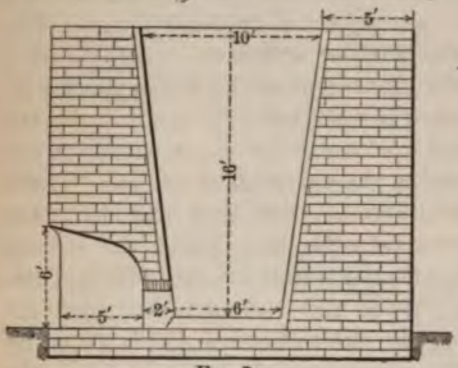

FIG. 7.

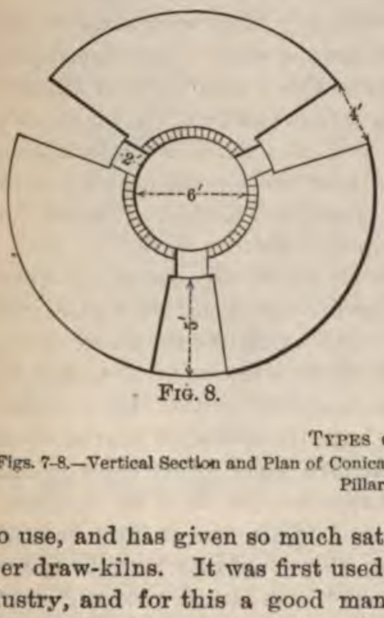

FIG. 8.

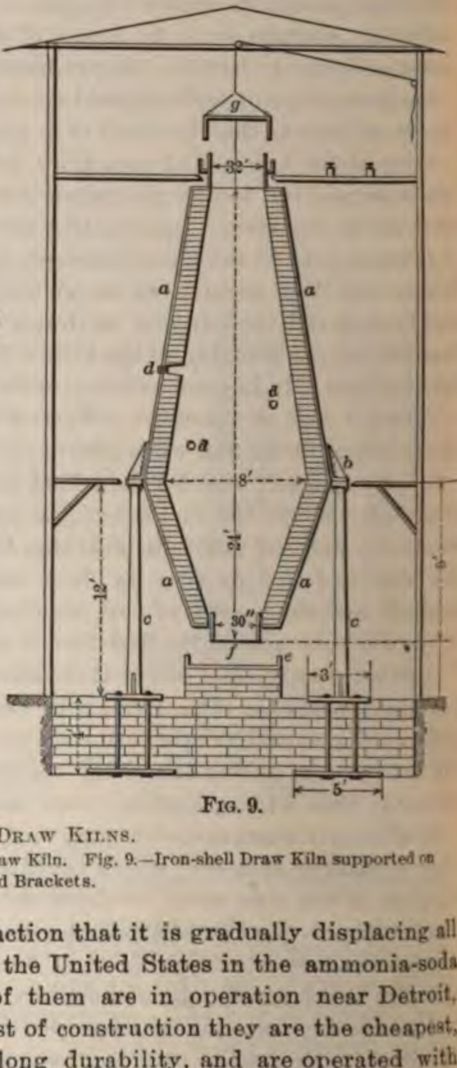

FIG. 9.

TYPES OF DRAW KILNS.
Figs. 7-8.—Vertical Section and Plan of Conical Draw Kiln. Fig. 9.—Iron-shell Draw Kiln supported on Pillars and Brackets.

with fire brick; in larger kilns a double row of brick is used; *d d* are 3-in. openings, which are closed by a threaded plug and serve to loosen the charge in case of clinkering, and also for observation of the burning process and admittance of air, if necessary, to any part of the kiln; *e* is a platform constructed of brick or stone, with iron top plate, on which the lime drops when drawing; *f f* are iron bars which form a grate, and can be removed when lime is to be drawn. In place of pillars the kiln may be supported by an arched stone foundation, as shown in Fig. 10, and surrounded by a second shell, the space between the two shells being filled with sand or ashes. The draw-opening in these kilns being in the center, the draft is very uniform and renders them independent of wind or weather.

For large production a series of kilns is generally arranged in a line with a tramway running along the top and an elevator at one end. The kilns may be inclosed in a frame shed, lined with sheet iron on the inside, the woodwork all being on the outside.

When the carbonic acid is to be utilized the kiln is supplied with a cover *g* and water or sand lock as shown in Fig. 9. In some instances steam is injected in one or more of the openings *d*, to hasten the process of burning the lime.

Petroleum and Gas Firing.—This type of kiln is also used for petroleum-firing. In this case openings are provided similar to *d*, Fig. 9, through which the fuel is injected. Oil kilns are of larger capacity than draw kilns, and should be preferable in locations where fuel oil can be obtained cheaply enough. The application of the oil is accomplished by injectors, several of which are in the market. Four burners are used, all being placed on one level about 1 ft. above the widest part of the kiln.

Kilns with gas fuel are used in the sugar industry, where it is required to obtain clean carbonic acid gas and pure lime. They are expensive to construct, but have the advantage that coal of inferior quality can be used. On the other hand, if the gas is introduced horizontally, the flame does not penetrate far enough toward the center of the kiln, and they have therefore to be constructed of smaller diameter and capacity. To avoid this difficulty attempts have been made to construct gas kilns with a perpendicular gas conduit extending from the bottom of the kiln into the interior, but it was found that the cone distributing the gas was soon destroyed. Altogether the vertical gas kiln has yet to establish itself, and even the advantages to be gained by it are doubtful.

Hoffmann Kiln.—The horizontal circular kiln first introduced by Hoffmann, which has proved so satisfactory in brick burning that in Europe it has practically revolutionized that business, is also used for lime burning. One of its main advantages is that brick and lime can be burned at the same time, and the quantity of either can be regulated according to market requirements. I have seen a Hoffmann kiln charged with drainage pipes, terra cotta, lime and brick, all at the same time. This kiln is fired with coal fines; even coal dust can be used, providing the percentage of ash is not too high. Of all kilns it shows the largest efficiency of heat and requires the smallest amount of fuel. In England such a kiln produces daily 1,200 to 1,500 cu. ft. of lime, with a consumption of only 5 lb. of coal slack per cubic foot of lime. In Germany, where the cost of fuel per cubic meter of lime, by the old kiln process, used to be 8·7 marks, it is now produced in the Hoffmann kiln with a fuel expense of 2·3 marks per cubic meter. This shows a fuel consumption between the two systems as 1 : 3·8 in favor of the Hoffmann kiln.

The lime produced by the circular kiln is lighter by about one-tenth than other lime. A volume of draw kiln lime weighing 55 lb. will weigh only 50 lb. if burnt in the Hoffmann kiln, and the same volume of draw-kiln lime

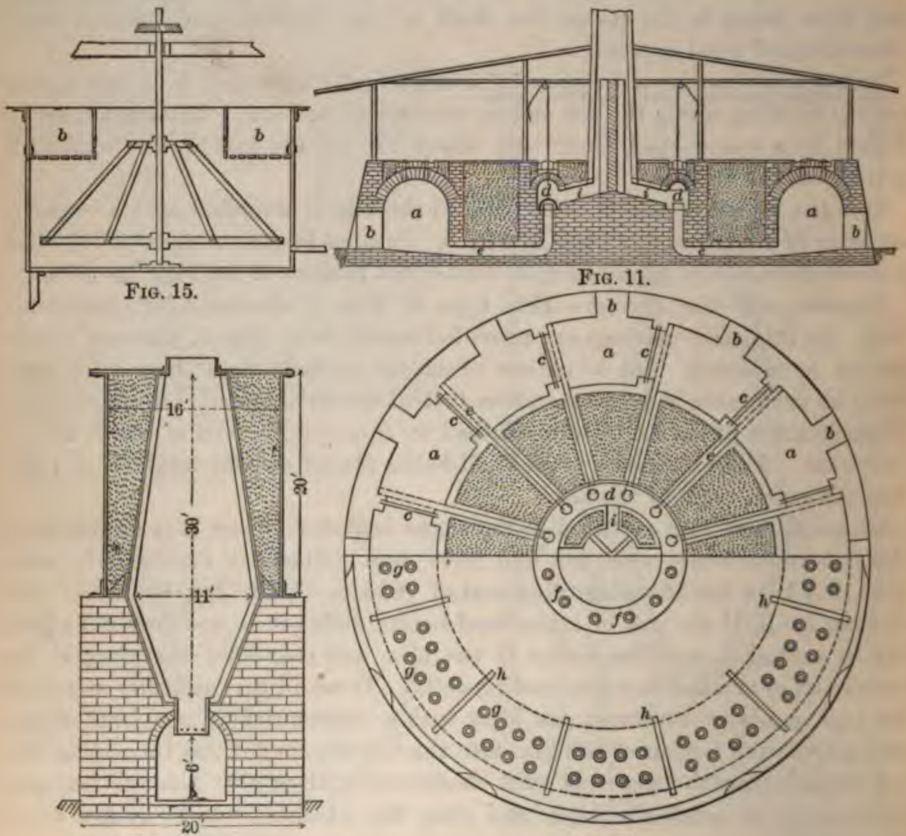

FIG. 15. FIG. 11.

FIG. 10. FIG. 12.

DETAILS OF LIME KILNS AND APPARATUS.

Fig. 10.—Iron-shell Draw Kiln supported by Stone Arch. Figs. 11 and 12.—Hoffmann Lime and Brick Kiln. Fig. 15.—Milk-of-lime Apparatus.

giving 12 measures of slaked lime will give 18 measures if the lime has been burnt in circular kilns.

The Hoffmann kiln requires, however, more labor to charge and build up the draught channels; it requires uniform material, and the lime is more friable, slakes more easily, and cannot be stored as well as other lime, especially if the air has been damp during the period of burning and cooling. It demands, therefore, a large and immediate consumption. In the vicinity of large cities and places where it can be used at the same time for brick burning it is certainly the most profitable kiln.

The Hoffmann kiln consists of an arched, annular room (which may also be elliptical or rectangular) divided by movable partitions into sections, of which there are generally twelve. The limestone is built up in this room, provision being made for draught and fire openings. The firing is done through numerous openings in the top of the kiln, which communicate through vertical tubes with the interior. These openings also serve for observation of the advance of the burning. The fuel is only fed into that section of the kiln which is in full heat, and the firing is advanced when the lime in a section is completely burnt, thus constantly traveling around the circle.

The air supporting combustion enters the section which is being emptied, about four to five compartments ahead of the full fire, and in passing the burnt lime it becomes heated while at the same time it cools the burnt lime, until it reaches that part of the kiln which is under full fire. It then passes on through the later-charged sections, giving off its heat to the limestone until it leaves the kiln quite cool from the last and freshly charged section and passes into the chimney located in the center of the kiln. The arrangement is shown in Figs. 11 to 14, in which *a* is the channel accessible through openings *b b*, and divided into sections by dampers *c c; d* is the main flue communicating with the kiln through smaller flues *e* and with the chimney through flues *i*. The bells *f f* permit connection or disconnection of either of the flues *e* with the chimney. The fuel is charged through openings *g g*, which are hermetically sealed by bells in a sand lock; *h h* are openings for the dampers *c c*, and are sealed with clay. When the damper is introduced, as shown in

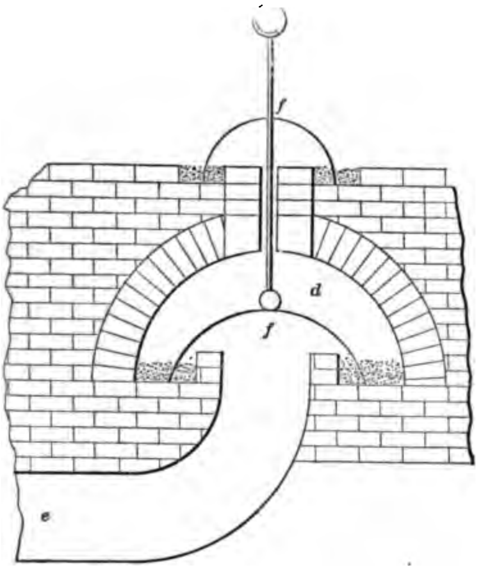

FIG. 13.

FIG. 14.

DETAILS OF HOFFMANN LIME AND BRICK KILN.

Fig. 14, the charge-door in front of it being open while all the others are closed, and flue *e* immediately behind it being open and all other flues *e* being closed, the air will enter the kiln through the charge-door *a*, and pass through the whole length of the kiln and the open flue *e* to the chimney. If the kiln is charged and in operation, the firing being done at No. 5, sections 4, 3 and 2 are filled with burnt lime, 1 is being emptied and recharged, 5 is being burnt, and 6 to 12 contain limestone at different degrees of heat. The air passing through 1, 2, 3 and 4 becomes intensely heated, so that when it reaches the combustion section (5), in which the lime has already been raised to a high red heat by previous exit gases, it takes but a short time and very little fuel to raise the lime to white heat. The extremely hot gases from section 5 heat the limestone in the following sections, 6 to 12, and gradually in their passage through these sections they give off their heat and leave the kiln through flue *e*, of section 12, which has last been filled with fresh stone, comparatively cool.

When section 1 is empty it is refilled with limestone, and the damper *c* of section 2 is inserted, the opening *b* of section 2 opened and closed at 1 by masonry, the flue *e* of section 1 is then opened, the damper from section 12 removed, and flue *e* of section 12 closed, so that the gases now enter at section 2 and pass through the kiln to 12 and 1, leaving the kiln at section 1. The firing then can be advanced to section 6. The draught can be regulated by raising or lowering the bell *f*. When first starting these kilns it is better for a while to burn brick, which do not require as high a temperature as lime, until the kiln is thoroughly heated. The limestone in circular kilns shrinks from 30 to 35% of its height during the burning process. Compared with the best constructed draw kilns with direct firing, the general cost of burning lime is stated to be on an average as 8 to 5 in favor of the Hoffmann kiln.

The circular gas kiln, a modification of the Hoffmann kiln, has so far not shown itself superior, except when in connection with by-product coke ovens. The expense for fuel is higher. Besides, it has all the faults of the draw kiln fired with gas, costs more to construct and repair, and is hard to regulate.

Properties of Burnt Lime.—Properly burnt pure lime slakes to a fine powder with one-third of its weight of water, forming calcium hydrate, or hydroxide (CaH_2O_2), which consists of 75·67% CaO and 24·33% H_2O, having a sp. gr. of 2·078. The lime first absorbs the water, and then binds it chemically with liberation of heat, which may reach 150° C., so that it will char wood. Large quantities of lime accidentally wet with water may set fire to wood or straw, with which it may chance to be in contact. The practice of storing lime in buildings containing wood or hay is therefore to be condemned, and the storage of lime packed in barrels is especially dangerous.

Lime slakes with more or less energy according to its nature, the mode of burning, and its age. The density of the lime also greatly affects its slaking; when burnt under 1,000° C. it slakes at once, while if burnt at white heat it may take hours before reaction sets in. With steam it slakes but slowly, but hot water accelerates the process. When no warm water is to be had the lime should first be soaked with about one-third of the required amount of water until it swells and develops enough heat. Insufficiency of water does not open the lime properly; it becomes gritty and loses in binding capacity, becoming "killed" lime; the best lime may be spoiled by improper slaking. The better a lime is the more water it will bind. For lime powder, such as is used in sugar refineries, for instance, 32 parts of water are required for 100 parts of lime; one-third of the water should be used to swell the lime before adding the rest. Lime slaked to powder in this manner can be kept for years and yet used for mortar if stored in piles covered with wood ashes.

For mixing mortar 4 to 5 volumes of lime are swelled in about one and a half volumes of water, and then the rest of the water is gradually added until the desired consistency is obtained. Two and a half to three parts water are generally required.

The yield or growing of the lime also depends greatly upon the mode of slaking. If the total amount of water is used at once during the slaking operation the increase of volume may be three and a half times that of the lime; if, however, the lime is slaked short, first immersed in water and permitted to fall to powder before the rest of the water is added, it will only yield two and a half times its volume, and if the lime is permitted to slake in the air it will increase only 1·7 times in volume.

Burnt lime in pieces will absorb in dry air but little carbonic acid; in damp air it absorbs it rather rapidly, falling to a sharp gritty powder of hydrated basic carbonate of calcium, which is unfit for mortar mixing. Slaked lime absorbs carbonic acid much more readily. The affinity of caustic lime for carbonic acid is claimed to be the highest at red heat in an atmosphere of carbonic acid, and 27 times greater than at a temperature of 100° C. Hence the requirement of draught in a lime kiln, which is an essential that is not realized by many lime burners.

Powdered lime, well compressed in an iron pipe containing minute perforations, exposed to an atmosphere of steam and then heated to full redness for several hours in presence of carbonic acid, will form, if permitted to cool in the carbonic acid gas, a hard core of carbonate of lime resembling marble. It is stated that artificial marble is produced by a similar method of heating lime, compressed to desired forms, in specially constructed furnaces, to a temperature exceeding 1,000° C. in an atmosphere of carbonic acid, and permitting it slowly to cool in presence of the gas.

Slaked lime, if kept under water, is very stable and can be preserved thus for a long time. The hydrate gives off its water only at red heat (about 500° C.).

Use of Lime in Sugar Refining.—Lime is soluble in about 780 parts of water at 15° C. and in 1,500 parts at 100° C. Alcohol only dissolves a trace. Sugar water dissolves it in larger quantity, whereby the lime enters into chemical combination with the sugar, forming sugar lime. This fact is technically employed for the separation of sugar from molasses. The molasses is treated with lime and the resulting sugar lime is decomposed by the action of carbonic acid, forming calcium carbonate and pure sugar. Strontium, however, has lately displaced the lime in this process. One hundred parts of cane sugar dissolved in water will dissolve from 50 to 55 parts of lime.

Preparation of Milk of Lime.—The slaking of the lime is mostly done in wooden boxes. For industrial purposes, however, where the lime is used in the form of milk of lime, as in the distillation of ammonia, for instance, mechanical devices are employed for this purpose. An apparatus is shown in Fig. 15 which receives the lime in the suspended basket *b*, from which the unslaked portions can be removed at leisure. For larger production a revolving perforated cylinder, placed horizontally in a receptacle containing warm water, is used. The cylinder is placed at a slight inclination, so that unburnt stone, etc., will collect at the lower part, whence it can be removed through a suitable opening. This apparatus does very effective work. The following table shows the amount of lime present in milk of lime of different density:

Degrees Baumé.	One Liter weighs Grams.	One Liter Contains.		Per Cent. Lime.	Degrees Baumé.	One Liter weighs Grams.	One Liter Contains.		Per Cent. Lime.
		Lime. Grams.	Water. Grams.				Lime. Grams.	Water. Grams.	
10	1258·6	133·3	1125·3	10·60	38	1497·8	294·9	1202·9	19·73
11	1274·0	142·5	1131·5	11·12	39	1498·6	296·5	1202·1	19·80
12	1292·0	151·9	1140·1	11·65	40	1499·0	297·9	1201·1	19·88
13	1308·0	161·3	1146·7	12·16	41	1499·6	299·3	1200·3	19·95
14	1336·0	170·3	1155·7	12·68	42	1500·0	300·6	1199 4	20·03
15	1345·0	179·7	1165·3	13·20	43	1500·4	301·7	1198·7	20·10
16	1363·0	188·7	1174·3	13·72	44	1500·8	302·9	1197·9	20·16
17	1382·0	197·9	1184·3	14·25	45	1501·2	303·7	1197·5	20·22
18	1399·0	207·3	1191·7	14·77	46	1501·4	304·7	1196·8	20·27
19	1417·0	215·9	1201·1	15·23	47	1501·9	305·7	1196·5	20·32
20	1436·0	224·3	1211·7	15·68	48	1502·4	306·6	1195·8	20·37
21	1451·0	233·3	1217·7	16·10	49	1502·8	307·5	1195 3	20·43
22	1462·0	240·1	1221·9	16·52	50	1503·2	308·3	1194·9	20·48
23	1469·0	246·9	1222·1	16·90	51	1503·5	309·1	1194·4	20·52
24	1474·0	252·9	1221·1	17·23	52	1503·7	309·9	1193·8	20·57
25	1478·0	258·3	1219·7	17·52	53	1504·1	310·7	1193·4	20·62
26	1481·0	262·6	1218·4	17·78	54	1504·3	311·3	1193·0	20·66
27	1484·0	266·7	1217·3	18·04	55	1504·6	311·9	1192·7	20·70
28	1486·0	270·1	1215·9	18·26	56	1504·9	312·6	1192·3	20 74
29	1488·0	273·7	1214·3	18·46	57	1505·2	313·3	1191·9	20·78
30	1490·0	276·7	1213·8	18·67	58	1505·4	313·7	1191·7	20·82
31	1491·2	279·5	1211·7	18·86	59	1505·6	314·1	1191·4	20·85
32	1492·4	281·9	1200·5	19·02	60	1505·8	314·7	1191·1	20·89
33	1493·4	284·5	1208·9	19·17	61	1506·0	315·3	1190·7	20·93
34	1494·4	268·8	1207·6	19·31	62	1506·3	315·7	1190·6	20·97
35	1495·4	289·1	1206·3	19·43	63	1506·5	316·3	1190·2	21·00
36	1496·4	291·3	1205·1	19·53	64	1506·7	316·8	1190·0	21·03
37	1497·0	293·3	1203·7	19·63	65	1507·0	317·2	1189·8	21·05

Other Uses of Lime.—The other applications of lime are manifold. By far the largest amount is used for building purposes. Besides this, it is used extensively in the manufacture of artificial stone, in the manufacture of glass and as an agent in many chemical processes.

According to H. le Chatelier in *Monit. Ceramique*, Vol. XXIX., Nos. 8, 10 and 11 the slaking of lime in practice is divided into four stages. (1) Simple absorption of water; (2) the mixture is then shoveled against the face of the main heap, is warmed in contact and also evolves heat by the chemical action going on, a portion of the added water being thus evaporated; (3) the mass cools and moisture is fixed by the silicates, although some of the free lime still remains unslaked; (4) the unslaked lime removes this water from the silicates and becomes completely hydrated. These necessary stages are frequently curtailed in practice to the detriment of the product, a sufficient time being essential to enable hydraulic lime of good composition to slake properly; that which becomes quickly and thoroughly hydrated is sure to be of poor quality.

MAGNESIUM AND MAGNESITE.

THE production of magnesite in the United States continues to be restricted to California, where there are two companies engaged in the business. Their market is practically limited to the Pacific Coast, the requirements of which are comparatively small, wherefore there is not much extension in the industry from year to year. The Atlantic Coast and Middle States, which consume a large amount of magnesite, are supplied from Europe, chiefly by Greece and Styria. In 1898 the United States imported 16,039 short tons of magnesite (crude or calcined), valued at $134,130. The importation of magnesite brick is not enumerated in the customs reports. There was in 1898 an increased demand for magnesite from the steel-makers, who use it as a furnace lining.

PRODUCTION OF MAGNESITE IN CALIFORNIA.

Year.	Crude.			Calcined.		Crude Equivalent of Calcined. Short Tons.	Total Crude.	
	Short Tons.	Value at Mine.	Per Ton.	Short Tons.	Value at Works.		Short Tons.	Value.
1894	450	$2,700	$6·00	460	$6,500	920	1,370	$8,220
1895	520	2,600	5·00	840	12,180	1,680	2,200	11,000
1896	489	2,300	4·70	789	11,135	1,578	2,067	9,715
1897	379	1,516	4·00	764	10,700	1,528	1,907	7,628
1898	200	800	4·00	1,013	14,200	2,155	2,355	9,420

In consequence of the increased demand for Grecian magnesite exploration for this mineral has been stimulated in Greece, and in 1898 new mines were opened near Chalcis, North Eubœa, and also on the Grecian mainland. The production of crude magnesite in Greece in 1898 was 14,829 metric tons, calcined magnesite 129, and magnesite brick 516, the figures for 1897 being respectively 11,311, 686 and 826.

The unworked deposits of magnesite in various parts of the world are believed to be large. Large beds of pure mineral are said to exist in the southern part of New Caledonia. In the Chalk Hills of Salem, in the Presidency of Madras, there are two areas containing veins of magnesite, one of them 1·25 miles square and the other 3·5. The veins are irregular, but according to the Indian Geological Survey they are numerous, and the resources in this mineral are great. At present it is mined only on a small scale, the high cost of transportation making it difficult to compete with Greece and Styria. However, a small amount is carted to Moganoor on the Cauvery River, about 40 miles south of Salem, whence it is boated to Porto Novo and shipped to Europe by the Porto Novo Iron Co.

MAGNESIUM.

Magnesium used to be produced in the United States, but the business was abandoned years ago, and at present the world's supply is furnished by the Magnesium Metal Co., of Patricroft, Manchester, England, this company being controlled by Johnson, Matthey & Co. and the Aluminium und Magnesium Fabrik, of Hemelingen bei Bremen, Germany, which is the only producer of magnesium on the Continent. The latter company works under a patented process, but notwithstanding its practical monopoly has been unable to declare dividends, owing to the small consumption of the metal. Its production is about 12 metric tons per annum, which is consumed chiefly by nickel works, which use it in refining nickel under the patents of the Westfaelisch Nickelwalzwerke of Schnerte (Westphalia). Nickel becomes malleable and assumes a beautiful soft appearance when magnesium has been introduced into the molten metal, and it is claimed that the quality of nickel-steel can be improved by the use of nickel refined with magnesium. Magnesium ribbon is made for use in magnesium lamps, of which various kinds are employed for marine and military purposes. These lamps are also used in paper mills, etc., where they are said to have the advantage of showing colors better than any other light. Magnesium powder is made for use in the arsenals of Germany and England, and also in photography and pyrotechny.

The price of ingot magnesium has been reduced from 30 marks to 22 marks per kg., but sales do not increase. The product assays about 99·75% Mg. The imports of magnesium into the United States in 1898 were valued at $12,533.

E. Hilberg, of Berlin, patented a new process for the production of magnesium from carnallite, wherein the mineral is melted, covered with asbestos, and an electric current passed through it. As soon as a smell of chlorine indicates that the decomposition has begun a vacuum is created in the decomposing vessel, thus avoiding the formation of magnesium oxide.*

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* English Patent No. 16,659, July 30, 1898.

MANGANESE.

The production of manganese ore in the United States is summarized in the subjoined table, which includes the production of psilomelane and pyrolusite, high-grade manganese-iron ore, manganiferous iron ore with 6 to 8% Mn, such as is produced in Michigan and Wisconsin, and the franklinite residuum of the New Jersey and Pennsylvania zinc oxide works, which contains about 14% Mn. The argentiferous manganese-iron ore of Colorado and other Western States, which is used as a flux in silver-lead smelting, is not included, although account is taken of that part of this product which is bought by the steel-makers for smelting to ferromanganese and spiegeleisen. Of the production reported below nearly all is used in making ferromanganese and spiegeleisen, chemical manufacturers consuming a comparatively small amount, high-grade pyrolusite being used by them.

STATISTICS OF MANGANESE ORE IN THE UNITED STATES. (IN LONG TONS.)

Year.	Production.											Imports.		Consumption.	
	Arkansas.	California.	Colorado	Georgia.	Michigan and Wisconsin.	New Jersey.	Tennessee.	Virginia.	Elsewh're	Totals.		Tons.	Value.	Tons.	Value.
										Tons.	Value.				
					a	*b*									
1894	3,000	250	5,000	1,000	198,466	27,667	921	1,438	*c* 66	237.808	$484,332	44,655	$432,561	282,463	$916,893
1895	2,500	880	7,968	2,614	112,265	42,726	......	1,556	......	170.509	328,107	86,111	747,910	256,620	1,076,017
1896	3,038	318	9,072	2,538	110.317	35,655	......	1,588	......	162.526	339,083	31,489	250,468	194,015	589,551
1897	4,430	450	18.600	962	80,261	50,000	190	2,408	*d*1,300	158,600	328,176	39,574	340,945	198,174	669,121
1898	2,775	393	17,792	2,477	142,318	47,470	1,250	3,307	......	217,782	481,127	14,885	831,967	332,667	1,313,094

(*a*) Manganiferous iron ore. (*b*) Franklinite residuum. (*c*) West Virginia. (*d*) Vermont manganiferous iron ore.

The price of manganese ore delivered at Bessemer, Pa., is determined by the Carnegie Steel Co. according to the following schedule, which is based on ores containing less than 8% silica and 0·1% phosphorus. For each 1% of silica in excess of 8% a deduction of 15c. per ton is made, and there is also a deduction of 1c. per unit of manganese for each 0·02% of phosphorus in excess of 0·01%. Settlements are based on samples dried at 212°F.

Tenor in Mn.	Price Per Unit.		Tenor in Mn.	Price Per Unit.	
Per Cent.	Fe.	Mn.	Per Cent.	Fe.	Mn.
Over 49	6c.	28c.	37 to 40	6c.	24c.
46 to 49	6c.	27c.	34 to 37	6c.	23c.
43 to 46	6c.	26c.	31 to 34	6c.	22c.
40 to 43	6c.	25c.			

WORLD'S PRODUCTION OF MANGANESE ORE. (*a*) (IN METRIC TONS.)

Year.	Austria-Hungary.	Belgium.	Bosnia. (*b*)	Brazil. (*e*)	Canada	Chile. (*d*)	Colombia.	Cuba.	France.	Germany	Greece.	India.
1893....	6,649	16,820	7,403	*Nil.*	193	36,741	....	18,922	38,080	40,798	5,250	....
1894....	8,803	22,048	6,588	1,390	67	47,994	....	(*c*)	32,751	43,702	9,319	11,583
1895....	7,733	22,478	8,145	5,490	113	24,075	6,025	(*c*)	30,871	41,327	7,250	16,079
1896....	(*c*)	23,265	6,821	14,120	112	26,152	10,668	(*c*)	31,318	45,062	15,500	57,783
1897....	(*c*)	28,372	5,344	8,800	14	23,528	....	(*c*)	37,212	46,427	11,868	74,862

Year.	Italy.	Japan.	New Zealand.	Portugal.	Queensland.	Russia.	South Australia (*d*)	Spain.	Sweden.	United Kingdom.	United States.
1893....	810·0	14,170	324	....	....	268,621	2,467	1,460	7,061	1,357	152,806
1894....	760·0	13,368	543	....	142	243,469	176	340	3,359	1,838	241,613
1895....	1,569·0	17,141	213	1,240	361	163,770	49	10,162	3,117	1,293	173,227
1896....	1,890·5	17,967	66	1,494	305	(*c*)	*Nil.*	38,265	2,056	1,097	165,155
1897....	1,634·0	(*c*)	182	1,652	403	(*c*)	*Nil.*	(*c*)	(*c*)	609	161,138

(*a*) From official statistics, except for the United States and Colombia, for which direct reports have been received from the producers. (*b*) Includes Herzegovina. (*c*) Statistics not yet published. (*d*) Export returns. (*e*) Shipments as stated in British Foreign Office Report, Annual Series, No. 2058, April, 1898.

Brazil.—Manganese ore has been exported from Brazil since 1893, the shipments by Airosa & Co. and C. Wigg, who exploit the mines, having been as follows: 1894, 1,390 tons; 1895, 5,490; 1896, 14,120; 1897, 8,800. The shipments in 1894–96 went to Middlesborough, England; in 1897 to Philadelphia, Pa. The ore comes from the district of Miguel Burnier, in the State of Minas Geraes, which is connected by the Central Railway with Rio de Janeiro, a distance of about 310 miles. The mines are situated at an altitude of 4,000 ft. in a pleasant and healthy climate. The labor employed is chiefly Italian and native, with some Spanish and Portuguese. A cargo of Brazilian manganese ore assayed by E. Riley, of London, March 31, 1897, showed 54·7% Mn, 0·077 P_2O_5, 0·027 SO_3, 3·4 Fe_2O_3, 0·83 SiO_2, 1·21 Al_2O_3, 0·21 MgO, 2·65 BaO, 0·96 K_2O, 0·05 Sb_2O_3, 0·77 CO_2, 4·62 H_2O, the remainder being oxygen combined with the manganese as peroxide and protoxide.

According to M. Ribeiro Lisboa, manganese ore deposits are found in the States of Mattogrosso and S. Paulo, besides Minas Geraes, but the last are by far the most important, not only on account of their extent, but also the purity and richness of their ore, the tenor in manganese averaging 50%. The principal deposits are found in small veins in the micaceous and argillaceous schists, as at Queluz, or as little veins of pyrolusite as in the deposits of Gandarela, or as masses or beds forming deposits of considerable extent and thickness as in the region which extends from Mariana to Lafayette.* The only deposits exploited up to the present time are those at Miguel Burnier and Queluz. They are worked open cast. The cost of production is increased unnecessarily by unsystematic removal of the overburden. The cost of digging the ore varies with the depth. At first it was only 3 milreis per ton, but at present amounts to from 12 to 15; by better methods of working, however, this might be reduced to 6 milreis. At Lafayette, where the narrow gauge and broad gauge railways unite, the ore has to be transhipped, increasing the cost of carriage, which amounts to 10·140 milreis per ton. The railway is

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* *Revue Universelle des Mines*, October, 1898.

unable to transport more than 1,800 tons per month. The ocean freight from Rio de Janeiro to Great Britain and the United States varies between 9 and 10s. per ton. These costs are increased further by the government taxes and the royalty to the owner of the land, the latter amounting to 0·5 to 1 milreis per ton. Consequently the cost of the ore ex-ship in England is from 57·340 to 49·295 milreis, according to the fluctuations in exchange. These costs, however, by improved methods of shipment might be reduced to 47·840 and 40·621 respectively.* The milreis is worth nominally 54·6c. United States currency.

Canada.—Manganese ore is produced in Nova Scotia and New Brunswick. In the former province considerable quantities of wad have been found, while many brown hematites contain manganese, some assaying as high as 14% MnO. Pyrolusite, however, is the only ore that has yet paid for mining. It occurs in small pockets and veins in granite and also in the quartzites of the Lower Cambrian, but the most extensive deposits are found in the marine limestone of the Lower Carboniferous. The most important manganese mine of Nova Scotia is situated at Tennycape, in Hants County, where the ore is chiefly pyrolusite, yielding from 88 to 95% of available oxide. The most extensive deposit of manganese ore in New Brunswick is situated at Markhamville, Kings County, where the mineral occurs in pockets (occasionally as large as 4,000 tons) in Carboniferous limestone.

A new source of manganese ore has been found in the large deposits of wad (bog manganese) which are known to exist in New Brunswick at two places, at least, and perhaps at others. These deposits occur in beds 5 to 30 ft. thick immediately under the turf. The Mineral Products Co. of New York carried on operations in 1898 at Dawson Settlement, where the soft ore is dried in a revolving cylinder furnace, bricked with a proper binder, and smelted to ferromanganese at Bridgeville. The ore, dried at 212° F., contains 48·24% Mn, 5·70% Fe, 0·096% S, 1·88% SiO_2, and only a trace of phosphorus. The deposit is favorably situated for shipment of ore by water to the United States or elsewhere. A bed of wad from 5 to 15 ft. thick exists at Point de Bute.

Chile.—The richest deposits of manganese in Chile are found in the Department of Freirina, the southern coast district of the Province of Atacama, and in the Province of Coquimbo. Analyses of three grades of manganese ore from the latter province showed 49·79, 52·66 and 40% Mn. The most important mines are situated at Astilla, connected by railway with the port of Carrizal Bajo; Palo Blanco, where the bed is 1 to 5 meters thick; Talquilla, with a bed 1 to 5 meters thick; Yerbas Buenas and Chañarcillo, both lying in the hinterland of the port of Totoralillo, and without railway connection; Canas, in the department of the Elqui, where the railway extends to the mouth of the valley; Corral Quemado, north of Ovalle; Cerro de la Gloria, near Combarbala; and Esquivel, near Illapel. The last two are without railway connection.

Cuba.—According to Raimundo Cabrera, manganese ore is abundant in the Province of Santiago, where many deposits of the mineral have been located, although few of them have been worked extensively. At Alto Songo veins of

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* *Die Chemische Industrie*, Jan. 1, 1899.

pyrolusite have been developed to considerable extent, but at Santa Margarita, Isabelita and Bostoyo little effort has been made to develop important deposits. The most noteworthy workings at the present time are those of Cristo and Ponipo. In general the manganese deposits of Cuba are worked by persons who secure leases from the proprietors of the land. The industry is restricted by the difficulty of transportation, which is mostly by ox carts over wretched roads. Many important mines have been abandoned temporarily on this account.*

France.—Las Cabesses Manganese Mining Co. was reported to be erecting a 120-ton furnace at Bordeaux to smelt the ore of its mines to ferromanganese.

Greece.—(By E. Grohmann): "The production of manganiferous iron ore in Greece in 1898 was 213,938 metric tons, against 182,850 in 1897; the production of manganese ore increased from 11,868 metric tons in 1897 to 14,097 in 1898. New explorations in 1898 proved that manganiferous hematite, or in its place iron-manganese carbonate, is far more extensive at Laurium than had been believed previously. Several new channels of ore, 5 to 30 meters wide and 2 to 8 meters thick, striking N.E. and S.W. under the entire eastern half of the district, were discovered, and a production for many years as large as that in 1898 seems to be assured. The occurrences of manganese ore in Greece are numerous, especially in a certain stratum of the Tertiary formation in Attica, Bœotia and Eubœa and in the Peloponnesus, but all of these are too high in silica to make mining profitable. It is only on the island of Milo that manganese is mined successfully, a bed occurring there in the Tertiary volcanic tuffs. A simple washing is sufficient to separate the tuff and quartz mixed with the ore and make a product which if not high grade is nevertheless marketable. A Belgian-English company has been making explorations for a considerable time on the island of Andros, but so far without any particular success."

India.—The manganese deposits of this Empire were described in THE MINERAL INDUSTRY, Vol. VI. So far the Province of Madras is the only one exporting this ore. Trade has made remarkable progress during recent years, which has been due chiefly to the success of the operations of the Vizianagram Mining Co., and the increased demand for the ore on account of the unsettled condition of the industry in the Caucasus.

Japan.—According to British Consular Report No. 2176, Annual Series, manganese mining was begun in Hokkaido in 1894. The output that year was 1,300 long tons. In 1895 it was 5,700 tons; in 1896, 7,500; and in 1897, 5,200. The mines are situated at Setanai, Aza Toshibetsu, at the head of Volcano Bay, and the deposits so far appear to be fairly extensive. The ore is of high grade, ranging from 51 to 59·2% Mn. The tenor in phosphorus ranges from 0·083 to 0·169%. The ores from this island, as well as those from the northern mainland, are shipped to Yokohama, whence they are forwarded to Europe and America. The principal mines on the northern mainland, however, do not appear to be as productive as formerly.

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* *Engineering and Mining Journal*, Sept. 10, 1898.

Newfoundland.—There was a production of 1,500 long tons of manganiferous iron ore in 1897, but none in 1895, 1896 or 1898.

Russia.—The conditions attending the production of manganese ore in the district of Sharopan, in the Caucasus, were exceedingly unfavorable in 1897, and a large proportion of the small traders and mine owners came to grief owing to ill-judged competition among themselves and the lack of principal wherewith to tide over the crisis. Application was made to the government for a reduction of the freight rate over the Tchiaturi branch of the Trans-Caucasian Railway, while the organization of a combination for the curtailment of the output and its marketing was considered. The exports of manganese ore (reported in tons of 2,240 lb.) from Poti and Batum since 1893 have been as follows:

Destination.	1893.	1894.	1895.	1896.	1897.
Great Britain....	42,930	65,110	606,160	77,754	68,650
France...........	4,100	None.	150	5,650	None.
Russia...........	None.	9,890	9,600	20,175	28,446
Belgium..........	3,125	2,520	None.	220	None.
Germany..........	40,405	51,455	59,565	58,925	70,810
United States....	36,070	28,300	55,787	3,600	42,200
Totals..........	126,630	157,275	185,718	166,324	210,106

The price of manganese ore at Poti in 1897 was less than the actual cost of production including the cost of transportation to that port, hence the increase in the shipments in 1897 was gained only by enormous sacrifice on the part of some of the dealers.

New deposits of manganese ore have been found in the valley of the river Tchorokh, about 50 miles from Batum, and the quality of the ore is said to be superior to that which exists at Tchiaturi. English capital has been invested in these deposits, the exploitation of which has already been begun, the ore being exported through Batum.

The unfavorable conditions in manganese mining in the Caucasus were not ameliorated in 1898, and the high freight rates on the Tchiaturi branch of the Trans-Caucasian Railway, which was built especially for the manganese traffic, compelled many large producers to cease operations notwithstanding the large demand for the product. Some of the small mine owners carted their ore to Poti, and in spite of the bad mountain roads found this method of transportation cheaper than by rail. As a consequence of these conditions the Caucasian manganese industry is still in a chaotic condition and the miners are not in a position to make prompt shipments, while prices are subject to many fluctuations. The uncertainty as to delivery and price has caused many consumers who formerly used Caucasian ore to now obtain their supply from India and Chile. This is reflected in the large increase in production shown especially by the Indian statistics.

Spain.—The manganese mines of Huelva, which are the only producers in Spain at the present time, are situated in the same metalliferous zone as the extensive deposits of pyrites in the same province, that is to say, in the band of Silurian schists, about 40 km. (25 miles) wide, which begins near Aznalcollar, in the Province of Seville, passes through that of Huelva from east to west, traverses Portugal, and loses itself in the Atlantic Ocean. The manganese deposits, easily traceable by their outcrops, are very numerous, but generally of slight extent. They rarely affect the form of veins, occurring generally in lenticular masses, averaging 40 meters (131 ft.) deep, while a great many of them disappear at half that depth. Their exploitation, which was begun in 1858, continued until 1883, when the high-grade oxide was exhausted, and manganese mining in Huelva was paralyzed until 1895, when it received a new impetus from the working of the carbonates, until then regarded as sterile. The most important mines are the Isabel and Resucitada, the latter working in a solid vein 100 ft. thick. The ore assays 39% Mn, 4·6% Fe, and from 8 to 10% SiO_2. The ore is calcined at the mines and sorted into two grades: No. 1 containing 40 to 47% Mn, and 10 to 13% SiO_2, and No. 2, 30 to 34% Mn, and 18 to 20% SiO_2. The manganese deposits of Huelva were described by Ramon Alonso in the *Revista Minera*, No. 1703 *et seq.*, Vol. XLIX., 1898.

According to Señor Adriano Contreras, in the annual statistical number of the *Revista Minera*, the exports of manganese ore from Spain in 1898 were 138,062 tons, of which 126,769 tons were sent to Belgium, 6,586 tons to France, 4,179 tons to England, and 528 tons to Germany. The increase over 1897 was 37,729 tons, or 37·6%.

Turkey.—According to R. Helmhacker, the Kasandra Mining Co. has a concession in Macedonia, and has shipped a small quantity of ore, chiefly pyrolusite. In Asia Minor manganese ores are mined and shipped from Smyrna, but very little is known about the mines. Near Trebizond there are manganese mines, but the output has never exceeded 1,000 tons a year, shipped to France. At Ordu, on the Black Sea, there are small mines which are not now worked.*

Manganese is mined in the Phinika district of Asia Minor, which produced 1,310 tons in 1897, although the output might easily have been more than 5,000 tons if the operator had been free from restrictions. The ore from this mine assays 52% Mn. The Zangan mine is also productive. Its ore assays 83% MnO_2 and 0.004% P. A few hundred tons were mined in 1897. The Province of Aidin contains a good deal of manganese, but the quality is inferior, the tenor ranging from 38 to 48% Mn.†

METALLIC MANGANESE AND FERROMANGANESE.

The greater part of the manganese ore production of the world is used by the steel works in the manufacture of spiegeleisen and ferromanganese. A comparatively small amount of the highest grade is employed in the chemical industry. The Caucasian ore, which goes as high as 56% manganese, is used chiefly in the iron and steel-industry, and is classed as high grade for this purpose. However, a higher grade is generally used in the chemical industry, where it is employed for the preparation of oxygen, bromine and iodine, in glass making and for coloring enamels, in dyeing and calico printing, for the

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* *Engineering and Mining Journal*, Nov. 26, 1898.

† *Montan Zeitung*, Sept. 1, 1898.

preparation of potassium permanganate and other salts of manganese, as a drier for varnishes, etc. The chief consumers of the highest grade of manganese ore used to be the manufacturers of chlorine, but chlorine is now prepared generally by a different process. In California and at Mt. Morgan, Queensland, considerable high-grade manganese ore is consumed annually for the preparation of chlorine by metallurgical works, where gold is extracted by the vat chlorination process, but in most places this process has been superseded by barrel chlorination, in which calcium chloride is the source of the chlorine.

The production of spiegeleisen and ferromanganese in the United States in 1898 was 213,769 long tons, against 173,695 in 1897 and 131,940 in 1896. The more part of the production in each year was ferromanganese, comparatively little spiegeleisen being consumed now in the United States, but there are no statistics to show the precise proportion. However, the production of spiegeleisen by the New Jersey Zinc Co. was 24,900 long tons in 1898, 20,000 in 1897, and 12,403 in 1896.

A small amount of metallic manganese is now made in the United States, one concern reporting in 1898 an output of 200 lb. of metal free from carbon.

MICA.

THE mica industry in the United States did not experience any material change in 1898, and this country continues to receive its chief supply of this mineral from India and Quebec. The apparent consumption of sheet mica was about the same in 1898 as in 1897, the decrease in the production being about offset by the increase in the imports. The domestic production of sheet and ground mica is presented in the following table:

PRODUCTION OF MICA IN THE UNITED STATES.

State.	1897.				1898.			
	Sheet.		Scrap.		Sheet.		Scrap.	
	Pounds	Value.	Short Tons.	Value.	Pounds	Value.	Short Tons.	Value.
New Hampshire	37,000	$9,250	2,198	$21,980	36,500	$9,125	2,875	$28,750
North Carolina and Virginia	70,852	65,548	644	6,440	71,468	80,307	644	10,987
South Dakota	8,500	3,000	30	300	*Nil.*	*Nil.*	*Nil.*	*Nil.*
Other States	2,500	5,500	10	100	2,000	2,000	10	100
Totals	118,852	$83,298	2,882	$28,820	109,968	$91,432	3,529	$39,837

The production grouped under "Other States" includes small amounts produced in Idaho, Nevada and Wyoming. Under the new classification required by the Dingley tariff act mica imported into the United States after July 24, 1897, was designated as "unmanufactured" and "cut or trimmed," a duty of 6c. per lb. being imposed upon the former and 12c. per lb. upon the latter, with an additional 20% ad valorem duty upon each. The imports in 1897 previous to July 24 amounted to 656,118 lb. ($140,353); subsequent to July 24, unmanufactured, 66,821 lb. ($10,981), and cut or trimmed, 226,771 lb. ($41,068); a total for the year of 949,710 lb. ($192,402). In 1898 the importation of unmanufactured mica was 877,930 lb. ($115,930) and of cut or trimmed 78,567 lb. ($34,152), a total of 956,497 lb. ($150,082).

Commercial Requirements.—An expert in the mica trade contributes the following notes as to the uses and requirements of commercial sheet mica: Sheet mica must split easily, *i.e.*, make a clean separation of its laminæ without cracks and tears such as occur when hard, brittle mica is split. It must be tough, flexible and soft. Nearly all mica is soft enough to be scratched with the finger-nail, but this test is not sufficient. In general, sheets of mica which show a smooth surface are softer than those which show blemishes or corrugations when split. Mica which is found cracked at the surface will not improve in character with depth. Imperfections caused mechanically are not referred to, since these will probably disappear, but cracks which originated at the formation of the crystals are meant. Some experience is required to discriminate between these blemishes. The common opinion that mica is easily injured in mining is erroneous, since actual tests in blasting mica with high explosives have showed that it is not shattered. A. J. Hoskins took advantage of this fact to introduce power drills in mica mining in New Hampshire, which was a great improvement in the method of mining.

Uses.—The most valuable property of mica is its electrical non-conductivity. On its freedom from streaks and iron stains its value in this respect is based. For electrical purposes Indian and Canadian micas are considered superior, yet there is no doubt that preconceived ideas govern the judgment as to this to a considerable extent. The color of mica affects its commercial value only when it is required for the stove trade, and in the small amount used for instruments, compass covers, etc. The Canadian mica is of amber color and the Indian a ruby color, but both are uniform in shade and show neither spots nor streaks.

The demand for mica, and consequently its price, is governed chiefly by the condition of the electrical machines industry, at least in so far as sheet mica is concerned. The scrap mica resulting from the dressing of sheet mica finds various uses. Some of it is ground to powder and compressed into various shapes for electrical insulating. The powder is also used in the manufacture of certain kinds of paints, in some kinds of wall paper, and as an absorbent for nitro-glycerine in the manufacture of dynamite. Small sheets of mica, useless for other purposes, are employed in the manufacture of micanite, which is made by splitting the mica into thin laminæ, reassembling these into sheets and solidifying them by a highly insulating cement under a high pressure and at a high temperature. This relamination is said to improve the insulating property of the mineral. Micanite is also claimed to have greater flexibility than the sheet mica from which it is made up, the cement used in it softening with heat and permitting the sheets to conform without cracking to the curved surface to which it may be applied. Upon cooling it retains the same position and becomes so dense as to emit a metallic sound when struck.

New Hampshire.—(By Albert J. Hoskins): "Mica mining in this State is confined to Grafton, Danbury and Alstead. The most important producer in Grafton is the old Ruggles mine, which has been in constant operation since 1803, and has produced over $8,000,000 worth of mica, and at one time furnished four-fifths of the total requirement in the United States. Owing to the owners having been involved in litigation for the last 20 years, the mine has not been worked to its greatest capacity, although it has shipped several tons of sheet and 1,000 tons of scrap mica within the last 12 months. This mine is operated in a very primitive way, the drilling being done by hand and no machinery of any kind being employed. The block mica after it is taken from the mine is split into sheets one-eighth of an inch thick and the rough edges trimmed with a common knife. It is then packed into boxes and shipped to the dealers, who cut it into the different shapes required by the manufacturers of electric machinery, in which it is used as an insulator in the manufacture of commutators, rheostats, armatures, etc.

"Several other mines have been opened at Grafton during the last eight years, but no one of them has proved successful. In Danbury there are two mines in operation, one owned and operated by the Empire Mica Co., the other by Frank Sargent. The Empire mine is equipped with modern machinery and has produced within three months about 100 tons of scrap mica, besides considerable sheet mica, and is considered a good paying mine. The quality of the mica is very superior, it being pure white. The Sargent mine is operated by hand, and is worked more for scrap mica than for sheet. The average monthly output of this mine is about 500 lb. of sheet and 50 tons of scrap.

"In Alstead there are three mines in operation: The Davis mine, owned by the Davis Mica Co., of Keene, N. H.; the Hoskins mine, and the Warren French mine. The Davis mine is a large producer, having turned out 600 tons of scrap in 1898 and a large amount of sheet. It is considered one of the best paying mines in the State. This mine is operated in a modern way, power being transmitted by compressed air. The mica obtained is rough-trimmed and shipped to Boston, where it is cut into squares and sold to the stove trade. The scrap mica is sold to Albert J. Hoskins. The Warren French mine is operated by hand drilling and produces about one ton per day, including sheet and scrap. The Hoskins mine is operated in a manner similar to the Davis. All of its product is cut into squares, cleaned, put up in pound packages, and sold only to the retail trade, this being the only mine in the State which does not sell to the trust.

"The scrap mica trade throughout the State is controlled by Albert J. Hoskins, who made large shipments in 1898. He has been working over the old dumps of the Davis Mica Co., those of the Bowers, the Sargent, the Mitchell mines, and several others of less importance. The scrap mica is put in bags and shipped to Homestead, N. J., where it is ground into different grades of fineness and sold to the wall paper trade and other industries in which ground mica is used. There is an increased demand for scrap mica, and the price has risen in one year from $5 to $15 per ton. After grinding the product is now worth $50 per ton."

North Carolina.—(By J. A. Holmes): "A number of new mica mines were opened in this State in 1898, and a number of old ones were reopened and worked actively, the industry having been stimulated by the increase in the import duty on mica under the present tariff law. The mica of North Carolina is obtained from pegmatite dikes, in which as a maximum not over 1 or 1·5% of the dike is mica, and about 0·1% is the rule. About 5% of this mica or less is merchantable as sheets; the rest, if utilized, is ground. The chief defects are the crushing and warping due to dynamic processes, and the 'ruling' which runs across the leaves, and is probably due to pressure."

South Dakota.—There were no shipments of mica from the Black Hills in

1898. There are several mines in this region from which some shipments have been made in previous years, and from one mine some shipments have been made since January 1, 1899.

PRODUCTION OF MICA IN FOREIGN COUNTRIES.

Canada.—The production of mica in Canada in 1898 was reported as $117,598 worth, against $75,000 in 1897. This was chiefly derived from Quebec, the remainder being obtained in the Ottawa Valley, Ontario. According to J. Obalski, Inspector of Mines for Quebec, the amber mica mines of that province shipped 550,000 lb. of thumb-trimmed mica in 1898, varying in size from 1×3 in. to 4×6 in., of a total value of $81,000 f. o. b. cars.

(By J. Obalski): "There was no white mica worked in Quebec in 1898, although there are a few important mines of this variety; but there was much activity in mining amber mica in Ottawa County, seven or eight mines, employing more than 250 men, being in operation, against only three or four worked by 50 men in the previous year. The increase in the demand for this product is to be attributed to the fact that it answers the purposes of the electrical industry and can be obtained easily at short notice of any grade and size required. The mica of Quebec is no longer sold in the cut form, but now entirely as thumb-trimmed, sorted at the mine according to size. The crude mineral from the mines is split into sheets about 1 in. thick, and then cleaned, trimmed, sorted and packed in barrels which contain 300 to 400 lb. Steam drills are now used in the mica mines of Quebec, but the miners seem to prefer hand work, which they claim gives them better control of the crystals. The mica mines, which occur in the Laurentian formation, are situated mainly in Ottawa County, in the vicinity of the Gatineau and Lièvre rivers, especially in the townships of Templeton (near Perkins' Mill), Hull, Wakefield and Bouchette. Sometimes the mineral occurs in the form of chimneys and is associated with pyroxene; at other times it occurs in veins in which pink calcite is the matrix containing crystals of mica, pyroxene and apatite. The latter form appears to be the more durable and promising, although some important mines of the former type have been operated."

India.—According to the official statistics the estimated production of mica in India in 1894 was 180 long tons, valued at 1,45,388 rupees; 375·9 (3,16,614 rupees) in 1895; 311·2 (3,42,470 rupees) in 1896, and 445 (1,99,003 rupees) in 1897. These figures do not correspond with the reported exports, and their accuracy is considered doubtful by the Government Reporter.

EXPORTS OF MICA FROM INDIA. (*a*) (IN METRIC TONS AND RUPEES.)

Province from Which Exported	1891-92.		1892-93.		1893-94.		1894-95.		1895-96.		1896-97.	
	Tons.	Rupees.	Tons.	Rupees.	Tons.	Rupees.	Tons.	Rupees.	Tons.	Rupees	Tons.	Rupees.
Bengal	116·7	4,28,393	168·1	4,33,772	246	7,65,879	260·4	6,04,719	448·8	9,95,102	447·3	9,62,724
Bombay	1·4	314	0·9	511	0·25	120	0·5	879	3·9	2,829	1·7	1,892
Burma	....	....	0·05	65	0·30	700	0·3	712	0·05	220	*Nil.*	*Nil.*
Madras	*Nil.*	*Nil.*	*Nil.*	*Nil.*	*Nil.*	*Nil.*	*Nil.*	*Nil.*	*Nil.*	*Nil.*	202·4	1,88,748

Countries to Which Exported	1891-92.		1892-93.		1893-94.		1894-95.		1895-96.		1896-97.	
	Tons.	Rupees.	Tons.	Rupees.	Tons.	Rupees.	Tons.	Rupees.	Tons.	Rupees	Tons.	Rupees.
United Kingdom	41·6	1,18,932	70·5	2,40,952	132·0	5,28,514	154·4	2,78,135	176·1	4,56,942	527·8	9,13,984
France..........	0·5	1,340	....	....	1·3	8,150	1·0	5,050	0·2	1,000	0·2	1,000
Germany.........	3·7	17,425	0·25	764	8·6	33,518	4·7	28,038	14·5	21,794	15·4	38,006
United States...	71·2	2,90,806	93·2	1,92,380	104·7	1,96,517	98·4	2,91,940	257·8	5,11,088	107·3	1,99,504
Other Countries	1·1	204	5·1	252	....	....	2·7	3,147	4·2	7,327	0·8	870
Totals.......	118·1	4,28,707	169·05	4,34,348	246·6	7,66,699	261·2	6,06,310	452·8	9,98,151	651·5	11,53,364

(a) From *Review of the Mineral Production in India.* Fiscal years ending March 31. A large part of the shipments to the United Kingdom is re-exported from that country to the United States.

The last annual report of the Geological Survey for India contains an account of a recent discovery of mica in Coorg, where five or six localities are said to be able to furnish sheets above the average market size. The largest were found on Elk Hill, where the sheets are quite free from warping, and measure as much as 30 in. in width. All the samples are of good color, and show a high degree of elasticity, though the value of some is reduced by the presence of ferruginous inclusions between the cleavage planes, and occasional warping. There is evidence that the mica-bearing pegmatites extend over a wide area.

Other Countries.—New discoveries of mica were reported in 1898 at a point 200 miles north of Kamloops, B. C.; near Paringa, New Zealand, and near Villa Corona, Durango, Mex., about 40 miles from railway, where sheets of good quality as large as 6 × 6 in. are said to be available.

GEOLOGY OF THE MICA DEPOSITS OF THE UNITED STATES.*

BY J. A. HOLMES.

THE commercially valuable deposits of mica in the United States, though widely separated, are limited to a few districts. They have been worked to some extent along the Appalachian system of mountains in New Hampshire, Virginia, North Carolina, and in the Black Hills region of South Dakota, the Cribbensville district in northern New Mexico, and in western Idaho. Additional deposits of promise have been found and developed on a small scale in Maine, South Carolina, Georgia, Alabama, California, Nevada and Wyoming. These deposits are all found in pegmatite dikes, which occur in gneissic and granitic rocks, usually classed as Archæan in age. The dikes yielding the best and largest quantities of mica are found in the hornblendic and micaceous gneiss and schists, in places parallel to but generally cutting across the schistose structure of these rocks at varying angles. Pegmatite dikes vary in thickness from a few inches to more than 250 ft., and can be traced for a distance varying from a few feet in the smaller ones to several miles in some of the larger ones. In some cases they are irregular and have arms branching out in many directions. Some dikes are vertical, some nearly horizontal and others vary.

In character the pegmatite has been called a very coarse granite, consisting mainly of quartz and feldspar in equal or variable proportions and muscovite mica. In some places the quartz and feldspar will be somewhat uniformly distributed throughout the pegmatite mass, while in others the two will be well

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* Abstract of a paper presented to the eleventh annual meeting of the Geological Society of America.

separated, the feldspar sometimes crystallizing out into masses more than a ton in weight. In addition to these common minerals there occur many others with varying degrees of rarity. Twenty or more of these accessory species are sometimes observed in a single dike in some regions, while the dikes in other regions are almost destitute of them. As a rule, crystals or "books" of commercial mica need not be looked for in dikes which are less than 2 ft. in thickness, although there have been instances in which books 2×2½ ft. in size have been found in dikes the width of which was scarcely more than 2 ft. On the other hand, some of the largest dikes contain no mica of commercial size. As to the distribution of this mica in the dike, generally the books or crystals are scattered promiscuously through the matrix of quartz and feldspar, and a large amount of useless material has to be blasted down and removed from the mine in order to secure the commercial product; in some cases, however, the books of mica occur in the outer part of the dike near the wall rock, and then the mica lead can be more easily followed. In some instances the mica constitutes as much as 10% of the total mass of the dike, but usually it proves to be less than 1% of the total. Of the mica taken from the dike in ordinary mining operations, usually less than 10% and sometimes less than 2% has a commercial value as sheet mica, the remainder being thrown away or pulverized.

The geological age of these pegmatite dikes varies considerably. In crystalline rocks exposed in the lower part of the Grand Cañon of the Colorado in northern New Mexico the dikes break through the granitic rocks, but come unconformably against the base of the Algonkian series there, and are consequently pre-Algonkian in age. All of the larger dikes observed in the Rocky Mountain region have been to a greater or less degree involved in the schistose and other structural modifications of the crystalline rocks, and consequently must have been formed either prior to or during the earlier stages of the uplift of these mountains. In the Appalachian region these dikes are not, in most cases, involved in the schistose structure of the rocks, though in some cases they have undergone considerable changes in connection with the production of the schistosity. The condition of the material in the dikes seems to indicate that they were formed either prior to or during the early stages of the uplift of these mountains, since sometimes they are involved in folds quite similar in character to those typical of the Appalachian structure. Occasionally the sheets of mica themselves have been folded under pressure, but as a rule they show no such disturbance, having been, like the coarse feldspar and quartz of the dike, but little modified in connection with the mountain uplifting. These blocks of mica, however, often have their commercial value in large measure destroyed by the production of what is called "rolled" or "ribbon" mica, the sheets of mica being cut into narrow strips with parallel edges. These edges of the rolled mica appear in all cases to be parallel to certain axes of crystallization, but the cause of the rolling and the conditions under which it has been produced are not well understood.

MICA MINING IN BENGAL, INDIA.*

BY A. MERVYN SMITH.

Position.—The mines described are situated between 85° to 86° 30′ east longitude, and 24° 25′ to 25° north latitude. The country is made up of a series of parallel ranges of low hills about 400 ft. above the surrounding country and 1,200 ft. above sea level. The direction of the ranges is nearly east and west. They form the boundary between the Hazaribagh district on the south and the Gaya and Monghyr districts on the north.

Geology.—The country is made up of gneiss, grading into mica schists, tourmaline schists, hornblende rocks, quartzites and a coarse feldspathic rock resembling conglomerate with intrusive dikes of fine-grained diorite. The hornblende rocks closely resemble the diorite. The mica schists are strongly developed and are made up chiefly of muscovite, with here and there a little biotite and lepidolite. This rock is highly schistose, breaking up into thin laminæ, and consists chiefly of small plates of mica, cemented together with a slight admixture of feldspar and quartz. It is this rock that contains the lodes that yield the mica of commerce.

Pegmatite Veins.—Running with the bedding of the schists, from east to west, are numerous veins of pegmatite varying in width from a mere thread to 20 ft. The schists are in places faulted or thrown out of their normal strike. The pegmatite veins are subjected to the same faulting, and frequently at the point of faulting short north-to-south veins or "cross-courses" of great width have been formed. The veins are made up of amorphous masses of quartz, large crystals of pink orthoclase and crystals or "books" of muscovite. The quality of the vein varies with the nature of the adjacent rock. Near to the quartzites the vein stuff is nearly pure quartz with little feldspar and mica. Where the adjacent rock is highly feldspathic, pink feldspar crystals are the chief constituent, while a large predominance of mica is to be looked for where the including rock is mica schist. A knowledge of this peculiarity enables the miners to discriminate between spots that will yield more or less mica. Near to the tourmaline schists large massive crystals of black tourmaline are not infrequent in the lode, and occasionally crystals of tourmaline penetrate the mica books across the planes of lamination and destroy the value of the sheets.

Mining.—The native mines are of the most primitive kind. Open cuts along the outcrops of the pegmatite veins where books of mica are seen in any quantity are continued down 20 or 30 ft. till the sides become dangerous. Where exceptionally rich stuff is met, and the vein stuff is decomposed and soft to 50 or 100 ft. in depth, inclines are put in, and follow down the rich shoots of mica in a tortuous course, zigzagging from side to side with the "leads" of rich ground. Work is conducted only in the dry months (November to May). The books of mica are split into sheets about ⅛ in. in thickness. All the rough edges and flaws are trimmed off by means of a sharp sickle (called "hasawah"), and the sheets are then sorted according to color and size. They are in some cases very large, 24 × 18 in. being the largest in this district. In the Itakuri

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* Abstract of a paper in *Transactions of the Institution of Mining and Metallurgy*, 1899.

mine, Nellore district, Madras, plates 60 × 40 in. have been obtained. Within recent years Europeans have taken up the mica industry, but mining is still conducted by the native methods.

After trimming, the plates are sorted for the foreign market, four kinds being recognized by the dealers: (1) Ruby mica, hard and tough; (2) white transparent mica; (3) discolored and smoked; (4) black mica and flawed. If 8 represents the value of ruby mica, 4 would be paid for white, 2 for discolored, and 1 for black and flawed sheets of equal size. The sizing is as follows: Specials, sheets measuring more than 50 sq. in.; No. 1, sheets of from 36 to 50 sq. in.; No. 2, 24 to 36 sq. in.; No. 3, 16 to 24 sq. in.; No. 4, 10 to 16 sq. in.; No. 5, 6 to 10 sq. in.; No. 6, 4 to 6 sq. in. Quoting from a recent sale of sheet at an auction in London, the following prices were realized: Best ruby, No. 1, 6s. 8d. per lb.; No. 2, 4s. per lb.; No. 3, 2s. per lb.; No. 4, 1s. per lb.; No. 5, 4d. per lb.; No. 6, 2d. per lb. Specials bring as much as £1 a lb. according to size of sheets. The sheets are trimmed irregularly into any shape they will take to clear them of flaws. Should square, rectangular, or diamond-shape sheets be wanted, a special rate has to be paid for these to allow for the great waste. The sheets are packed in boxes of 112 lb., and transported on carts to the nearest railway station, 100 miles distant. Thence they go to Calcutta for shipment to London or New York.

Quantity of Mica Available.—The pegmatite veins in this locality are numerous and of large size, and are in places extremely rich in mica, nearly one-third of the vein stuff being of this material. The quantity of mica available is therefore very large. The present method of mining is wasteful in the extreme, and fully nine-tenths of the mica extracted from the mines is injured and rendered unfit for export, while thousands of tons of waste mica are to be found at the mouths of the mines and at the dressing floors. Without doubt when European methods of mining are introduced the cost of production will be materially reduced, the waste will be much less, and a better quality of mica will be secured. Only the decomposed vein stuff is mined. In the hard vein stuff the mica is hard and tough, and this commands a better price.

Some of the veins yield black mica (biotite), which is largely used as a drug by Hindus and Mohammedans. Reduced to a powder, it is supposed to be very efficacious in cases of dysentery. The sheets of mica are at times very queerly marked. In places one-half of each sheet will be white and the other half black, the line of division between the two colors being a perfectly straight line, and there being no apparent change beyond the color in the uniformity of the sheet. Other sheets are marked with a checkered pattern in black lines, the lines being due to magnetite. Again there are dendritic inclusions of white quartz between the laminæ. All these markings detract from the value of the sheets. The most esteemed colors are pure ruby, amber, light green, transparent white. There is also a silver white, prized for inlaid work.*

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* Mica mining in Nellore, the new district of Madras, which is becoming an important producer, was described by Robert W. Thompson in the *Journal of the Society of Arts*, June 17, 1898.

MOLYBDENUM.

MOLYBDENUM and ferromolybdenum are now regularly produced in the United States, there having been a production of 9,550 lb. of the former and 2,100 lb. of the latter (containing 50% Mo) in 1898. This was smelted mainly from molybdenite mined in Arizona and New Mexico, the output of this mineral having been 16,000 lb., which assayed from 42 to 56% Mo. No mineral with less than 42% Mo is used. Its value ranges upward from $50 per 2,000 lb., according to its tenor in molybdenum and freedom from undesired impurities. Certain ores of molybdenum carry injurious elements which can only be removed at great expense, and such ores have no value whatever on this account. The value of the metallic molybdenum produced was about $1·25 per lb.

Besides the above-mentioned production of molybdenite, about 12 short tons of wulfenite ($PbMoO_4$=39·3% MoO_3+60·7% PbO), assaying 90% $PbMoO_4$, were shipped from the Mammoth mine, Pinal County, Ariz., realizing $12 per ton at the mine. The destination of this shipment was not reported.

Occurrence of Molybdenite.—Molybdenite is found at many localities in the United States, mention of which was made in THE MINERAL INDUSTRY, Vol. VI. Samples from Maine recently tested showed 13 to 15% Mo; some from the West 15 to 17% Mo and 6% Cu. Ore of this grade has to be concentrated. Concentrated ore from Saxony has about 60% Mo. Good molybdenite is also found in Greenland, Finland, Italy and Bohemia. About 16 miles from Sonora, Cal., there is said to be a 3-ft. vein of molybdenite with silicious gangue in slate, in close proximity to several parallel gold-bearing veins.. The vein is reported to be remarkable for the uniformity with which the molybdenite is diffused through the gangue, which is a hard, white crystalline quartz. C. W. Willimott reports the occurrence of molybdite in the township of Egan, Ottawa County, Quebec, where it is found in connection with molybdenite in earth of a dull yellowish color, which, in addition to the usual constituents of soil, contains a very large proportion of ferric hydrate and calcium, magnesium and iron sulphates, and approximately 7·5% of molybdenum trioxide. The same authority reports the occurrence of molybdenite in Pontiac, Ross and Desert, some of the deposits being large.

Molybdenum Steel.—Molybdenum is used chiefly in the preparation of special steels, for which purpose it is employed in the form of dark-blue, powdered metal, with 95 to 99% Mo; as ferromolybdenum, with 50 to 55% Mo; and as molybdenum-nickel, with 75% Mo and 25% Ni. For tool steel from 2 to 4% Mo is added; for other qualities from 1 to 2%, according to what is desired. The influence of molybdenum on steel is similar to that of tungsten, but it gives greater toughness, while molybdenum steel is more readily worked when hot and stands hardening better than tungsten steel. The properties of molybdenum steel have been described more fully by Prof. W. von Lipin,* who made a series of tests on the relative resistance of tungsten and molybdenum steels prepared under the same conditions. Both steels were made in Siemens regenerative furnaces from a charge consisting for the molybdenum steel of 20·6 kg. basic open-hearth steel, 2 kg. Swedish charcoal pig iron, 2·3 kg. Swedish charcoal blooms, 0·1 kg. ferrosilicon, and 1 kg. molybdenum in metallic form, but combined with a small quantity of carbon. For the tungsten steel the charge consisted of 16 kg. basic open-hearth steel, 3·7 kg. Swedish pig iron, 2·9 kg. Swedish charcoal blooms, 0·1 kg. ferrosilicon, and 1·8 kg. ferrotungsten having 48% tungsten. Both steels were cast into ingots, which were afterward rolled into rods of a suitable size for the testing machine. By analysis the percentage of tungsten was found to be 3·8, and that of molybdenum 3·7. The experiments showed that annealing makes molybdenum steel softer than tungsten steel, while high heating makes it harder. Oil tempering and high heating after hardening increased the limit of elasticity in the molybdenum steel. Common tempering in oil had a greater influence on the tungsten steel than on the molybdenum, but, on the other hand, the molybdenum steel was stronger than the tungsten after heating and hardening in water. Tungsten steel was more apt to split than the other when worked, and broke sooner when bent cold. The molybdenum steel withstood forging and hardening better than the tungsten steel.

Estimation of Molybdenum.—H. Borntraeger† describes the following method of determining the amount of molybdenum available in molybdenite: About 1 g. mineral is digested with 25 c.c. concentrated nitric acid for two hours in an Erlenmeyer flask in order to convert the molybdenum into molybdic acid. The latter is dissolved in ammonia and filtered off. The residue is treated a second time with nitric acid and ammonia. The combined filtrate is then acidified with nitric acid and evaporated to dryness. The molybdic acid is separated from the dry ammonium nitrate by leaching with 50% alcohol, which dissolves only the ammonium nitrate. The molybdic acid is then collected upon a weighed filter, or, better, is dissolved in normal ammonia and the excess of alkali titrated with acid.

The analysis of molybdenum compounds has also been described by Harry Brearley, who says‡ that molybdenum may be estimated indirectly by adding an excess of lead acetate to a solution of alkali molybdate, filtering off the lead molybdate and determining the amount of lead in the filtrate by any of the trustworthy volumetric methods of determining lead. The determination by means of potassium chromate and ferrous sulphate has given good results, but needs careful working.

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* *Stahl und Eisen*, 1897, No. 14. † *Zeits. f. Anal. Chem.*, 1898, 37, p. 438. ‡ *Chemical News*, Oct. 21, 1898, p. 203.

Schindler has described* a method whereby lead acetate is added to an acidified solution of a molybdate until the latter ceases to give any coloration on a test-plate with tannin. This method gives rapid and approximate results, but toward the end of the reaction the indications are rather uncertain. In neutral or faintly acid (acetic) solutions the precipitated lead molybdate is granular and easily filterable. Utilizing this fact Brearley has made the following modification of Schindler's method: Add lead acetate and use the tannin reaction as long as it is available. The degree of accuracy thus obtainable will vary little more than 1% from the truth. Pour a little of the hot solution through a ribbed paper, and test the filtrate first with lead acetate, and if necessary with sodium molybdate. Generally the first test will show that some molybdenum is still unprecipitated, and the intensity of the cloudiness, after a little experience, indicates approximately how much. The one or the other reagent is added as required, and the solution tested in this way until with absolute certainty it can be said that neither sodium molybdate nor lead acetate is present in other than negligible quantities. As the solution is passed and repassed through the same filter until the end is reached, there is no loss through absorption in the paper. And, finally, it is always open, if desired, to complete the gravimetric operation. This process is shorter than the gravimetric, and on pure solutions is equally accurate. Large amounts of metallic salts do not interfere.

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* *Journal of the American Chemical Society*, November, 1895.

MONAZITE.

As in previous years, monazite was produced in the United States in 1898 only in North Carolina. There was not much activity in monazite mining in the early part of the year, but during the latter months several of the deposits were worked on a limited scale. A portion of the product was shipped for home consumption, but the larger part was purchased for export. The production was 150,000 lb. ($7,500), against 40,000 ($2,000) in 1897, 17,500 ($875) in 1896, 1,900,000 ($114,000) in 1895, 750,000 ($45,000) in 1894, and 130,000 ($7,600) in 1893. Up to 1895 the United States and Norway were the principal sources of monazite, thorite and orangite, but since 1895 Brazil has dominated the market. The production in Norway was never large. The only deposit (thorite and orangite) worked commercially in that kingdom is that of Langesundfjord, between Arendal and Christiania. The monazite deposits of the United States and the methods of mining monazite sand have been fully described in the previous volumes of THE MINERAL INDUSTRY.

North Carolina.—(By A. E. Heighway): "Monazite mining in North Carolina has been checked by the competition of the cheaper mineral from Brazil, where vast supplies exist. After a year of inactivity in Cleveland County, N. C., a spasmodic effort to revive the business was made. Several carloads of monazite which had been mined during the previous year or earlier, and had remained on hand, scattered among a large number of holders, were bought up early in 1897 and shipped. Owing to the length of time this sand had remained on hand, and a total lack of buyers at any price, it was possible to purchase these mixed lots at a low price. Even with this advantage it was found impossible to meet Brazilian competition; much less can this be done where it is necessary to mine the mineral. Nearly all the mines or streams have been worked over once, and any new work must be at a disadvantage, labor being less skilled, while all the old tools have been lost or worn out. The present inquiry for monazite sand is brisk enough, but the conditions imposed are practically prohibitory. The nature of monazite mining in North Carolina is such that only comparatively small quantities can be obtained from any one locality—at the outside 10 tons, and this only after considerable time. Buyers want at least car lots and regular quick shipments, besides demanding a guarantee as to thoria contents. The last requirement is a serious obstacle, since the tenor of the mineral in thoria varies from 1·5 to 6·5%. Consequently only well known mines which produce a sand high in thoria can be worked at all. About 5% thoria is an acceptable percentage. Only an analysis can correctly determine the thoria contents. Some bright yellow 90% monazite sands may be far lower in thoria than apparently inferior sands. Few of the miners have any capital, and they are unable to carry on hand more than 500 lb. of sand, or to give a guarantee. There is not profit enough in the business to induce any one with capital to take it up."

Brazil.—The discovery of monazite sand in Brazil was made by M. Gorceix, the director of the Ouro Preto School of Mines. It is found in the Province of Bahia, at Salabra and Caravellas; in Minas Geraes at Diamantina; also in the Provinces of Goyaz, Cuyaba, Sao Paulo and Rio de Janeiro. The most extensive deposits occur in Bahia, existing as sand banks at the edge of the ocean, particularly at the southern extremity of the province, near the island of Alcobaca, and not far from a small village called Prado. The continuous destruction of the rocks by the sea waves has gradually enriched the deposits, and the sands can be loaded directly upon vessels, thus making the cost of handling very low. The Brazilian monazite sand is finer than the American, which has to be ground before use, and it is also of more uniform grade, containing an average of 4·5 to 5·2% of thoria. In the gold placer deposits and diamond fields of Minas Geraes, Cuyaba and Goyaz, the monazite exists in large nodules, colored a brilliant yellow by orangite. Monazite has also been discovered in the gold placers of Rio Chico, at Antioquia, in the United States of Colombia, and in the river sands of Buenos Ayres, in the Argentine Republic.

According to recent reports to the State Department by H. W. Furniss, United States Consul at Bahia, the monazite industry in Brazil was disturbed in 1898 by a dispute between the federal and state governments as to the ownership of the deposits. The former has title to a strip of land along the coast, called "marinhas," 33 meters in width, measured inland from midway between high and low water marks, in which it is claimed the deposits occur chiefly. The state government, on the other hand, asserted that they are found inside of this strip. Several years ago the federal government gave to John Gordon, an American citizen resident at Rio de Janeiro, the right to dig the sand contained in the "marinhas." In addition to this federal grant, Mr. Gordon obtained a municipal grant from the town of Prado, which is in close proximity to the largest deposits of sand. He also acquired by purchase considerable private property bordering on the deposit. For some time he had ships go to Prado and load this sand under the name of ballast; but finally the state and federal governments became aware of the value of the sand (at that time it was worth about £80 per ton), and after considerable discussion he was forced to pay as export tax on each ton 22% of its estimated value to the state, exclusive of whatever may be allowed as expenses of shipment, 2% on the same basis, for what is styled "statistical purposes," 2·5% federal tax, and a municipal tax of about 1% if landed at Bahia. The trade was large until 1896, when the governor claimed, in the name of the state, part of the deposit, and refused to allow any sand to be removed therefrom, unless under state concession. He claimed that the federal lands were not as extensive as had been said. After much argument, the governor decided to force the contention, and finally gave state concessions as follows: June 8, 1898, to Rebeiro & Co., 5,000 tons; July 18, 1898, to S. S. Schindler, an American resident in Bahia, 5,800 tons; and September 2, 1898, to Manuel Duarte, 5,000 tons. Pending the settlement of this dispute little sand was removed. The first cargo in several months reached Bahia in October, for shipment to Germany. This cargo, which contained 810 metric tons, was assessed for taxation at £25 per ton with a deduction of £5 per ton to cover duties and shipping expenses, making the tax in this instance 26% on 550 reis paper (about 9c. United States currency) per kg.

In the latter part of December Mr. Gordon made a contract, whereby the governor, on behalf of the state, granted to him for 20 years the exclusive right to remove monazite sand from the lands of the state, and that in the districts of Alcobaca and Porto Seguro, Gordon binding himself to pay the state £1 at the exchange of the day for each ton removed, in addition to the state and other export taxes. All concessions already granted—for 15,800 tons—were transferred to him by the concessionnaires, Gordon binding himself to pay them pro rata £2 for each ton until the 15,800 tons are removed. The concessionnaires also agreed to stop all suits, etc., relative to the title of the land and the right to remove the sand. Gordon now has the exclusive right for the extraction of monazite sand in Brazil.

The production of monazite sand in Brazil since the beginning has been as follows: 1895, about 3,000 metric tons; 1896 and 1897, 400; 1898, 1,950—total, 5,350, of which 2,600 tons have gone to Hamburg.* Austria has also been a large consumer, and recently small amounts have gone to England.

THE COMMERCIAL VALUE AND TECHNOLOGY OF MONAZITE.

Commercial monazite, which is a phosphate of thorium, cerium, lanthanum and didymium, with 4 to 9% ThO_2, is valuable only for its content of thoria, no technical use for the ceria, lanthana or didymia, which form, on an average, 58·9% of its composition, having yet been found. One works in Saxony is said to have 50,000 kg. of Di_2O_3 in stock at the present time awaiting a day when there may be a demand for it. The price of thorium nitrate in Germany in 1898 was 40 marks per kg., against 70 in 1896 and 450 in 1895. In New York it was quoted in 1898 at $8 per lb. in 1 lb. lots, and at $7·50 per lb. for 12½ lb. lots, these prices being for the salt assaying 49 to 50%.

Efficiency of Welsbach Mantles.—E. A. Medley controverts the claim that Welsbach burners maintain their candle power practically constant until the mantles are destroyed, and shows that the depreciation of the candle power with them is as rapid as in the filaments of incandescent electric lamps. Economy dictates the substitution of a new incandescent electric lamp when the initial candle power has depreciated to 80%, but taking into account the high first cost of Welsbach mantles and the low cost of operation, a mantle would

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* Statistics compiled by a German authority; communicated by Arthur Dieseldorff, Freiburg im Baden.

undoubtedly be continued in use until its candle power has dropped much lower. In this case the customer would be obliged to consume the same amount of gas and thus be at the same expense, although the light obtained would be much diminished in power. For example, if a Welsbach mantle having an initial candle power of 32 were burned for 500 hours, its candle power would drop about 27%, leaving approximately 23·4 candles. Mr. Medley sums up as follows: "The mantles give a wonderfully good light when new, but the extreme rapidity with which the light falls off and the consequent shortness of their efficient life, even when treated with every consideration due to their somewhat delicate constitutions, makes them far from an ideal illuminant. With the possible exception of the German mantle No. 3, a great deal of improvement is necessary before, for domestic lighting at any rate, they can be considered satisfactory."*

Analysis of Incandescent Mantles.—E. Hintz† states that almost without exception commercial mantles consist of ThO_2 with 0·5 to 2% of CeO_2; and, apart from a small quantity of CaO, they contain only minute amounts of impurities derived from the thorium nitrate, namely, ZrO_2, Nd_2O_3 (neodymium oxide), La_2O_3 and Yt_2O_3. The estimation of these elements is a laborious operation. Photometric experiments made by Hintz showed that (1) ZrO_2, La_2O_3 or Yt_2O_3, up to 1%, do not increase the lighting power of pure ThO_2 mantles; and Nd_2O_3 increases it but little, the admixture of 1% giving 13 Hefner candles with an hourly consumption of 111 liters of gas (=11·4 English candles with 3·9 cu. ft.); (2) admixtures of up to 1% of Nd_2O_3, La_2O_3 or Yt_2O_3, or from 0·2 to 1% of ZrO_2, have no influence on the light given out by thoria-ceria mantles; the influence of 0·1% of ZrO_2 is doubtful; (3) the addition of up to 1% of ZrO_2, Nd_2O_3, La_2O_3 or Yt_2O_3 has no influence on thoria-ceria mantles containing 0·5% of CaO; the CaO in mantles is probably derived from the ash of the fabric; it causes serious shrinkage and should be avoided.

Hintz, in conjunction with H. Weber, examined Knorre's process for estimating ceria,‡ in which the solution of cerous salt is acidified with H_2SO_4, and oxidized to the ceric state by boiling with ammonium persulphate, the yellow ceric compounds being reduced by dilute peroxide of hydrogen to the colorless cerous state, and the excess of H_2O_2 titrated with permanganate (10 atoms Fe=2 mols. $KMnO_4$=5 mols. H_2O_2=10 mols. CeO_2; therefore 56 parts Fe=140 parts Ce=172 parts CeO_2); 3 g. of ammonium persulphate are sufficient to oxidize from 0·2 to 0·3 g. Ce. The process gives correct results with thoria-ceria mantles, when applied carefully, as follows: Acidify the solution (see below) containing about 1 g. of ThO_2 and 0·01 g. of Ce_2O_3 with 5 to 7·5 c.c. of dilute sulphuric acid (1 vol. H_2SO_4+6 vols. of water), dilute to 200 c.c. and add part of the ammonium persulphate. Heat to boiling for one to two minutes, cool to 40° to 60° C.; add a second amount of persulphate, again boil and cool; add a third portion and boil for 10 to 15 minutes. Toward the end of the boiling add 2 c.c. more of dilute H_2SO_4. Cool completely, add dilute H_2O_2 from a burette until the yellow color has just gone, and titrate back at once

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* *Electrical Engineer*, Sept. 29, 1898.

† *Zeits. Anal. Chem.*, 37 (2), pp. 94–111.

‡ See *Journal of the Society of Chemical Industry*, 1898, p. 72.

with permanganate solution. Small quantities of nitrates do not affect the results, but acidification with HNO_3 instead of H_2SO_4 leads to low results.

A simplified process for analyzing incandescent mantles is as follows: Take at least 12 mantles, in the unburnt condition, and cut off the upper and lower ends, retaining only the middle parts, as the upper ends are sometimes treated with a strengthening fluid, and the lower ends may contain an excessive proportion of cerium nitrate. Weigh the quantity taken and extract completely with water containing a few drops of nitric acid. Ignite the remaining fabric, fuse with $KHSO_4$, dissolve in water and HCl, precipitate with ammonia, add the nitric acic solution of the precipitate to the main solution, and dilute to a convenient bulk. (1) Neodymia (1 part of Nd_2O_3 to 100 parts of ThO_2) may first be detected in some of the concentrated solution by its characteristic color and absorption bands. (2) Precipitate part of the solution with oxalic acid; after removal of free acid by evaporation, collect the precipitate (*a*) on a hardened filter paper, wash into a beaker, heat with strong ammonium oxalate solution, dilute, allow to cool, filter after standing a considerable time, ignite, and weigh. A little thorium oxalate will remain insoluble, so that, even in absence of Nd_2O_3, La_2O_3 and Yt_2O_3, the weight of the precipitate will exceed that of the CeO_2 present; at the same time the CaO affects the estimation. If the weight, however, exceeds that of the real CeO_2 present by not more than 1%, it may be assumed that Nd_2O_3, La_2O_3 and Yt_2O_3 are not present in appreciable quantity. If the excess be greater than 1% the precipitate must be dissolved with sulphuric acid and examined further for these earths. To test the filtrate from (*a*) for ZrO_2, evaporate to dryness, ignite gently, dissolve in HCl, evaporate off the excess of acid, and transfer to an object glass with a few drops of water. Evaporate to dryness, add a drop of water and a drop of potassium binoxalate solution, again evaporate, and examine under the microscope for the characteristic crystals of zirconium-potassium oxalate. The presence of 0·1% of ZrO_2 may be thus detected in the solution from a single mantle. (3) Precipitate a part of the solution, after evaporating off the free acid, with oxalic acid, ignite, and weigh the precipitate, consisting of the total rare earths other than ZrO_2. The result is influenced by CaO, but for practical purposes, in a mantle consisting of say, 1·2% of CeO_2, 98·4% of ThO_2, and 0·4% of CaO, it does not matter if the CaO be reckoned as ThO_2. (4) Estimate ceria in an aliquot part of the solution volumetrically, noting that any organic matter extracted from the mantle necessitates the use of more persulphate than is mentioned above. On deducting this result from that at 3, the difference, in the presence of negligible proportions of Nd_2O_3, La_2O_3 and Yt_2O_3, gives the ThO_2.

522

NICKEL AND COBALT.

PRODUCTION ceased entirely at the Lancaster Gap mine in 1898, and the small domestic output was from matte from Mine la Motte, Mo.

UNITED STATES NICKEL STATISTICS.

Year.	Production.						Imports. (b)		Exports.	
	From Canadian Ore.				From Domestic Ore					
	Ni in Sulphide, Oxide, etc. (a)		Metallic.		Metallic.					
	Pounds.	Value.	Pounds.	Value.	Pounds.	Value.	Pounds.	Value.	Pounds.	Value.
1894......	(c)	(c)	(c)	(c)	9,616	$3,269	9,286,733	$310,581	1,235,588	$247,568
1895......	1,503,216	$300,643	1,175,445	$293,861	10,302	3,091	20,355,749	629,910	1,061,285	239,897
1896......	2,096,890	419,378	1,600,049	400,012	17,170	4,464	102,412	32,532	2,756,604	606,833
1897......	1,459,390	505,241	2,640,000	913,968	33,700	11,668	177,392	53,013	4,255,558	997,391
1898......	3,516,427	1,213,167	3,611,357	1,245,918	11,145	3,845	(d)	(d)	5,657,620	1,359,609

(a) The nickel reported as in oxide is now mostly converted into metal before consumption. (b) Includes nickel oxide, nickel alloys, in which the nickel is the chief element of value, and all forms of nickel, except manufactures. (c) The total product of fine nickel and nickel in sulphide, oxide, etc., was 4,223,115 lb., valued at $818,869. (d) None reported.

Commercial Conditions in the Nickel Industry.—(By Robert M. Thompson): "During 1898 the consumption of nickel largely increased on account of the demand for nickel steel for use in armor plate and other parts of war vessels being built by several nations. It has now been fully demonstrated that nickel steel is a material as much superior to any of the ordinary steels as steel itself is superior to wrought iron. Wherever the question of cost is not a controlling factor nickel steel must and will enter largely into commercial uses.

"The metallurgy of nickel remains a difficult problem, on which many skilled metallurgists are working. So far America has held its own in the race, and the nickel from the Canadian deposits at Sudbury, refined by the Orford Copper Co., holds to-day fully one half the market of the world. Certain parties in Canada have been agitating for an export duty on Canadian nickel, but as yet without success. Such a policy would of course be suicidal, and it would be an obstruction of the same nature as tearing up the railways connecting the mines and markets, and going back to the old turnpikes.

"All sorts of solutions of the nickel problem are being presented. Ludwig Mond and his corps of chemists have worked out a very beautiful solution in what is known as the 'Mond process,' in which nickel is converted into a gas and redeposited as a metal exactly similar in appearance to the metal deposited by the electrolytic process. Francis J. Clergue is proposing to smelt nickel ores in an electrical furnace, the power being furnished by turbines driven by water power at the Sault Ste. Marie The Canadian Copper Co. is working on a very promising electrolytic process. The Société 'Le Nickel' continues to employ its very successful process for treating the New Caledonia ores, and the Orford Copper Co. continues its well-tried and well-established process.

"The policy of the producers of nickel has been to make as little disturbance in the price of the metal as possible, and in the face of the very largely increased consumption prices have been but little advanced, and the large makers of steel are now satisfied that they can secure ample supplies at a reasonable and steady price. The danger in the situation is, as it is in all business, that competition may come into a market where supplies still far exceed the demand, and so bring on a scale of prices that will be wholly unremunerative. Still, the very large expenditure necessary for plant for any of the proposed processes, and the great skill required, make capital timid about entering into a field where the competition must be keen and the losses for all engaged in it great.

"No new sources of nickel supply were developed during 1898, though several 'discoveries' were announced. In most cases these finds have turned out to be valueless; in fact, many of them have no trace of nickel at all. There seems to be a great lack of knowledge with regard to nickel ores among prospectors and miners.

"The market for nickel is comparatively so small and the number of producers so limited that the fluctuations were small and prices remained about on the same level throughout the year. The closing quotations in New York were 33@36c. per lb. for ton lots, and 35½@38c. for smaller orders. The London price was 14@16d. per lb., according to size of order. This is about on a parity with the New York quotation, allowance being made for the duty on imports into the United States."

THE WORLD'S PRODUCTION OF NICKEL.

Year.	New Caledonia.				Canada.	Norway.	United States.		World's Total.
	Prussia.	France.	England.	Total.			Domestic	Imported	
1893	893	(a) 1,600		2,493	1,807	113	11	1,320	4,424
1894	522	1,545	855	2,422	2,226	103	4	1,920	4,755
1895	698	1,545	305	2,548	1,764	c103	5	1,220	4,420
1896	822	1,545	405	2,972	1,541	c103	8	1,685	4,624
1897	898	1,245	1,355	3,498	1,813	c103	15	1,859	5,429
1898	(b)	(b)		(b)	2,503	(b)	5	3,234	....

(*a*) Nickel contents of ore shipped from New Caledonia to France and England; the production of metallic nickel by the French works alone was much larger than this, amounting to 2,045 metric tons (accumulated stocks being drawn upon). (*b*) Statistics not yet reported. (*c*) Estimated.

NOTE.—The figures for France and Prussia in the above table are from the official reports of the government statistical offices; the amount credited to England is obtained by deducting the output in France, officially reported, from the combined output of the French and English works as given by the *Metallgesellschaft*, Frankfurt-am-Main. The actual output of the English works (meaning all in the United Kingdom) is somewhat greater, since there is a little Canadian ore smelted there. The Prussian figures, on the other hand, represent a slightly larger output than should be credited to New Caledonia, since there is a small amount of nickel of domestic origin included therein. These inaccuracies are not, however, of much consequence. The Canadian figures are those of the Geological Survey of the Dominion. The production of Norway is given as officially reported where figures are available. For purposes of comparison the production of metallurgical works in the United States is appended; it is not included in the world's total because all but a few tons of this metal has already been counted as Canadian. The world's production of nickel, as given in the above table, is probably overstated slightly, since Messrs. Basse & Selve, which is the most important nickel smelting firm in Germany, have produced from time to time some metal from oxide imported from the United States. Since the latter was derived originally from Canadian ores, its metal contents have been counted twice in the method of statistical computation employed here, once as Canadian and once as Prussian; for the same reason the total for New Caledonia, arrived at by adding the Prussian report to that which is smelted in France and England from New Caledonia ore, is correspondingly high The error is not, however, very important.

Canada.—According to the report of the Dominion Geological Survey the production of nickel in Canada in 1898 was 5,517,690 lb., against 3,997,647 lb. in 1897, the entire output each year being derived from the Sudbury district in Ontario. The detailed statistics of Ontario, compiled by A. Blue, Esq., director of the Ontario Bureau of Mines, are as follows:

Schedule.	1893.	1894.	1895.	1896.	1897.	1898.
Ore raised..................tons	64,043	112,037	75,439	109,097	93,155	123,920
Ore smelted................tons	63,944	87,916	86,546	73,505	96,093	121,924
Per cent. nickel................	2·21	2·92	2·67	2·67	2·08	2·28
Per cent. copper................	2·38	3·14	2·73	2·54	2·86	3·43
Per cent. cobalt	0·08	0·0721	....	....	....	....
Ordinary matte............tons	7,176	10,410	12,525	9,733	13,706	20,101
Bessemerized matte.......tons	452	1,470	103½	....	328	....
Nickel contents............tons	1,653	2,570½	2,315¾	1,948½	1,999	2,784
Copper contents...........tons	1,431	2,748	2,365½	1,868	2,750	4,187
Cobalt contents............tons	19	3¼	....	....	....	....
Value of nickel................	$454,702	$612,724	$404,861	$357,000	$359,651	$514,220
Value of copper................	115,200	195,750	160,913	130,660	200,067	268,080
Value of cobalt................	9,400	1,500	....	....	....	....
Wages paid......................	252,516	311,719	209,960	247,151	253,226	315,501
Men employed..................	495	655	444	485	537	609

(By A. McCharles): "Nickel mining in the Sudbury district made far more progress in 1898 than in any season during the previous five years. The Canadian Copper Co. opened two new mines, and now operates five mines in all. This company added two new furnaces to its smelting plant, which now comprises five furnaces and has a capacity of over 600 tons per 24 hours. In one of the new mines a fine body of high-grade ore has been found within a short distance from the surface. The ore in the other new mine also grades above the average, and the deposit, though not of unusual extent, is evidently one of the best that has so far been uncovered on the whole range. Thomas M. Kirkwood, of Sudbury, took hold of a nickel property in the township of Garson, on which he did considerable development work, erected a rock house and a lot of other buildings, and cleared the ground for a roast yard and a smelter. But lately he sold out, it is reported, to the Canadian Copper Co. This mine is one of the smaller class of deposits, but the ore in it is of very good grade, both in nickel and copper. The Sault Ste. Marie Pulp and Paper Co. completed the purchase of a fine nickel property in the township of Creighton, and is preparing to get out ore for the double purpose of obtaining sulphurous acid for use in its pulp mill and to experiment on a new electrolytic process for making nickel steel. There are three large deposits of ore on this property, high in nickel, but carrying very little copper on the outcrops. The ore has to be hauled 5 miles to the nearest siding on the railway and then shipped 170 miles to Sault Ste. Marie. The government diamond drill was engaged in testing the extent and character of the ore body in the Violet mine, which is a sort of adjunct to the Murray mine, both belonging to the Vivians of Swansea, Wales, but lying idle for some years. A good deal of preliminary development work was done on various other locations in the district in 1898, and several important new discoveries have been made, one of them being only 8 miles from Sudbury and 4 miles from the railway track.

"Experience of 11 years in this district has led me to the conclusions that (1) the workable ore bodies do not occur on the larger areas, but on the smaller tongues of diorite, which in some cases are no wider than the mineral outcrop; (2) with very few exceptions the big bodies are lower grade ore than the medium sized and smaller ones, and, other things being equal, the latter are therefore the most valuable mines; (3) the gold, platinum and other by-products in these nickel mines are carried, not by the pyrrhotite, but by the copper pyrites in the ore."

A new deposit of nickel was discovered in 1898 on Lots 11 and 12, Concession 9, of Calumet, Pontiac County, Que., and the Pontiac Development and Mining Co. was organized to exploit the property. Mr. J. Obalski reported, December 3, 1898, that the ore at this locality is disseminated through a diorite belt, and had been opened to a depth of 30 ft. for a width of about 12 ft. The ore is mostly pyrrhotite, but in some places small veins of niccolite, and probably also some cobalt ore, are found. One specimen showed nickel 3·33% and cobalt 0·35%, but it is said that other samples have gone higher in cobalt.

Chile.—Nickel and cobalt minerals are found associated with the ores of silver at several places in Chile. Among these are Mina Blanca de San Juan, Department of Freirina; Minillas, Cambillos Buitre, in the Province of Coquimbo; and Tajon del Yeso, in the Province of Santiago. The ores of the Colorado mine of Chanarcillo are nickel bearing. Veins of nickel ore are found also at Portezuelo del Carrizo, near Morado, south of Huasco; also at San Pedro, near Flamenco, a small port south of Chañaral.

Germany.—Explorations were continued in 1898 at the nickel mines in the upper Westerwald, in the Province of Hesse-Nassau, with satisfactory results, it is claimed. The ore is sulphide, of which the best is said to assay 4 to 5% Ni.

New Caledonia.—Nickel mining in New Caledonia increased in activity in 1898, and there was a large increase in the exportation of ore, which may be assumed to average about 7% Ni. The statistics of production and exports, in metric tons, are presented in the following table:

			PRODUCTION.			
Kinds.	1893.	1894.	1895.	1896.	1897.	1898.
Nickel ore.	69,614	61,243	29,623	6,417	26,464	(*a*)
Cobalt ore.	2,200	4,112	4,277	(*a*)	3,200	(*a*)
			EXPORTS.			
Nickel ore	45,614	40,089	38,976	37,467	57,439	74,614
Cobalt ore	520	4,156	5,302	4,823	5,395	2,373
Cobalt matte	169	7	*Nil.*	*Nil.*	*Nil.*	*Nil.*

(*a*) Not yet reported.

New nickel mines were opened in several parts of the Colony, and the Société "le Nickel" sent out engineers to report on the advisability of erecting smelting works under the new conditions. The Camden Syndicate of London had representatives investigating the resources of the island, and the Smelting Company of Australia made arrangements to buy nickel and copper ores for treatment at its works at Lake Illawarra, New South Wales.

Tasmania.—A discovery of nickel ore in connection with gold ore was made in 1898 in the 110 and 155-ft. levels of the Rocky River mine.

PROGRESS IN THE METALLURGY OF NICKEL.

The principal developments in the metallurgy of nickel in 1898 were in the perfection of Mond's nickel-carbonyl process and the designing of electrolytic processes.

The Mond Process.—About 10 years ago Dr. Ludwig Mond in collaboration with Dr. Carl Langer discovered that finely divided nickel has the property of removing carbon from carbon monoxide at a temperature of 50° C., the monoxide being converted to dioxide while the nickel combines with the carbon monoxide as nickel carbonyl, which is volatile at that temperature but upon heating is dissociated into carbon monoxide and nickel, the latter being deposited in a dense metallic film upon the side of the vessel in which the decomposition takes place. Iron forms a similar carbonyl, but no other metal has been found to do so. Upon this principle was developed a process for the recovery of nickel from its ores, which has been experimented with extensively and lately brought to a point where it may be considered an actual working process, about 80 tons of metallic nickel having been already extracted by it from different kinds of matte at the experimental works at Smethwick, near Birmingham, England. The latest details of the process were described, with the indorsement of Dr. Mond, by Prof. W. C. Roberts-Austen in a paper read before the Institution of Civil Engineers, November 8, 1898, of which the following paragraphs are an abstract.

The process is especially adapted to the extraction of nickel from ores which contain copper in addition to nickel and iron, such as those of Sudbury. Such ores are desulphurized, smelted to a matte, and bessemerized in the ordinary manner. The bessemerized matte is subjected to the following operations: (1) Dead roasting; (2) extraction of part of the copper by H_2SO_4; (3) reduction of the nickel and the remaining copper to the metallic state, care being taken to avoid reducing the iron; (4) volatization of nickel. From the volatilizer the ore is returned to the reducer (3) and continues to circulate between stages (3) and (4) for a period varying between 7 and 15 days, until about 60% of the Ni has been removed as carbonyl. The residue from this operation, amounting to about one-third of the original calcined matte, and not differing much from it in composition, is returned to the first operation and put through the same course as before. The nickel carbonyl produced in (4) passes to the decomposer (5), where the carbonyl is dissociated into CO and Ni, the former being returned to the volatilizer to take up a fresh charge of nickel.

In the practice at Smethwick the calcined matte, assaying 25·27% Ni, 41·87% Cu, and about 2·13% Fe, is crushed in a ball mill to pass a 60-mesh screen and treated in lots of 336 lb. in a lead lined mixer with 200 lb. H_2SO_4, diluted with 20 cu. ft. of mother liquor from a previous operation, at a temperature of about 85° C., for 30 minutes. The solution of $CuSO_4$ is drawn off for crystallization in the ordinary manner. A small amount of nickel is dissolved and accumulates in the mother liquor, wherefore the latter has to be replaced with fresh water from time to time, and the nickel recovered, which is done by evaporating to dryness, calcining the sulphates and adding the oxides to the ore going to the reducing tower.

The residue from the leaching, containing 52·5% Ni, 20·6% Cu and 2·6% Fe, is accumulated to 5 or 6 tons and then charged at the rate of 0·5 ton per hour into the reducing tower. The latter consists of a vertical tower 25 ft. high, containing a series of hollow shelves heated to 250° C., by producer gas inside of them. There are 14 of these shelves, of which only seven are heated, the five lower shelves being cooled by a stream of water to reduce the ore to the temperature at which the volatilizer is worked. The reducing tower has a central vertical shaft with rabbles which stir the ore on the shelves and move it from each shelf to the next lower. Water gas passes up the tower to effect the reduction of the ore. The water gas is generated by the decomposition of steam by anthracite in a gas-producer and is collected in a gas-holder. It contains about 60% H. The reducing operation is so regulated that the gas escaping from the reducer contains only 5 to 10% H. This waste gas is subjected to the action of a fine water-spray, which condenses the steam generated by the combustion of the hydrogen. Part of this waste gas is used for making the CO required in the volatilizer by passing it through a retort charged with incandescent charcoal, which reduces the CO_2 contained in it and produces a gas containing about 80% CO, which is stored in another gas-holder. To avoid the formation of iron carbonyl in the volatilizer the temperature in the reducer has to be kept very low. If this be done the nickel extracted from a matte assaying 6 to 10% F will not contain more than 0·5% Fe. However, if the amount of iron in the residue rises above 10% the extraction of the nickel is much delayed on account of the low temperature which must be maintained in the reducer; and in such a case it is necessary to resmelt the residues before proceeding with the extraction of the nickel and copper.

The volatilizing tower into which the ore passes from the reducing tower is similar to the latter, but the shelves are not hollow, not having to be heated, since a temperature of only 50° C. is required and the heat of the ore and gas is sufficient to maintain this. The amount of nickel extracted during the first treatment in the volatilizer is 61%; the residue which is retreated amounts to 33% of the original ore and assays 35·48% Ni, 38·63% Cu and 4·58% Fe. The retreatment brings the recovery up to 80% and the residue from it, amounting to only 10% of the original ore, assays 35·83% Ni, 35·56% Cu and 7·82% Fe.

The carbon monoxide charged with nickel from the volatilizer passes through a filter to separate the matte-dust and into an apparatus called the decomposer. This is a vertical cylinder, with a conical, hopper-shape bottom and a central vertical tube, extending nearly to the bottom inside, through which the nickel carbonyl gas is admitted. Surrounding the cylinder is a wrought-iron jacket, forming an annular chamber, which is divided into flues through which hot air is passed to maintain in the cylinder the temperature of 200° C. required for the decomposition of the carbonyl, proper arrangements being provided for the regulation of the temperature. In order to prevent deposition of nickel in the central tube it is cooled by a current of water through a smaller pipe inside of it. The cylinder itself is filled with granules of nickel, through the interstices of which the gas is drawn from apertures in the central supply pipe. The nickel of the carbonyl is deposited on the granules, which consequently increase in size. In order to prevent them from cohering they are kept slowly moving by continuously withdrawing them from the hopper by means of a screw conveyor, which delivers them into a sifting drum, whence the undersize is returned automatically to the top of the cylinder. The CO gas escaping from the cylinder, having been deprived of its nickel, passes to a blower, which returns it to the volatilizer. The nickel granules of the proper size are ready for any desired use. Two assays of them showed: 99·82% Ni, 0·10% Fe, 0·0068% S, and 0·07% C; and 99·43% Ni, 0·43% Fe, 0·0099% S, 0·087% C, and 0·026% insoluble residue.

Electrolytic Process of Nickel Winning.—Titus Ulke devised a process for the treatment of Sudbury nickel-matte, based on his own experiments and those of Dr. Foerster, which he describes as follows: This process consists in depositing electrolytically commercially pure nickel, in any desirable form and thickness, from a neutral, or nearly neutral, nickel sulphate solution, heated to at least 120° F. in a preferably steam-jacketed tank. The general arrangement of an electrolytic nickel refinery of this type is similar to that of an up-to-date copper refinery. The principal difference is in the construction of the tanks, which in the nickel refinery are equipped with coils of steam pipes, either immersed in the electrolytic solution or placed alongside of the walls of the tank in a separate water jacket. In this way the temperature of the hot electrolyte may be kept fairly constant. Each tank has its separate valve and connection with the pipe leading from the steam generating plant and with the return pipe carrying back steam and condensed water. In nickel refining a higher voltage and a greater amperage are required than in the electrolytic refining of copper. An electrolytic nickel refinery, however, may be operated in much the same way as a modern copper refinery, and should have about the same appliances. The precious metals are partially recovered as by-products, while the nickel is deposited as a commercially pure and forgeable solid metal. The cost of this electrolytic process should not exceed 2c. per lb. of crude nickel treated.

Preparation of Nickel Sulphate.—T. Savage proposes to prepare nickel sulphate by digestion of nickel or nickel ore in sulphuric acid with addition of sodium nitrate in the following proportions: 4 parts sulphuric acid, 2 parts nickel, and 1 part sodium nitrate, the acid being diluted 1:2. The mixture is heated by steam pipes and the nickel sulphate crystallized out from the saturated solution.*

Electric Smelting of Nickel Ores.—W. S. Horry gave an account of experiments for the electric smelting of nickel ores at Sault Ste. Marie. The ore was obtained from Sudbury and contained 3% Ni and 42% S. It was roasted down to about 7% S, then pulverized, mixed with lime and carbon, and charged into an electric furnace, heated by an alternating current, wherein it was reduced to ferronickel. Two types of furnace were used, giving widely different results. One had a crucible, which served as an electrode, and a carbon pencil movable vertically as the other pole. This pencil was drawn up as the smelting proceeded and as more mixture was added. When the crucible was full

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* English Patent No. 9,099, April 9, 1897.

the current was shut off and the fusion allowed to cool. The mass of metal obtained at the bottom of the pot contained 40% Ni, 28% Fe, 12% C, 15% SiO_2, no sulphur, and 5% undetermined. The second furnace was continuous in principle. There were two electrodes side by side with a bridge between them. The reduced metal was drawn off as fast as it was formed. In this furnace the smelting took place rapidly, but the metal obtained assayed only 7% Ni and was high in carbon. It was found to be stronger than ordinary cast iron and so hard that it could be used as a cutting tool in a lathe, cutting rough iron castings with ease.

Assaying Nickel Ores.—The method used in New Caledonia, devised by Mr. Moore, the official analyst to the government of the colony, is as follows: Make two solutions. *A*: Cyanide of potash, 11 g. (take the ordinary 98% of commerce); nitrate of silver, 0·5 g.; water, 1 liter. *B*: Citric acid, 50 g.; carbonate of soda, 38 g. (approximately); potassium iodide, 7·5 g.; water, 500 c.c. The citric acid and the carbonate of soda must be in such proportions as to produce a neutral or very slightly alkaline mixture; the above weight of carbonate of soda is only approximate; 35 g. or so may safely be used at once, the remainder added decigram by decigram until neutrality is obtained before adding the potassium iodide. It is very important that this solution be either absolutely neutral or only very slightly alkaline.

The assay is made as follows: (1) Weigh 2·5 g. of the dried ore into a small flask having a 250 c.c. mark in the neck. The ore must be very carefully dried at 100° C.; at 120° the water of hydration begins to escape, as shown by the darkening color of the sample. The ores contain from 15 to 25% hygroscopic moisture. (2) Dissolve in 20 c.c. hydrochloric acid. The ore readily decomposes on boiling, leaving the silica beautifully white. In most samples a few shining black particles are to be seen in the insoluble residue. They are chrome iron ore and need not be decomposed. (3) Add cold water to produce 250 c.c. of liquid. Thoroughly mix by agitation. (4) Filter. The filter paper is not to be previously moistened. (5) Take 50 c.c. of the filtrate (equal to 0·5 g. of the ore) by means of a pipette, transfer it to a clean 200 c.c. beaker. It is not necessary to wait until the whole of the 250 c.c. of liquid has passed the filter. As soon as sufficient has run through to yield 50 c.c. proceed. (6) Add 10 c.c. of solution *B* to the 50 c.c. of filtrate. The office of this solution is to prevent the precipitation of the iron on adding dilute ammonia in the next stage. (7) Neutralize with dilute ammonia, and then add a drop or two in excess, or sufficient to produce the characteristic blue coloration. Cool the solution. Use the ordinary ·880 liquid ammonia, diluted with double its bulk of water. It is important that the assay be quite cool before proceeding, and that no more dilute ammonia be used than is sufficient to produce the bluish tint. (8) Titrate with solution *A* gently, and stirring all the time. At first a white cloudy precipitate forms, which should suddenly disappear by the addition of the last drop of solution *A*. The standard solution of pure nickel is prepared and titrated in the same manner as the ore. On highly ferruginous ores the cloudy precipitate is likely to reappear after being cleared. If a not too excessive quantity of iron be present continue the titration until permanent clearing.

The process is not applicable to ores containing large quantities of iron, manganese or cobalt, 25% being the limit for iron and manganese and 1% for cobalt. The whole process takes but 30 minutes or so and 20 or 30 assays can be run through in less than two hours. Results are very concordant, but are apt to be a trifle higher than the truth, small fractions of cobalt being included in the nickel.*

COBALT.

Cobalt, which is used in the arts chiefly in the form of oxide, is obtained from ores derived for the most part from New Caledonia, Australia and Germany, and smelted in France, Germany and Great Britain, the Messrs. Vivian, of Swansea, being the chief buyer in the last-mentioned country. The production of cobalt oxide in Europe is controlled by a combination, which, by virtue of the limited extent of the industry, is able to maintain a firm grip on it. Cobalt oxide is produced in the United States by one works. The domestic production and importations are summarized in the following table:

UNITED STATES PRODUCTION AND IMPORTS OF COBALT OXIDE.

Year.	Production	Imports.		Year.	Production	Imports.	
	Pounds.	Pounds.	Value.		Pounds.	Pounds.	Value.
1893	8,422	28,164	$41,105	1896	12,825	27,189	$36,212
1894	6,763	24,020	29,857	1897	19,300	24,771	34,773
1895	6,400	36,155	39,839	1898	9,640	33,731	49,245

The value of the cobalt oxide produced in the United States in 1898 was \$1·60 per lb. at the works. The value of cobalt ore containing a minimum of 4% CoO in England in 1898 was 38s. per unit of a long ton (*i.e.*, 22·4 lb.) with a rise of 6d. per unit for every tenth of a unit over 4%, *e.g.*, an ore assaying 4·2% CoO would be worth 39s. per unit. However, no large increase over the present supply could be marketed without lowering the price.

New Caledonia.—The cobalt ores of this island occur usually in the vicinity of the nickel ores, in deposits of irregular shape and size, with no definite boundaries in any direction. The ore is cobaltiferous wad, containing 25 to 30% Mn and 2 to 8% CoO. Wad and limonite occur in extensive areas of chrome ore and they are sometimes cobaltiferous and sometimes not so, wherefore prospecting is not easy, since there is no clear distinction between the two kinds, which pass gradually into each other. Cobalt ore sold in New Caledonia in 1897 at 38s. 6d. per unit of cobalt protoxide for a minimum grade of 4% CoO, freight and charges prepaid to any European seaport, with a variation of 6d. for each one-tenth of a unit in excess of the minimum guaranteed grade. Ores of less than 4% protoxide were not considered worth shipping. This corresponds to \$7 per unit of 20 lb., f. o. b. in New Caledonia, allowing 28s. per ton for freight and commissions.

New South Wales.—J. B. Jaquet described the new deposits of cobalt ore at

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* H. W. Edwards, *Engineering and Mining Journal*, May 28, 1898.

Port Macquarie. Cobalt appears to have been found first at this place about 11 years ago, but the promising deposits were discovered only recently, by Messrs. Wyborn and Tellefson. The ore occurs in nests or pockets, possessing no defined form, which are scattered irregularly through clay and decomposed serpentine. The most promising bodies have been found at the base of a red ferruginous clay stratum and in the upper layer of the underlying serpentine which is decomposing *in situ*. The ore is an earthy cobalt-bearing wad (asbolite) which sometimes contains as much as 32% Co. It has a bluish black color. Generally speaking the bluer the ore the higher its tenor of cobalt, and this characteristic is made use of in sorting. An average sample of a few tons of picked ore gave the following result: 4·98% hygroscopic water, 12·21% combined water, 8·06% SiO_2, 18·95% Al_2O_3, 14·78% Fe_2O_3, 31·05% MnO_2, 7·48% CoO, 1·36% NiO, 0·41% Cr_2O_3, 0·05% CuO, 0·05% CaO, 0.06% P_2O_5—total, 99·44%. Both as regards the character of the ore and its mode of occurrence there is a close resemblance to the cobalt ore of New Caledonia. The Port Macquarie deposits are handy to water carriage and the ore can be cheaply shipped although it is, comparatively speaking, light and bulky. The prospectors were offered £5 10s. per ton for ore containing 5% CoO delivered in Sydney. In Mr. Jaquet's opinion the deposits at Port Macquarie are worth exploration. Cobalt wad has also been found in New South Wales at Carcoar and Bungonia, from which places a small production has been made.*

South African Republic.—The exploration of the cobalt deposits about 25 miles north of Middleburg, owned by the Henderson's Transvaal Estates Co., which were small producers in 1889-90, was resumed in 1898, and a good deal of ore assaying 7 to 10% Co is reported to have been exposed.

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* *Australian Mining Standard*, Feb. 17, 1898.

OCHER AND OXIDE OF IRON PIGMENTS.

UNDER this caption we have grouped all the pigments, natural and artificial, in which oxide of iron is the essential coloring matter. A part of the natural pigment of this class is known in the trade as "metallic paint" and a part as "ocher," the chief difference between these substances being in the amount of ferric oxide contained in them. Since, however, these pigments pass into each other by insensible gradations it is impossible to pronounce definitely what is the dividing line. There are some natural pigments which consist almost wholly of ferric oxide, while there are others sold in the trade under the name of "metallic paint," which contain only 67·30% Fe_2O_3, and others with as little as 33%, these being analyses of samples purchased from dealers. Under these circumstances the classification we have adopted seems the most rational. Our statistics of Venetian red include only the pigment which is made by the calcination of copperas, no distinction being made between Indian red and Venetian red proper; the former is pure ferric oxide, while the latter contains some lime which is mixed with the copperas in the calcination. Ocher is usually yellow, but some kinds are red or brown. Umber and sienna differ in composition from ocher by containing manganese in addition to ferric oxide. Raw umber is of a brown color, while burnt umber has a somewhat richer and redder hue than the raw. Raw sienna is of a brownish yellow shade, affording a rich russet brown when burned. Intermediate shades are obtained by mixing natural products with varying proportions of iron and manganese oxides, and, if necessary, by mixing burnt with raw mineral.

The domestic production of ocher and oxide of iron is obtained chiefly from New York, Pennsylvania, Tennessee, Vermont and Georgia, the kinds rich in iron coming especially from New York, Pennsylvania and Tennessee. Umber and sienna are produced chiefly in New York, Missouri and Pennsylvania, the last mentioned State being the most important. A new deposit of sienna was opened in 1898 by the National Paint Co. in Big Cottonwood Cañon, Utah, the product being prepared for the trade in Salt Lake City. A natural pigment resembling umber, called Spanish brown, is produced in Maryland. A substance resembling yellow ocher, which accumulates under the salt tanks at East Tawas and elsewhere in the Michigan salt region, is now being utilized as a pigment. New deposits of yellow ocher near Harriman, Tenn., were reported in 1898. The Cherokee Ocher and Barytes Co., of Cartersville, Ga., erected a plant at that place for the preparation of its product by grinding and levigation. This ocher is of a rather dark color and on that account has not yet been received favorably in the paint trade, though its strength and covering capacity are claimed to be high, and so far it has been used chiefly in the linoleum and oilcloth trade. The Peruvian Ocher Co., of Emerson, Ga., has gone into the hands of a receiver, by whom the business is to be continued. In March, 1899, a combination of the principal producers of metallic paint was reported, including the Prince Manufacturing Co., Alpha Paint and Mineral Co., New York Metallic Paint Co., Bass Paint Co., of New York, Rutherford & Barclay, of Philadelphia, and C. K. Williams & Co., of Easton, Pa.

The production of ocher, umber and sienna and oxide of iron pigment in the United States in 1898 was 41,950 short tons ($461,450), against 42,590 ($468,490) in 1897; of Venetian and Indian red, 3,270 tons ($64,570), against 2,196 ($40,790). The values of ocher, etc., are reckoned nominally at $11 per ton each year.

IMPORTS OF OCHER, UMBER AND SIENNA INTO THE UNITED STATES.

Year.	Ocher of All Kinds.						Umber.		Sienna.			
	Dry.		Ground in Oil.		Total.				Dry.		Ground in Oil.	
	Pounds.	Value.	Pounds	Value.	Pounds.	Value.	Pounds.	Value.	Pounds.	Value.	Pounds	Value.
1891	6,246,890	$63,040	52,206	$5,272	6,299,096	$68,312	633,291	$6,498	....	....	....	....
1892	8,044,836	97,946	49,714	5,120	8,094,550	103,066	1,028,038	6,256	....	....	....	....
1893	6,225,789	55,074	52,468	3,354	6,278,257	58,428	1,488,849	16,636	1,626,536	$138,889	5,857	$610
1894	4,937,738	45,276	22,387	2,100	4,960,125	47,376	632,995	6,275	337,909	9,424	18,877	895
1895	7,107,987	56,020	41,153	2,239	7,149,140	58,259	*b*1,560,786	13,075	456,861	11,021	6,576	501
1896	8,954,252	68,196	27,023	1,561	8,981,275	69,757	*c* 689,075	8,360	(*f*)	(*f*)	(*f*)	(*f*)
1897	*a*7,720,075	59,272	20,123	1,000	7,740,198	60,272	*d*1,447,889	14,479	580,468	12,340	7,058	481
1898	5,898,725	46,571	31,460	1,546	5,930,185	48,117	*e*1,123,079	9,051	544,713	11,451	4,008	280

(*a*) Includes 241,452 lb. entered as crude and 1,416,587 lb. as powdered. (*b*) Includes 6,137 lb. ground in oil and 1,554,649 lb. dry. (*c*) Includes 5,292 lb. ground in oil and 683,783 lb. dry. (*d*) Includes 14,471 lb. ground in oil and 1,433,418 lb. dry, crude or powdered. (*e*) Includes 4,608 lb. ($323) ground in oil and 1,118,471 lb. ($8,728) crude, powdered, washed or pulverized. (*f*) None reported.

THE PRODUCTION OF OCHER, SIENNA AND UMBER IN ITALY.

The world's supply of sienna is derived largely from Italy, the substance taking its name from a city in that kingdom, although none is now obtained near that point. In Italy, according to the *Chemical Trade Journal*, the storing and manipulation of sienna earths is centered at Leghorn, where the trade is chiefly in the hands of Enrico Ganni & Co. These earths come from a district close to Monte Amiata, in the Province of Grosseto. The yellow ocher is a surface earth, the sienna lying beneath it in three strata, with the third quality uppermost and the first quality undermost. The last yields an extra fine grade by careful selection by hand, this selected earth being known locally as "spurgo di bolo" or "giallone."

At the pits the raw material is exposed to the sun after digging, so that it may lose some of the large amount of water which it contains. Work is carried on only in the summer months, and a rainy season interferes seriously with operations. Burning is also extensively practiced at the place of excavation, at all times of the year, charcoal being the fuel employed, but this is also done at the works in Leghorn. The cost of this process is trifling. The principal operation at the Leghorn works is the grinding of the raw material to an impalpable powder. The pigments leave Leghorn in four different forms, namely, raw for burning and grinding at place of destination, burnt for grinding, raw already ground, and burnt already ground.

The processes in Messrs. Ganni's works are extremely simple. Such of the raw material as is to be ground unburnt is first dried in kilns by a gentle heat. Sun-dried earths and ochers still contain 20 to 25% of water, and even in kiln-dried about 5% of water remains, the percentage always being specified in the invoices of shipment. The material to be calcined is burnt in brick kilns or on brick hearths by the application of intense heat. The raw or burnt material ready for grinding is then crumbled in the rough grinder, and when thus reduced to quite small proportions is placed in the final grinders and ground by huge granite stones.

The Turkey umbers come from Cyprus in large sailing ships at cheap rates of freight, and are warehoused at low cost. Turkey umbers are of three principal kinds: light color or limasol; dark or larnaca; very dark, commonly called black. It is always the unburnt material which is imported. The umber is subjected to very careful sorting before being burnt or ground. There is an import duty of 4 lire per 100 kg., but by special regulation the umbers may be taken out of bond for manipulation, and if exported the duty is waived. The process of burning is somewhat different from that generally adopted for earths and ochers. The umber is kneaded into rounded masses about the size of a melon and placed in a different kiln upon iron grids. This method is said to improve the color. Sometimes, however, the umber is merely kiln-burnt like other earths and ochers.

It is difficult to convey an idea of the price of these different articles, owing to the great variety of quality and shades of color. During 1897 raw siennas fetched £4@£12 per ton; raw ochers (unwashed), £3 10s.@£5 10s.; washed, from £8@£10; raw umbers, £2 per ton. Burnt siennas of the three principal qualities (known in Italy as "chicco," "gripolo," and "polvere"), £4@£20 per ton; "spurgo di bolo," or superior quality, a small and special business, £60 per ton; ground siennas (both raw and burnt), including all qualities, £6@£30 per ton; and ground "spurgo di bolo," £70 per ton. Burnt ochers, £4@£10, and ground ochers (raw and burnt), £10 and more per ton. Burnt umbers, £3, and ground umbers, £5 10s. per ton. The reason why there is little difference between the raw and burnt ground material, though the latter has had to bear the cost of burning, is that the dearer qualities of raw ochers and siennas are usually reserved for grinding.

Italian siennas easily take the lead of all others for body, color, strength and durability. The absolute purity and genuineness of their character, still legible in the pictures of Italy's old masters, may be safely guaranteed in all acquisitions made in Leghorn. Local traders have to compete in some markets with a German article which appears to be a compound of German earth with the genuine sienna impalpable pigment, but nothing can well hope to dislodge Italian siennas from the English color works.

THE DURABILITY AND TESTING OF IRON PIGMENTS.

The value of iron pigments as a preservative for exposed ironwork has been studied by H. Baucke.* His tests were made upon a sample containing 63·94% ferric oxide, 7·07% volatile matter, 10·07% matter insoluble in chlorhydric acid and 18·92% of other constituents, these percentages being of the sample after drying; the original sample had 7·08% moisture. This sample was heated in a muffle for different periods and at different temperatures, so as to prepare a series of pigments. The color of the product changed gradually from brown to red and finally to a dark purple, progressively according to the time of furnacing and the temperature. No. 2 was brown, No. 3 pale brown, No. 4 red, No. 5 purplish red, No. 6 purple, and No. 7 deep purple. These products were treated with 1% and 5% sulphuric acid in the cold, being uniformly and periodically agitated for 200 days, while the percentage of metal dissolved was estimated at regular intervals. Each sample was also separated into five portions of different degrees of fineness and the action of acid observed. The results showed that iron oxide becomes less hygroscopic the longer it is furnaced. The organic matter is quickly removed at the outset. The insoluble matter increased greatly at the change from No. 2 (brown) to No. 3 (pale brown). The specific gravity rose in two leaps between No. 2 and No. 3 and between No. 3 and No. 4. The proportion of ferric oxide rose largely from No. 1 to No. 2, and then gradually to No. 7. The acid-resisting power increased with prolonged heating, but the most noteworthy alteration occurred between Nos. 3 and 4, Nos. 5, 6 and 7 behaving very similarly. The specific gravity increased from No. 1 to No. 6, that of No. 7 being the same as No. 6. The specific gravity appeared to be a good indication of the industrial value of these pigments, and having regard for cost as well as quality, the cost of the pigment increasing naturally with the calcination, a specific gravity of 4·2, corresponding to the red oxide, may be assumed to mark the best material. The size of the particles made no difference with respect to the acid solubility.

H. Loesner has patented a method for increasing the moisture-resisting power of pigments, which consists in grinding the pigments in linseed oil in the presence of a certain proportion of hydraulic cement or cement diluted with sand. Precautions are taken to prevent the cement from absorbing water before the paint is applied to the iron work, so that when the film is finally exposed the new ingredient is caused to set slowly by means of the moisture in the atmosphere, producing at last a thin layer of hardened cement, or cement mortar, embodied in a paint of ordinary composition. The incorporation of the cement is said not to interfere with the proper spreading of the pigment, which is claimed to be absolutely damp-proof and permanent.† Loesner determines the moisture-resisting power of pigments by applying the sample under examination in a thin film to pieces of sheet iron thoroughly cleaned by polishing with emery cloth and setting them aside to dry for four days, after which a second coat is applied and allowed to harden for the same period of time. The plates, painted side downward, are then

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* *Zeits. f. Anal. Chem.*, 1898, 37, p. 668.

† English Patent No. 28,484, 1897.

laid on strips of wood over a vessel in which water is boiling vigorously, arrangements being provided to maintain constantly the level of the boiling water 50 mm. below the plates, which are consequently completely surrounded by the rising steam. After 15 hours' boiling the paint is dried at 100° C., removed from the iron by the aid of aniline and a brush, and the surface of the metal carefully examined for spots of rust. One hour of this test is regarded as equivalent to two or three months' exposure to wind and rain, and 12 hours is taken as the minimum resistance for a good paint, since scarcely any composition will withstand the steam longer, except the patented material described above.

To test the coloring power of a pigment, say red oxide of iron, the following method has been recommended: One part of the color to be tested is placed on a sheet of glass, under which is a sheet of black paper, and mixed with three parts of raw linseed oil. A similar quantity of a standard pigment is treated in a similar manner. Then to each sample add more oil, drop by drop, spreading out the pigment at each addition until each one shows signs of losing its power of hiding the black paper. The amount of oil used in each case will indicate the relative covering power, since the pigment which requires the largest amount of oil to spread it out to its greatest tenacity without losing opacity is the one which possesses the greatest body or covering power. The respective areas of each sample will also relatively measure their covering power. If the standard pigment requires five parts of oil and that which is being compared with it takes only three, the relation is 6:10; in other words, it would require 10 lb. of the pigment under examination to cover the same area as could be covered by the 6 lb. of the standard.

To test red oxide for color intensity, take a sheet of unglazed note paper and scratch a straight line on it with the point of a pin; on one side of this line a quarter of an inch away from it place a small heap of the standard and on the other side a similar heap of the pigment to be examined. Place the edge of a long-bladed flexible palette knife on the line and flatten out the heap of color which lies to the right, using the knife in such a way that the surface of the color remains smooth and intact; do the same with the heap to the left. A careful examination of the two colors will enable any one who is not deficient in color sense to discriminate between the two samples.

To test red oxides for coloring power, mix equal amounts of the sample to be examined and the standard with equal amounts of china clay in a wedgewood mortar, or on a palette. The sample which has the greatest coloring power will be the darker. To ascertain the relative power, mix china clay with the darker until it is reduced to the same color as the inferior sample. The comparison of the strength can then be made by the extra amount of china clay required.

PETROLEUM.

THE year 1898 was noteworthy for the large falling off in the production of the Appalachian and Lima fields and the rapid development of the new field in Texas. The statistics reported below are in general based on pipe-line receipts.

PRODUCTION OF CRUDE PETROLEUM IN THE UNITED STATES. (BARRELS OF 42 GAL.)

Year.	Appalachian Field. (*a*)	California. (*b*)	Colorado.	Indiana. (Lima Field.) (*c*)	Kansas.	Ohio. (Lima Field.)	Texas.	Wyoming.	Other States.	Total.
1894........	30,622,336	600,000	803,000	3,688,666	40.000	13,891,795	None.	2,300	2,000	49,650,097
1895........	30,406,698	1,245,339	530,000	4,386.132	44,000	18,415,631	None.	3,500	1,800	55,033,495
1896........	33,455,571	1,257,780	400,000	4,659,290	113,000	15,362,176	None.	2,878	3,600	55,254,795
1897........	34,724,700	1,911,569	650,000	4.353,138	(*c*)90,000	15,307,376	65,000	15,000	(*d*) 8,000	57.124,783
1898........	31,100,360	2,249,088	650,000	3,751,307	88,000	13,377,590	544,620	3,500	(*d*)10,000	51,774,465

(*a*) Includes, New York, Pennsylvania, West Virginia, and part of Ohio. The statistics of production in this field and the Lima field in Ohio are based on pipe line receipts as reported by the *Oil City Derrick*. (*b*) Statistics of California State Mining Bureau for 1895, 1896, 1897 and 1898. (*c*) Statistics of State Geological Survey. (*d*) Estimated.

The total net stocks of oil in the hands of the various pipe-line companies, in barrels of 42 gal., December 31, were as follows:

Appalachian.				Ohio-Indiana.			
1895.	1896.	1897.	1898.	1895.	1896.	1897.	1898.
5,161,905	9,550,583	10,789,652	11,541,753	21,474,848	23,302,779	22,762,779	15,180,892

Exports.—The movement of crude oil was mainly to France, which country took three-quarters of the total crude; the other European countries taking about one-eighth, and Mexico and Cuba most of the remainder. Of the refined products, including naphthas, illuminating oil, lubricating oil, etc., Great Britain, France and other European countries, as usual, received by far the greater part, this being mostly in the form of kerosene. Exports of residuum were relatively unimportant. The total export trade was very slightly below the point attained in 1897, the record year, and would have greatly exceeded it but for the Spanish-American war.

EXPORTS OF MINERAL OILS FROM THE UNITED STATES. (IN GALLONS.)
(1 = 1000 in quantities and values.)

Year.	Crude Petroleum.		Naphthas.		Illuminating.		Lubricating and Paraffine.		Residuum.		Totals.	
1894.....	114,269	$4,617	14,832	$904	726,727	$29,799	38,975	$5,137	119	$10	894,922	$44,463
1895.....	116,108	6,286	12,922	1,000	686,006	43,540	47,876	6,239	170	15	863,082	57,131
1896.....	118,133	6,032	13,641	1,123	758,076	49,704	51,705	6,770	521	28	942,076	63,657
1897.....	121,864	5,044	13,704	1,020	804,446	46,876	52,679	6,732	12,247	335	1,004,941	60,007
1898.....	120,436	5,019	17,255	1,071	764,823	38,895	65,526	7,626	30,436	815	998,476	53,423

Prices.—As will be seen by the subjoined table, the range in prices in 1898 was considerable, with a strongly marked upward tendency in the fall and early winter months, owing to lowering of stocks under heavy demand for export. As compared with recent years, the market may be said to have been satisfactory to producers. The figures quoted are for regular pipe-line stock and do not, of course, represent special and exceptional cases, as for example natural lubricating oil on the one hand and low-grade heavy oil on the other.

MONTHLY AND YEARLY AVERAGE PRICE OF PIPE-LINE CERTIFICATES PER BARREL OF CRUDE PETROLEUM AT THE WELLS IN THE APPALACHIAN FIELD.

Year.	Jan.	Feb.	March.	April.	May.	June.	July.	Aug.	Sept.	Oct.	Nov.	Dec.	Yearly Aver'ge
1893...	$0·53¼	$0·57⅝	$0·65¼	$0·68⅝	$0·58¼	$0·60¼	$0·57⅛	$0·58⅞	$0·64⅛	$0·70⅛	$0·73⅞	$0·78¼	$0·64
1894...	0·79½	0·80⅝	0·82	0·84¼	0·86	0·89⅜	0·83¼	0·81	0·83	0·83	0·83	0·91⅛	0·83¾
1895...	0·99	1.04¾	1.09¾	1·79	1·74¼	1·53¼	1·46⅝	1·26⅛	1·22⅝	1·24½	1·48⅛	1·42	1·25¾
1896...	1·45¾	1·39	1·33¼	1·22⅝	1·10⅜	1·14¾	1·09½	1·05	1·12⅛	1·15½	1·15½	0·97½	1·19
1897...	0·88	0·90½	0·92⅛	0·85½	0·85¾	0·86¼	0·76¼	0·71	0·69⅜	0·67⅞	0·65	0·65	0·78¾
1898...	0·65	0·67¾	0·78¾	0·73⅜	0·82¼	0·87½	0·93¼	0·97⅜	1·01⅜	1·13⅛	1·16⅝	1·17⅜	0·91½

AVERAGE MONTHLY PRICES OF CRUDE OIL IN THE LIMA FIELDS. (*a*).

Month.	1895.			1896.			1897.			1898.		
	North Lima.	South Lima.	Indiana	North Lima.	South Lima.	Indiana	North Lima.	South Lima.	Indiana	North Lima.	South Lima.	Indiana
January ..	$0·56½	$0·51½	$0·51¼	$0·89	$0·79	$0·76½	$0·58	$0·53	$0·53	$0·46	$0·41	$0·41
February .	0·61¼	0·56½	0·56½	0·85	0·75	0·75	....	....	....	0·50	0·45	0·45
March	0·67½	0·62½·	0·60	0·83¾	0·73¾	0·73¾	0·57	0·52	0·52	0·58	0·53	0·53
April......	1·01	0·98	0·88¾	0·73¾	0·65¾	0·65¾	0·51½	0·47½	0·47½	0·54½	0·49⅜	0·49⅜
May	0·85¾	0·83⅞	0·70¼	0·66⅞	0·61¾	0·61¾	0·51	0·46	0·46	0·59	0·54	0·54
June......	0·77	0·75	0·66½	0·64	0·59	0·59	0·49	0·44	0·44	0·64⅛	0·59⅜	0·59⅜
July.......	0·74	0·72	0·64	0·60½	0·55¼	0·55¼	0·48	0·43	0·43	0·67½	0·62¼	0·62¼
August ...	0·69	0·67	0·60	0·60	0·55	0·55	0·47	0·42	0·42	0·71⅜	0·66½	0·66⅜
Septemb'r	0·65	0·63	0·56	0·63	0·58	0·58	....	....	....	....	....	....
October...	....	....	....	0·65	0·60	0·60	0·46	0·41	0·41	0·76	0·71	0·71
November	0·82¾	0·79¾	0·73¼	0·64⅜	0·59⅜	0·59⅜	....	....	....	0·77	0·72	0·72
December.	0·90	0·80	0·75	0·61	0·56	0·56	....	....	....	0·79	0·74	0·74
Average...	$0·75⅝	$0·71⅛	$0·65¼	$0·69⅝	$0·63½	$6·62⅞	$0·51⅛	$0·46⅛	$0·46⅛	$0·63¾	$0·58⅞	$0·58⅜

(*a*) Based on reports of the *Oil City Derrick.*

California.—The known oil-bearing territory in southern California has been extended, even under the ocean, as at Summerland, where wells are driven 200 ft. from shore. The California oil is of a heavy, asphaltic character, and is mostly used in the crude state for fuel. The next important use is in the preparation of a distillate, also used as fuel and for gas making. As by-products, illuminating oil, gasoline and lubricating oil are produced, the kerosene being mixed with Eastern oil. The residue is an asphalt which has a limited market for paving on the Pacific Coast, efforts to introduce it elsewhere having been met by the drawbacks of high freights and competition with the Trinidad article. The local market cannot absorb the abundant supply of crude oil and its possible products. In 1898 there was continued activity in the Los Angeles, Santa Fe, Newhall and the Ventura and Santa Barbara fields. The only noteworthy success in the Los Angeles field was on the Maltman tract, where several shallow wells are remunerative. The only really new wells in Los Angeles County were those of the Home Oil Co., near Whittier. Oil has been struck in paying quantity at Piru City. The product of the wells in Los Angeles City decreased, and prospecting for new wells within the city limits did not meet with much success. The Los Angeles Oil Producers' Trustees organization ceased to exist at the end of 1898, its work having been taken up by the Oil, Storage & Transportation Co., its successor.

Colorado.—The output of the Florence field in 1898 was about the same as in the previous year—650,000 bbl. The oil territory has been extended westward by new finds. The petroleum is found at a depth of from 1,200 to 2,200 ft. in black shale, which is from 500 to 1,500 ft. thick. This oil has a sp. gr. 31° B., and yields about 35% illuminating oil, 12% naphtha and gasoline, and 53% residuum. There is considerable demand for crude petroleum and residuum as fuel for roasting ores prior to smelting, cyaniding and chlorinating, and for steam raising.

Illinois.—A small production of natural lubricating oil comes from a few wells in Montgomery County, ranging from 200 to 1,000 bbl. yearly. This oil brings a high price—$3 to $5 per bbl., according to the market.

Indiana.—In 1898 there were 1,102 wells completed, at a cost of $1,763,200, or $1,600 per well. Of these, 344 were "gassers" and classed as dry holes. For 1897 the corresponding figures were 1,041 new wells, costing $1,500 each, of which 280 were dry. The total number drilled up to the close of 1898 is 10,566, representing an outlay of $16,905,600; and of these 3,348 were dry. The average daily output was 22,899 bbl., or 4,460 bbl. less than in 1897, so that the year can hardly be considered a satisfactory one. The increase of 61 new completed wells was offset by the additional 64 dry holes, and the general average yield is shown to have been seriously lowered.

Indian Territory.—The Cherokee Oil and Gas Co. and the Creek Oil and Gas Co. have drilled several productive wells; but the output of the Territory is still small, owing in part to lack of transportation to market.

Kansas.—A petroleum refinery employing 200 men has been established at Neodesha. The total production of Kansas in 1898 was about the same as in 1897.

Kentucky.—The Kentucky Oil and Pipe Line Co. is erecting a refinery and storage tanks at Somerset, Pulaski County, on the Cincinnati Southern Railroad. This refinery will take oil from the Wayne County wells. The new pipe line is 36 miles long. The Inter-State Petroleum Co. has for some time been operating an experimental refinery at Oil City, in Barren County, and has found that about 33% of kerosene can be obtained from the local petroleum. It is believed that heavier products, including a high grade of lubricating oil, can also be made profitably. Previous attempts to refine Kentucky petroleum had been only partially successful. The wells in the Kentucky-Tennessee field are not large producers, but with facilities for refining near by are expected to be remunerative.

Missouri.—A find of oil in fair quantities is reported to have been made near Braymer, a station on the Cedar Rapids and Kansas City branch of the Chicago, Milwaukee & St. Paul Railroad, and leases covering 20,000 acres, in a territory about 10 miles long by 6 miles wide, have been taken. Oil and gas were discovered some time ago about 200 miles northwest of Braymer.

Ohio.—The southeastern oil territory properly belongs to the Appalachian field. The western, or Lima field, running across into Indiana, is distinct geographically and in the character of its product, which is of the heavy variety. The latest Ohio district to attract attention is in the neighborhood of Scio, in the Connotton valley, and is an extension of the Berea field. The oil found at Scio does not differ much from that found in other Berea grit pools. It does not equal in quality some of the oils of Pennsylvania and West Virginia, although the difference is not great enough to make a distinction in the market price. The Tiona product is superior, and that of the Milltown field, a few miles from Pittsburg, and the Gantz sand oil of Washington and Greene counties, Pa., are much richer in by-products. The Berea grit or rock formation in which the oil is found at Scio is about 35 ft. thick, and the richest pay is found midway between the top and the bottom. It lies 200 ft. lower from the surface than the Muddy Creek pool, in Butler County, Pa., the most northern Berea pool yet discovered.

Pennsylvania and New York.—The decrease in the production of the Appalachian field in 1898 was due to this part of the district, West Virginia returning an increase. Less than 400 new wells per month were completed, against over 500 in 1897, the total decrease being 1,284. No new pools of any magnitude were tapped and no new territory of any consequence was added to the oil-producing region, the production of which has passed its zenith and will probably continue to decline.

Tennessee.—John H. Compton, of the Inter-State Petroleum Co., of Providence, R. I. (property in Tennessee and Kentucky), furnishes the following data: Interest in oil in Tennessee centers in the Bob's Bar well, on the east fork of Obey's River, in Fortress County, owned by the above-named company. The production is small but regular, having remained the same from day to day after 700 days' pumping, during which time it yielded 14,000 bbl. The company has another well within 1,000 ft. of the Bob's Bar, which is in oil but is not yet pumping. In the same neighborhood various other wells were drilled in 1897, but with the exception of one on the Lacey farm, near Spurrier, none has been productive. The field has received little attention from oil men, because hitherto nearly inaccessible for transportation, the country being rough and the nearest railroad 30 miles distant, while there is no local market. The National Transit Co. has provided limited storage in anticipation of expected development, but has not yet laid pipe lines to producing points. The field is believed to be extensive (it is 20 miles southwest from the successful developments in Wayne County, Ky., to Bob's Bar), and means will undoubtedly be found to get the products to market. So far the wells are small producers, but it is thought that even 3 and 5-bbl. wells in sufficient number will afford a basis for business. The Inter-State Co. proposes to build a refinery, with a capacity of 450 bbl. weekly, at some point in the State not yet determined. It is to have improved apparatus for distilling and refining, and will turn out gasoline, benzine, kerosene and lubricating oils of various grades, with a wax plant in connection, and depending upon the local supply of crude oil. Another small refinery is projected by the Southern Oil Manufacturing Co. at Jamestown, Tenn.

Texas.—The Corsicana field increased its production largely in 1898. At the opening of the year there were 50 wells, not gushers, but steady producers. By December 1 there were 352 completed wells, of which 27 were dry and four gassers, or only 9% non-producers of oil. The daily output was then about 2,200 bbl., and was increasing at the rate of 100 bbl. a day each month; another estimate placed the daily production at about 3,000 bbl. at the close of the year. The new Hardy-Holbert-Barth well at Corsicana began spouting heavily, so much so as to make its completion difficult. Another new well, the Mirus, several miles from Corsicana, came in at this time, flowing 20 bbl. a day. The wide territory being explored and the small number of dusters make the field important; while a peculiar feature is the absence of rock covering until oil is reached, which allows fast drilling by the rotary system. One well is reported to have been put down 1,000 ft. in 32 hours, and another one the same depth in 30 hours. The Corsicana refinery began shipments of refined oil early in 1899, starting on a stock of over 400,000 bbl. of crude, and had to continue building reservoirs to hold the output of the field.

Oil has also been found near Dallas. The tract at Sour Lake is being developed, each well drilled there having tapped oil. The 25 old wells at Nacogdoches are being overhauled, though they flow less than 1 bbl. each. per day.

Stocks are accumulating in Texas, the local consumption not being sufficient to absorb the increasing supply.

Washington.—Steps are being taken to develop the petroleum field which is known to exist under the beds of glacial drift on which Tacoma is built. It is believed that the porous sandstone in which the oil occurs will be struck at 1,200 to 1,500 ft. depth. Oil springs are found in the vicinity, the oil being of good quality.

West Virginia.—Prof. I. C. White, State Geologist, reports that the petroleum industry of this State was highly prosperous in 1898. Several new and prolific fields were opened. The production is entirely of the "white-sand" type, and therefore of high quality. The output was greater in 1898 than ever before, amounting to 13,603,135 bbl., against 13,090,045 in 1897. Consequently West Virginia furnished nearly one-half of the production of the Appalachian field in 1898.

The deepest Berea grit field in West Virginia is the Ogden-Hendershot development in Wood County. This field not only had the largest daily production —3,000 bbl.—for a given time, but is the largest in area. Active developments have been in progress for more than two years, and the producing territory extends northeast and southwest for a distance of nearly 8 miles, and has an average width of more than 3 miles. It is thought to be nearly or quite defined. Starting with the Muddy Creek pool, and moving southwest to the Ogden-Hendershot pool, the most southern Berea grit development, the formation has a gradual dip and lies 2,200 ft. below the surface, or 1,000 ft. deeper than the same formation at Scio, Ohio. The production of the wells in Wood County showed a higher average, but the difference in depth and increase in production as compared with Scio leave the latter the more desirable territory to operate. The wells in Wood County have sufficient rock pressure to make them flow for a long time after they have been completed and shot, which materially reduces the expense of operating them.

Wyoming.—Until recently the Denver & Gulf Railroad furnished the only outlet for Wyoming oil, but at the close of 1898 the Fremont, Elkhorn & Missouri Valley line made rates permitting shipment to Omaha and Chicago. The transportation question is the critical one in Wyoming, as the local market is very limited. It is expected that better rates will stimulate drilling and production at all points within 50 miles of the first-mentioned railroad.

OUTPUT OF THE PRINCIPAL PETROLEUM-PRODUCING COUNTRIES. (*a*) (IN METRIC TONS.)

Year.	Austria.	Hungary.	Canada.	Germany	India.	Italy.	Japan.	Russia.	United States.
1892	89,871	20	109,080	14,527	29,084	2,548	9,825	4,896,179	7,046,199
1893	96,331	14	111,700	13,974	31,079	2,652	13,276	5,790,692	7,023,235
1894	111,930	2,061	116,000	17,232	44,506	2,853	23,837	5,191,552	6,926,361
1895	188,634	2,085	101,587	17,051	52,032	3,594	(*b*)22,463	7,056,330	7,077,364
1896	262,356	2,168	101,682	20,395	60,228	2,524	31,414	7,066,232	7,708,236
1897	275,204	2,299	99,310	23,303	76,834	1,932	(*c*)	(*c*)	7,992,046

(*a*) From the official reports of the respective countries. This table is only partially complete since it does not include the production of Sumatra, Borneo, Java, South Africa, Peru, and some other countries. (*b*) Estimated from hectoliters. (*c*) Not yet reported.

Austria.—The latest official returns are for 1897, in which year the output was 275,204 metric tons. In Galicia the oil producers' association has been reorganized, and now includes 85% of the total product, the only outsider being the Austro-Belgian Oil Co. Replying to unfavorable criticisms on the quality of Galician kerosene exported, R. Zaloviecki, in a recent paper,* while admitting defective care in refining, asserts that the oil, being midway between the American and Russian in point of capillarity, will burn well if used in properly constructed lamps. A product equal to American oil could be obtained by reducing the yield of illuminating oil by 10 to 20%; but this would be unprofitable to the refiner at the low prices ruling.

The importation of petroleum products into Austro-Hungary during the first 10 months of 1898, with the corresponding figures for 1897 following in brackets, was as follows: Russian refined oil, 278,038 metric centners (405,038); crude mineral oil, light, 30,757 (36,759); Rumanian crude oil, 185,696 (156,760); mineral oil, refined, heavy, dark and bright, 70,178, (79,010); lubricating oil, 71,352 (60,375); petroleum, 29,690 (34,783); paraffin, impure, 12,945 (21,225); paraffin refined, 39,049 (33,342); ozokerite (crude), 25 (23); ceresin, 31 (40); petroleum, bbl., 52,945 (75,393). The exports during the same period were: lubricating oil, 15,062 (11,343) centners; crude mineral oil, 8,022 (13,053); petroleum, 24,081 (122,170); benzine, 181,913 (178,238); paraffin unrefined, 81 (34); paraffin, refined, 99 (150); ozokerite (crude), 36,198 (39,297); ceresin, 11,470 (10,817).†

Bosnia.—There is a petroleum refinery at Brud, which is said to be operated at a small profit.

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* *Chem. Rev. Zeits.- u. Harz- Ind.*, Vol. V., 1898, p. 201.

† *Ungarische Montan- Industrie- und Handelszeitung*, V., 2, Jan. 15, 1899.

Burma.—The present yearly output is somewhat over 450,000 bbl., the production of refined oil being in the neighborhood of 40,000 tons, and the export of paraffin to Europe about 15,000 tons. The yield of crude oil in 1897 (the latest year for which returns are available) was 18,904,710 gal. (450,112 bbl.), an increase of 27·59% over that of the preceding year. Several companies have been drilling in Upper Burma, but the results have not come up to anticipation. With the exception of a few native refineries worked on primitive methods the Burma Oil Co.'s works were the only ones in operation in 1898. The most important field has been pretty thoroughly tested. The known oil-bearing localities spread over a considerable area, however. Dr. F. Noetling, who has lately made a report (published by the Geological Survey of India), states that all the localities are situated on the crests of anticlinal arches. The oil occurs in beds of Miocene age, and the occurrence, according to Dr. Noetling, seems to be limited to a zone about 40 miles in breadth which runs along the eastern side of the Arrakan Yoma. The crude petroleum yields about 50% illuminating oil, 40% lubricating oil and 10% paraffin, etc.

Canada.—Shipments from Petrolia, Ontario, in 1898 amounted to 150,436 bbl. crude and 180,033 bbl. refined oil, or together equal to 601,532 bbl. crude petroleum. The barrel is calculated at 35 imperial gallons (about equal to 42 United States standard gallons). Pittsburg oil men found oil on Maintoulin Island, Ontario. As the island is an Indian reservation, arrangements were made, through the Canadian government, to exploit the land, paying the Indians a royalty of 2½c. per gal. on all oil produced.

In Newfoundland tests were made of the find at Parson's Road, on the west coast, by a local company, and 110 bbl. were pumped from two wells at this locality, merely as an experiment. An English company has taken over the property, and proposes to develop it. Another promising locality was tested in 1898, at Port au Port bay, where four holes were put down, all yielding oil.

Germany.—The most important oil field is in Alsace; another, of less moment, is in Prussia, near Hanover. The domestic production is increasing, amounting to about 150,000 bbl. yearly, but this is insignificant compared with the importations of Russian and American oils.

Hungary.—According to an official report, the oil-bearing strata in the district of Kovosmezo, in eastern Hungary, are of Middle Eocene (early Tertiary) age, and consist of pale gray, finely micaceous thin-bedded sandstones, veined by calcite; then come gray marls and dark clay slates, which are interbedded with a highly micaceous, rotten, bituminous sandstone. The last occurs at a high horizon, generally in the middle of the basin, while the first mentioned sandstones occur at the edge of the basin. Numerous folds characterize this series of rocks. Within the last 20 years several companies have put down borings, only four of which have been carried to any considerable depth, but the results so far are unpromising. Down to 780 ft. no great quantity of petroleum is available, and in order to ascertain whether it occurs in workable quantity lower down, one would have to bore through the entire Middle Eocene series. The disturbed condition of the strata does not permit an exact estimate being made as to the depth this would imply, but it could hardly be less than 1,600 ft.

India.—Besides Burma, the oil-bearing provinces are Assam, the Punjab and Beluchistan; in the last-mentioned region, however, operations have practically ceased at present. The Punjab product is mainly from Rawalpindi, with a little from Bannu. In Assam oil has been found in large quantity at Digbo in the Lakhimpur district, where there were 29 wells in 1898. The total production of crude in India in 1897 was 76,834 metric tons.

Italy.—The domestic production is still small, notwithstanding the incentive to development given by the extremely high import duty. Considerable prospecting has been done in recent years, but without very satisfactory results.

Japan.—Although the consumption of petroleum in Japan is steadily increasing, the progress of the industry has been rather disappointing. The richest oil field, which is in the Province of Echigo, supports a large number of producers, but with a single exception they are all unprosperous. The output of Niigata in 1897 was about 500,000 cases, but in 1898 it fell off considerably. The industry gives employment to 8,000 to 10,000 laborers of both sexes. Most of the apparatus used is imported from the United States, a small part being manufactured in Tokio. The local products do not compete in price with the American, and are for most purposes mixed with the latter.

Mexico.—There are several oil fields in the Republic, one of the most promising being that in the State of Tamaulipas. The Vera Cruz oil-bearing territory is said to comprise 10 sq. miles, near the city of Papantla; in this field wells have been sunk to the oil strata, some of the bores running to waste at last accounts. An English company has spent about $60,000 in development there. Petroleum has also been found in Oaxaca. The government has recently granted a concession to the Mexican Petroleum Development Co., of Monterey, to establish a petroleum industry in the Republic, with a view of supplying local demands and possibly exporting. The company has a capital of $800,000, and agrees to invest not less than $250,000 within five years, the term of the concession being 10 years. Mexican and New England capitalists are interested in opening oil fields in southern Mexico.

Persia.—Petroleum is reported to have been found in the Tertiary deposit on the S.W. border of the Persian highlands, near the Persian gulf. Along the gulf there are oil springs which were known as far back as the second century. Oil springs also occur in the Bakhati range in southern Persia, near the Dakkhi River.

Peru.—There are three companies engaged in the production of petroleum in the department of Piura, namely, the London Pacific (British), Faustino G. Piaggio (Italian), and the Compagnie Française (French). The properties of these companies are on the seacoast N. W. of Payta. The headquarters of the London Pacific Co. are at Talara, on the coast, where it has an extensive plant, including a refinery. The wells are situated in the Negritos district in latitude 4° 40′ south and longitude 81° 17′ 30″ west, or 6·5 miles S.E. of Talara. The geological formation, which has been proved by borings to a depth of 900 ft., shows that the production is not due to surface filtration, but is from an established formation. The strata dip east at an angle which in some cases is as steep as as 45°. Within this district, which at present embraces an area of about 1·75 miles north and south and 1 mile east from the coast, 60 wells have been sunk, some of which have already given out, while others have yielded only a small quantity, though in the opinion of experts the old wells may become productive again if they are deepened. During the first six months of 1897 work was almost at a standstill on account of litigation, but during the second half the average output was about 800 tons of crude oil per month. This is pumped (no flowing wells existing) into a storage tank, whence it is drawn through a pipe line 6·5 miles long to Talara.

The product of the refinery is approximately as follows: Benzine 10%; kerosene 30%; residuum 50%, with apparently a loss of 10%. Unfortunately the Peruvian petroleum industry has met with the difficulty of not being able to produce a kerosene that can compete in quality with the American product, which sells on the coast at about $3·75 per case of 10 gal. This is due to the fact that the Peruvian crude oil is of an asphaltic nature, like that of California. The business is restricted at present, therefore, to the sale of crude oil for smelting purposes at the Casapalca silver mines, although there is a limited production of kerosene.

The property of the Faustino Piaggio Co. is situated at a point called "Zorritos," a considerable distance north of Talara. It embraces an area of 216 hectares and has been in operation since 1864; since then over 100 wells have been sunk, all of which have been productive. There are at present only 20 yielding, which, according to data furnished by the owner, produce 10,000 gal. per day, or say 1,200 tons per month. Two extra deposit tanks of 1,000 tons each have been built, and the company contemplates obtaining, like Talara, a special steamer, when it proposes to augment its production by sinking other wells. Two grades of kerosene are produced and the residuum is converted into lubricating oil. This company also has an extensive plant, including a refinery. The kerosene produced by this company is more or less of the same quality as that of Talara.

The property of the Compagnie Française is near the port of Grau, and comprises an area of 220 hectares; its capital has been increased from 2,000,000 to 3,000,000 fr. It has erected machinery, workshops, accommodations for employes, deposit tanks and a refinery for kerosene. Six wells have been sunk, the total production from them being about 13 tons per day.

The Peruvian kerosene gives a dull light and smoky flame, and brings only 2·5 to 3 soles per 10 gal., while the American fetches from 8 to 9 soles.

Rumania.—The following information is from a late report of the Austrian vice-consul at Plojesti: Operations have generally been very successful. New oil pools have been discovered at several places. Tools and machinery of the latest construction have been adopted for drilling, and in several instances oil has been struck in unexpected quantities. The total production of 1897 greatly exceeded that of 1896. The Steana Romana, the most important oil concern, and operating with a working capital of 10,000,000 fr., owns most of the oil territory around Campina, and has about 60 wells, with an average daily production of 11 tank cars of 10 tons capacity each (a total of about 770 bbl.). Four other large companies, one English, two Dutch, and one Austrian, operate numerous wells on the communal lands of Campina, Bustenari, Doftanetz, Baicoi-Zintea, and several other villages which produce on an average from one to two tank cars per day. Producers who operate on their own lands, but are remote from railway communication, suffer much loss during periods of bad weather, owing to the impassableness of the roads. Of the total crude product, about 70% is worked up in native refineries, and the remainder is exported, mostly to Austria-Hungary. The Steana Romana alone controls five refineries in various cities, while 15 more are located in the Plojesti district. Their products, gasoline, illuminating oil, benzine, etc., are shipped to all parts of the country. The export of these products, however, is still in its infancy. In 1896 fully 80% of the product was obtained from wells dug by hand; in 1897, however, a slightly larger production came from driven holes. The depth of the open wells ranges from 60 to 300 meters, and that of the bore holes from 140 to 335 meters.

Russia.—According to James C. Chambers, United States Consul at Batum, in a report to the Department of State, the production of petroleum at Baku continued to increase in 1898. The average daily production of crude was over 20,000 bbl. (of 42 gal.) more than in 1897, although the yield per well decreased. The number of wells increased largely; in 1896 the average daily production per well was 226 bbl., in 1897 it was 211, and in 1898, 199. From 1892 to 1896 the average daily production per well did not vary 5 bbl. from 225 bbl.; the first weakening of the wells was noticeable in 1897; and as it increased in 1898, it gives the falling off an appearance of permanency. The average daily production in 1897 increased over that of 1896 about 12,000 bbl., by the completion of 209 wells, *i.e.*, about 60 bbl. per new well; the increase in 1898 from the completion of 258 wells was about 20,000 bbl., or about 80 bbl. per well. This would apparently refute any idea of a falling off in the unit production of the wells, without the explanation that in 1898 one well produced about 4,000,000 bbl. in less than two months, making about 11,000 bbl. per day for the whole year. Consequently, if this phenomenally large well be eliminated, the increase in 1898 from the other 257 wells completed was only about 10,000 bbl., or not more than 40 bbl. per well. By leaving out of the account a very few other exceptional flowing wells in the Bibi-Eibat district the increase in 1898 from the remaining 255 wells is brought down to only about 4,000 bbl. a day. Attention is called to these facts in order that the statistics may be properly understood, as it is believed they mean that unless deeper and equally rich strata are found in this district, or other rich territory is found to replace it, it will require much more drilling in the future than in the past to increase materially the Baku production.

The Romani district, which is really an extension to the northward of the Sabunchi field, is the newest territory in the Baku fields; but it has been drilled more rapidly than the older territory, because it has been exceedingly rich, though at greater depths than the older ones. It cannot be said that this territory is showing signs of exhaustion, because it has many good pumping wells; but, like Bibi-Eibat, the drilling is much deeper than in other districts, for the 21 wells finished in 1898 had an average depth of 1,448 ft., against an average of 1,118 ft. for the 22 wells finished in 1897. Consequently, it would seem that it is not at all likely to produce more big wells, unless rich strata are found deeper or the territory is extended.

The other two districts in the Baku field—Balakhani and Sabunchi—although the oldest part of the territory, appear now the most important part, as, while they have been exceedingly rich in oil up to the present, the average depth of the drilling has been much less than in the other districts, having been only 845 ft. in 1898, against 815 ft. the previous year; and, as the Romani territory was richest at over 1,100 ft., Balakhani and Sabunchi may also be found to be very rich at somewhere about that depth. In January, 1899, a well in this territory, which had been very productive at shallower depth, having been deepened to about 1,250 ft., was producing at the rate of 60,000 bbl. per day.

The yield in merchantable products of the Baku crude is a matter which has not received as much attention lately as in former years, because, no doubt, it is pretty generally accepted that about 30% illuminating oil can be obtained from it. Fully that proportion of illuminating oil can be obtained from the crude, but the average result of refining has not been so great for many years. Following are the figures of the apparent yield for the last four years:

Description.	1895.	1896.	1897.	1898.
	%	%	%	%
Illuminating oils	26·5	24	22·3	20·5
Lubricating oils	2·3	2·25	2·4	2·4
Residuum	50·2	56·25	56·5	54·4
Totals	79	83·5	81·2	77·3

These results are based upon the gross production of crude, its shipment and stock, and the output of the refineries; but it must be explained that there is not really as much loss as they indicate, because at least 8% of the gross production is consumed by the wells and refineries as fuel, so that only about 92% of the total crude is refined.

Notwithstanding the enormous gain in production and a continuous increase throughout the year, the price of crude steadily advanced until it reached 55c. per bbl. about the close of the Volga navigation season, and practically did not recede from that figure to the close of the year. This state of affairs explains the great increase and energy in drilling. There is no doubt that increased depth raises the cost of crude, but such advance in cost is insignificant in comparison with the recent increase in the price. The highest estimate of the cost of producing crude oil in 1897 was about 17c. per bbl., and this cost, plus the most liberal allowance for increase because of deeper drilling, certainly leaves a sufficiently wide margin for profit to induce the greatest energy in drilling; besides, there is always the chance of a big flowing well, one of which in a year will materially reduce the producer's cost of crude, and consequently greatly increase his profits. But taking the average well—*i.e.*, about 200 bbl. per day—at present prices, the income is not less than $100 per day, and at that rate it does not take many days to get the cost of the well back.

The use of residuum, or fuel oil, is rapidly increasing in Russia, largely because of greater activity in manufacturing. Throughout a large part of Russia (that is, wherever transportation conditions permit) it has supplanted coal, except for domestic heating and blacksmithing. It was worth, spot, 57@58c. per bbl. at the close of 1898. To save cost of refining much crude is mixed with residuum, and is sometimes sold alone, for fuel purposes. It has a limited export, and that mainly for making cheap lubricants.

Illuminating oil is looked upon as of minor importance. Its export would have been impossible, except at considerable loss, had it not been for the reduction in the railway freight from 9·7c. to 6·18c. per pood (36·11 lb.), which went into effect in December, 1897. This was neither more nor less than a subsidy. The principal exporters are, however, producers, so that their crude costing less, a smaller margin is possible. The price of refined oil at the end of 1898 was 24 copecks (12·3c.) per pood at Baku, and about 38 copecks (19·57c.), or about 4c. per gal. f. o. b. at Batum. With the unfavorable conditions for refining which have existed for some time, it is not at all strange that comparatively few refineries were working last year. Three years ago there were at Baku about 120 refineries; but the statistics show that only 63 worked in 1897, and of that number 20 produced over 90% of the total output. Statistics of the refining industry for 1898 are not at hand, but probably the number working throughout the year was less than in 1897.

The pipe line from Baku to Batum is far from complete. The 145-mile section, Michailovo to Batum, is under construction, but was delayed by difficulty in obtaining pipe. It is to be an 8-in. pumping line, with pumps of American make. The intention is to carry refined by rail from Baku to Michailovo (415 miles) and then pipe it to Batum when the line is finished.

There was a small increase in the export of all oil products from Batum in 1898, but the increase in illuminating oils was only about 15,000,000 gal.; so that this trade cannot be held accountable for any falling off in the American export, and until the completion of the pipe line there can be no material incease in the Russian export, owing to the limited capacity of the railway.

Much foreign capital has been invested in Baku lately, probably not less than $10,000,000 of British money alone in 1897 and 1898. The earlier investments were extremely successful, and led to a boom in oil lands which appears to have carried prices beyond reason.

The outside Grosni field exhibited no redeeming features in 1898, as, notwithstanding considerable drilling, there was a decrease in the production. The yield in 1897 was 2,754,000 bbl.; it was only 1,672,080 bbl. for the first 10 months of 1898. The oil is sold mainly for fuel, as it gives much less illuminant than the Baku oil and deteriorates more rapidly from exposure, being reduced in a few weeks to Baku standard for residuum. The Grosni petroleum, although of higher specific gravity (0·9) than the Baku, nevertheless carries higher contents of light oils, as gasoline and benzine (up to 20%).

There are also some wells near Noworossisk, producing a very heavy oil, and a small refinery at that place which also receives Grosni oil, which is run almost wholly for residuum.

OUTPUT OF ALL PRODUCTS FROM BAKU IN 1897 AND 1898.
(In thousand gallons.)

Year.	Illuminating.	Lubricating.	Residuum.	Crude.	Other Products	Total.
1897	458,035	45,860	1,127,100	130.045	5,925	1,766,965
1898	173,205	52,105	1,211,805	219,460	....	1,956,575

STOCKS OF ALL PRODUCTS AT BAKU AT CLOSE OF 1897 AND 1898.

Product.	Dec. 31, 1897.	Dec. 31, 1898.	Product.	Dec. 31, 1897.	Dec. 31, 1898.
Crude at wells, bbl	592,547	859,583	Illuminating oils, gal	67 274,980	51,805,615
Crude at refineries, bbl	1,840,212	1,462,344	Lubricating oils, gal	9,610,725	11,605,275
			Residuum, gal	263,532,270	265,121,885
Total crude, bbl	2,432,759	2,321,928			

SHIPMENTS OF PETROLEUM PRODUCTS FROM BATUM IN 1897 AND 1898. (IN GALLONS.)

	Crude and Residuum.		Lubricating.		Illuminating Distillate.		Refined.	
	1897.	1898.	1897.	1898.	1897.	1898.	1897.	1898.
Austria	205,190	176,850	2,850,120	2,985,615	12,381,270	11,176,665	2,826,530	4,705,470
Belgium	3,546,595	3,723,720	8,086,275	8,047,315	233,185	159,490	1,879,690	8,106,380
Bulgaria	700	27,450	58,850	46,150	....	....	6,939,730	2,270,040
China and Cochin China	....	....	....	....	....	....	19,406,270	9,834,880
Egypt	6,750	20.450	140,600	114,550	....	....	11,001,690	13,153,390
England and U. K.	2,116,810	2,634,475	4,725,945	5,830,140	13,646,660	4,321,665	23,485,345	41,572,465
France	1,637,395	2,373,630	6,699,545	9,338,175	12,855,135	14,864,665	856,270	1,639,775
Germany	1,236,865	2,020,955	8,518,310	10,601,800	97,000	78,305	7,257,905	7,910,465
Holland	....	....	100,000	5,000	....	....	....	2,071,805
India	....	89,050	1,000	100,000	....	....	34,984,880	39,488,950
Italy	1,843,840	1,056,340	559,800	316,750	....	....	5,862,220	7,567,680
Japan	....	....	....	....	....	....	6,142,140	2,718,800
Java	....	....	....	....	....	....	1,362,250	5,198,840
Manila	....	....	....	....	....	....	1,626,000	1,668,280
Malta	....	....	....	....	....	....	1,674,415	1,475,835
Rumania	4,100	1,000	233,500	142,550	....	....	515.740	423,200
Spain	326,655	491,850	759,445	246,845	....	....	....	....
Turkey	73,450	72,850	130,700	79,400	....	....	32,336,325	28,730,005
Suez Canal, bulk	....	20,500	....	10,000	....	....	56,402,320	55,730,190
Africa	....	....	....	....	....	....	1,140,360	1,217,200
Elsewhere	....	1,000	4,500	15,250	....	....	511,170	685,100
Total exports	10,992,350	12,710,120	32,869,090	37,879,540	39,213,250	30,600,790	217,210,980	236,168,750
Russia	255,470	754,745	1,143,555	1,965,200	33,700	72,450	31,438,865	27,596,940
Total shipments	11,247,820	13,464,865	34,012,645	39,844,740	39,246,950	30,673,240	248,649,845	263,765,690
Increase	7,897,890	2,217,045	16,771,870	5,832,105	7,445,170	....	62,931,744	15 115,845
Decrease	....	....	....	....	....	8,573,710	....	....

Distillate to United Kingdom was gas oil. Suez Canal bulk shipments were for points beyond the canal, but unknown at Batum.

In the Western Caucasus, near Anaclia, sand beds impregnated with "naphtha" were tested in May, 1898. The water which accumulated in the preliminary excavations soon became coated with a layer of liquid "naphtha." Boring was expected to follow. If productive, the locality, being on the Ingur near its outlet into the Black Sea, would have advantages in point of accessibility to European markets over the Baku field on the Caspian. Petroleum springs were discovered by Herr Klein in the summer of 1898 in the vicinity of the river Nuton, on the east coast of Saghalien, Siberia. The crude oil has a sp. gr. of 0·905 as reported.

United Kingdom.—The peculiarity of the Scotch oil-shale industry is the closeness with which all by-products are utilized, without which competition with other producers more favorably situated as regards natural resources would be impossible. The products are illuminating, lubricating, and gas oils, naphthas, paraffin and ammonium sulphate. In 1897 the amount of oil shale mined was 2,223,745 long tons (2,259,431 metric tons), valued at £555,935 at the quarry. While sulphate of ammonia was selling in 1897 at £8 15s. per ton, early in 1898 it rose to over £10 a ton, and after slight fluctuations closed at that price, which is a figure satisfactory, on the whole, to the makers, though nothing like what has occasionally been got in former years. Lubricating oils strengthened perceptibly after the close of the engineering trades strike, but without any rise in price at first, for the position of these mineral oils is strictly governed by the quotations for rival lubricants in the market, particularly those introduced by the American petroleum refiners. On the whole, the level of profit was about maintained in 1898, the range of price at the end being £4 15s.@£6 10s. per ton for gravities ·865° to ·895°. Naphtha is relatively a small outcome of the distillation of shale, but did well during the year, steadily extending its sphere as an agent in many different manufacturing operations, while at the same time maintaining its hold as an illuminant peculiarly suitable under certain conditions of use. Its price advanced from 6½d.@7d. per gal. to 8½d. per gal., which is a record figure within late years. Scotch burning oil stood at rather under 6d. per gal. delivered at the beginning of 1898, at which figure it is not remunerative to most of the refiners, but later it rose to 6¼d., as the American oil advanced. Solid paraffin had been selling at 1¾@ 2d. at the beginning of the year, and was not profitable; but at the close an understanding was reached with the American makers to hold the price at a slight advance, which is a matter of considerable importance to the Scotch makers, paraffin being one of the main reliances in the oil-shale industry.

PHOSPHATE ROCK AND PHOSPHORUS.

THE phosphate industry in the United States was far more prosperous in 1898 than in 1897, on account of the higher price for phosphate rock, and consequently the production increased largely, as shown in the subjoined statistics, which have been compiled in the same manner as explained in THE MINERAL INDUSTRY, Vol. VI., except that in the case of South Carolina the shipments of land rock reported have been adopted as equivalent to the production.

PRODUCTION OF PHOSPHATE ROCK IN THE UNITED STATES.
(In tons of 2,240 lb.)

Year.	South Carolina.	Florida.	North Carolina.	Tennessee.	Other States.	Totals.	Value.	
							Total.	Per Ton.
1894	493,800	558,900	9,000	....	....	1,080,853	$3,296,602	$3·05
1895	515,734	530,356	7,500	45,078	....	1,098,668	3,296,004	3·00
1896	382,068	498,400	7,418	49,047	439	937,372	2,812,116	3·00
1897	333,626	543,490	7,000	121,251	2,000	1,007,367	3,022,101	3·00
1898	434,273	546,881	2,200	272,191	2,100	1,257,645	4,355,025	3·46

SHIPMENTS OF PHOSPHATE ROCK.
(In tons of 2,240 lb.)

State.	1897.			1898.		
	Foreign.	Domestic.	Total.	Foreign.	Domestic.	Total.
South Carolina	65,829	272,619	338,448	64,175	364,881	429,056
Florida	436,472	124,959	561,431	430,993	123,225	554,218
Tennessee	16,175	105,076	121,251	70,000	202,191	272 191
North Carolina	....	7,000	7,000	....	2,200	2,200
Pennsylvania	....	1,600	1,600	....	2,100	2,100

Prices.—The price of high-grade Florida rock rose during 1898 from $5 to $9 per long ton, f. o. b. Fernandina, while land pebble was quoted at the end of the year at $5·50@$6, delivered in New York. South Carolina high-grade kiln-dried land rock, guaranteed 60% bone phosphate, sold at the end of the year at $3@$3·25 at Beaufort, while river rock, guaranteed 55%, brought $2·80@$2·90. Local manufacturers in South Carolina were supplied at $3·15@$3·25 for high-grade, kiln-dried land rock delivered at local works, while one contract for 200,000 tons delivered within the next five years was reported to have been made at $2·75. In Tennessee the price rose from $1·45 per ton early in the year to $2·20@$2·30 in December, f. o. b. Mt. Pleasant.

The freight rates from American ports during the latter part of 1898 were as follows: Tampa to Ghent, 20s., and to Yokohama, 50s.; Coosaw to Dublin 19s., and to London, 18s. 6d.; Fernandina to Rotterdam, 18s. 3d.@19s. 6d.

Acid phosphate was sold in the South in 1898 at $5·75@$6, f. o. b. cars or vessels at factory of sellers, for shipments in bulk, containing 13·5 to 14% available phosphoric acid; bagged and tagged goods, 13%, brought $7@$7·25, f. o. b. cars or vessels.

There is a tendency to combination in the fertilizer business. This is prominently brought out in the case of the Virginia-Carolina Chemical Co., in the Southern States, which has absorbed many plants, and is now the largest factor in the trade. The business is gradually extending to the Western States, as the farmers realize that to secure good crops they must fertilize well; hence the demand increases year by year, and must continue to do so to meet the demands upon land by an increased population.

George F. Payne, in his report on "Commercial Fertilizers and Chemicals," for the season 1897-98, makes the following estimate of fertilizer consumption in the United States in 1896 and 1897, the figures for 1896 being given first and those for 1897 immediately following in parentheses: Georgia, 335,617 tons (401,979); South Carolina, 199,497 (240,000); North Carolina, 185,000 (213,000); Virginia, 171,704 (200,000); Alabama, 100,000 (130,000); West Virginia, 39,350 (26,000); Mississippi, 32,000 (42,000); Florida, 26,588 (31,000 estimated); Kentucky, 19,550 (20,950); Tennessee, 19,445 (22,645 estimated); Louisiana, 10,051 (11,780); New York, 150,000 (150,000 estimated); Pennsylvania, 150,000 (150,000 estimated); Connecticut, 20,000 (20,000 estimated); Vermont, 13,000 (13,000); Ohio, 46,000 (100,000); Indiana, 50,000 (50,000); Minnesota, no report in 1896 (100 in 1897); Wisconsin, 200 (20); other States, 7,640 (2,950); grand total, 1,575,632 (1,825,605).

Continental acid phosphate manufacturers planned for a largely increased demand for their product in 1898, but they underestimated this and were obliged to draw heavily upon English production. Realizing that the continental consumption has increased enormously, the manufacturers made unusual efforts to meet the demand, and during the last six months of 1898 were active buyers of all kinds of rock, especially high-grade, for which they have made contracts up to shipments in 1900 at extreme prices. Manufacturers in the United Kingdom bought fairly well for 1899 shipment, but held back somewhat, hoping for reduced ocean freights, the ship owners having absorbed a large part of the increase in values in 1898.

At the end of the year the price paid for 75 to 80% Florida rock for 1899 shipment c. i. f. to ports in the United Kingdom, was 8d. per unit, against 6·5d. at the end of June, 1898. Tennessee 78 to 80% rock was quoted at 1d. to 1·25d. per unit below the price of Florida; Algerian 58 to 63% brought 6·5d., and 63 to 70% 6·75d., the bulk of the output of 1899 being already sold. South Carolina rock was offered at 6d. per unit.

Florida.—Messrs. Auchincloss Bros. in their annual report state that the increase in the price of phosphate rock, which began early in 1898, led to general prosperity in phosphate mining in Florida, and the miners who have survived the speculative era of the past few years, profiting by their experience, are now working more in harmony and with a view of obtaining a fair profit, so that the industry may be regarded as having passed the speculative stage and settled down to a sound basis. Although the improved prices have stimulated production, it is improbable that the prospective increase will more than meet the growing demand of Europe and the new foreign and colonial markets. The price for the product at the mines at the end of 1898 no more than returned a reasonable remuneration to the miners for the increasing difficulties of mining, the cost and steady depreciation of plant and the progressive exhaustion of their property, and if the demand for Florida high-grade rock continues to increase at the average rate for the last three years the trend of prices will be toward a higher level.

The shipments of high-grade phosphate rock from Florida in 1895-98, according to Messrs. Auchincloss Bros., are given in the following table:

Month.	1895.	1896.	1897.	1898.	Month.	1895.	1896.	1897.	1898
	Tons.	Tons.	Tons.	Tons.		Tons.	Tons.	Tons.	Tons.
January......	15,780	16,996	12,924	11,682	August.......	14,588	19,914	19,292	27,409
February.....	17,252	16,853	20,668	26,850	September....	25,388	25,116	59,966	46,961
March.........	31,283	37,155	37,243	34,049	October.......	27,783	30,605	27,664	21,476
April..........	41,445	36,559	32,608	22,274	November....	18,160	38,402	20,184	30,595
May...........	45,053	45,846	45,715	31,992	December....	17,003	23,618	18,537	22,155
June...........	31,027	16,511	32,837	31,948					
July...........	21,284	15,296	22,639	53,114	Totals.....	306,046	322,871	350,277	306,305

The stock of high-grade rock on hand and ready for shipment January 1, 1898, was only half as large as on January 1, 1897. In 1898 there was a further large decrease in stocks. This explains the difference between the statistics of shipments and those of production.

The present composition of the Florida high-grade rock is illustrated by the statistics of the Dunnellon Phosphate Co., which in 1898 shipped 74,662 long tons of rock averaging 79·38% bone phosphate, 2·33% iron oxide and alumina, and 1·28% water. The range was: bone phosphate, 78·35 to 80·72%; iron oxide and alumina, 1·42 to 2·90%; water, 0·43 to 2·17%.

North Carolina.—The Hanover Land and Improvement Co. was organized to take over the Castle Hayne phosphate lands, under contract with the State.

Pennsylvania.—The phosphate mines of this State, which are situated at Ross Farm, Juniata County, on the Tuscarora Valley Railway, 22 miles from the main line of the Pennsylvania Railroad, are operated by the Tuscarora Fertilizer Co., which in 1898 erected a large plant for the manufacture of acid phosphate from the mineral obtained there. The average Pennsylvania phosphate rock is of low grade, but the good and the poor are so distinctive in appearance that they can be easily separated, and it is believed that a large supply of rock containing upward of 50% bone phosphate is available. The mineral occurs in beds dipping steeply into the mountain.* It is estimated that the Ross Farm mine will yield 300,000 tons above the water level. The

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* *American Fertilizer*, March, 1896, and June, 1897.

acid phosphate works are situated within 300 yards of the mine opening, and it is expected that the cost of mining and delivering the rock to the works will not exceed 50c. per long ton.

South Carolina.—A. W. Jones, the State Phosphate Commissioner, in his report for the year ended December 31, 1898, states that at the beginning of the year the condition of the industry in South Carolina was far from encouraging, the mining plants of all the large companies being closed down with the exception of the Beaufort Phosphate Co. The price of rock was about at the cost of production and sales were hard to effect. About February 1, a demand for river rock arose and continued throughout the year at advancing prices, so that shipments would have exceeded those of 1897 if it had not been for the advance in ocean freights. The price of hot-air dried river rock January 1, 1898, was $2@$2·25 f. o. b.; on December 1, 1898, one company refused an offer of $2·65. The Coosaw Co. resumed operations in February and the Central Phosphate Co. (successor to the Farmer's Mining Co.) resumed about April 1. The severe storms of August 28 and October 2 did great damage to the river mining industry, and caused a loss in output of about 8,000 tons.

The total production of 1898 was 99,315 tons, against 76,313 in 1897. The stock on hand December 31, 1898, was 33,015 tons, against 26,659 at the corresponding date in 1897. The shipments in 1898 were 94,098 tons, against 95,237 in 1897. Of the shipments in 1898, 64,174 tons were exported, 11,761 tons went to coastwise ports and 18,163 tons to Charleston and Beaufort. The shipments of South Carolina land rock coastwise amounted to 98,508 long tons in 1898, against 91,046 in 1897; shipments by rail, 51,450, against 52,165; local consumption of land rock, 185,000, against 100,000; total shipments of land rock, 334,958, against 243,211.

Tennessee.—At the beginning of 1898 the price of phosphate rock in Tennessee was at the same low level as for several years previous, but about May there was an increased demand and an upward tendency in the market which continued to the end of the year. This caused great activity in mining and resulted in a largely increased output. The export trade also increased largely, and bids fair to consume all of the high-grade product of the Tennessee field, leaving only the second class for the domestic market. Since, however, the latter can be guaranteed 77% bone phosphate and 4·5% iron oxide and alumina, it easily satisfies the American demand. Aside from the increase in the export trade the most noteworthy feature of 1898 was the opening of mines in Perry County, whence a few hundred tons was shipped. The rock is of high grade, and if the new district proves to be extensive it will become a considerable factor in the market. The increased demand and price for phosphate rock led to a large increase in the value of phosphate land in the district. The Tennessee rock is higher in bone phosphate than the best grade from Florida, running 78 to 84% in bone phosphate, although it is also higher in iron oxide and alumina, which run up to 4%.

The Mount Pleasant phosphate field is situated in the S. W. part of Maury County, in the valleys of Sugar Creek and the east fork of Bigby Creek. It is triangular in area, the base of the triangle being about 4·5 miles long, central at the junction of the two creeks above mentioned, and about perpendicular to the course of the lower part of the east fork of Bigby Creek. The apex of the triangle is between the sources of the two creeks and about six miles S. E. from Mount Pleasant. The district comprises about 9,000 acres. The same formation exists outside of this area, but the phosphate rock there is high in silica and contains only 40 to 60% bone phosphate of lime.

The largest producers in 1898 were the Central Phosphate Co., which shipped 25,000 tons, of which 16,500 were exported; the Tennessee Phosphate Co., which shipped 65,000 tons, of which 30,000 were exported; the Blue Grass Phosphate Co., Columbian Phosphate Co., and Petrified Bone Mining Co.; the International Phosphate Co., which in 1898 shipped 35,000 tons, of which 15,000 were exported; the Read Phosphate Co., Birmingham Fertilizer Works, and National Acid Co.; Robin Jones, who shipped 40,000 tons in 1898, of which 6,000 were exported; the Mount Pleasant Phosphate Co., which shipped 28,500 tons, 3,000 tons being exported; and F. Hardy & Co. The remainder of the shipments in 1898 were made by various small miners. Most of the above-mentioned companies have local offices at Mount Pleasant.

THE WORLD'S PRODUCTION OF PHOSPHATE ROCK. (*a*) (IN METRIC TONS AND DOLLARS.)

Year.	Algeria.		Belgium. Cubic Meters.		Canada.		France.		Norway.	
1893	....	....	331,230	$881,000	7,437	$70,942	....	....	1,513	$26,541
1894	64,260	$230,888	371,776	879,300	6,224	41,166	539,000	$3,311,142	2,086	36,460
1895	157,886	634,323	506,730	683,140	1,653	9,565	526,784	3,160,600	1,601	23,760
1896	165,738	500,905	297,470	537,320	517	3,420	582,667	3,502,027	1,106	17,280
1897	222,141	912,564	350,560	436,762	824	3,996	355,390	2,852,887	(*c*) 872	12,960

Year.	Redonda. Br. W. Indies. (*d*)		Russia. (*b*)		Spain. (*b*)		United Kingdom.		United States.	
1893	....	....	13,706	$36,904	211	$415	3,353	$28,855	1,006,668	$3,467,853
1894	579	$5,525	14,072	31,680	40	80	711	6,385	1,098,147	3,296,002
1895	5,778	24,880	6,327	18,682	1,040	2,081	2,540	21,875	1,116,247	3,296,004
1896	(*e*)	(*e*)	3,780	9,000	770	3,080	3,048	26,250	952,370	2,812,116
1897	(*e*)	(*e*)	(*e*)	(*e*)	2,084	16,672	2,032	17,500	1,023,485	3,022,101

(*a*) From the official reports of the respective countries and *Annual General Reports on the Mineral Industry of the United Kingdom*, by C. Le Nevé Foster. (*b*) Phosphorites exported. (*c*) Apatite exported. (*d*) Phosphate of alumina exported. (*e*) Statistics not yet published.

Algeria.—The production of phosphate rock in the Tebessa basin in 1898 was 222,348 metric tons, the amount being less than was expected on account of an accident to the wire rope of the Crookston Co. and the partial destruction of its works by fire. Attempts were made during the winter of 1898-99 to form a combination between the producers of Algeria and Tunis to maintain prices and control production. Up to date, however, they have failed for various reasons, one of these being the discovery of large phosphate deposits in France. It is now the general belief in the phosphate industry that Algeria can only with difficulty put 200,000 to 250,000 tons of phosphate rock per annum on the market of Europe and its competition is no longer feared by the American producers. The industry in Algeria is hampered in several ways, especially by royalties and freight rates. The rock of the Tebessa district has to be hauled 220 to 235 km. over the Bonê-Guelma Railway to the port of Bonê, for which the rate is 7·68 fr. per metric ton. In the autumn of 1898 the railway company sought to increase this rate by 1·25 fr. per ton. The exports of phosphate rock from Bonê in 1898 are summarized in the following table:

Month.	Exports.			Destination.				
	France.	Foreign.	Total.	England.	Italy.	Austria.	France.	Elsewhere.
January	8,105	9,975	18,080	7,265	1,080	....	8,105	1,630
February	4,300	13,710	18,010	4,900	6,210	300	4,300	2,300
March	3,585	15,309	18,894	3,435	5,540	....	3,585	6,334
April	3,682	14,800	18,482	9,720	3,160	....	3,682	1,920
May	2,464	17,850	20,314	2,320	4.500	....	2,464	11,030
June	7,461	13,474	20,935	5,645	4,535	794	7,461	1,500
July	6,409	14,435	20,844	6,225	2,540	....	6,409	5,670
August	1,880	17,225	19,105	9,205	3,020	....	1,880	5,000
September	8,227	7,050	15,277	1,230	2,670	300	7,877	3,200
October	5,330	7,200	12,530	1,650	1,730	300	5,330	3,520
November	2,497	9,390	11,887	4,650	640	....	2,697	4,100
December	7,270	20,720	27,990	13,180	2,040	....	7,270	5,500
Totals	61,210	161,138	222,348	70,425	37,665	1,694	60,860	51,704

Of the production in 1898 the Constantine Phosphate Co. shipped 100,000 tons, Crookston Bros, 81,000, and the Société Francaise des Phosphates de Tébessá, 41,348.

France.—A new discovery of phosphate rock was made in 1898 about 32 km. south of Oloron (Basses-Pyrénées), and subsequent investigation showed that the occurrence was extensive, the mineral being identified at a large number of points outside of the Basses-Pyrénées, especially in the Ariége, Haute-Garonne, Aude and Tarn. According to M. David Levat, who discovered the deposits, they are situated in the upper part of the Devonian formation between the marble or calcareous "griotte" of the Pyrénées and the subjacent schists. The mineral is of a brilliant black color, resembling anthracite, and smuts the fingers. The deposit occurs in the form of a bed in which are segregations of nodules, forming a quarter or a third of the total mass, which assay from 65 to 75% bone phosphate and contain insignificant amounts of iron and aluminum oxide, many samples showing none at all. The mass of the bed in which these nodules occur itself contains 14 to 16% phosphoric acid, and moreover contains an important percentage of organic matter, which tests 3 to 5 kg. per ton in nitrogen. In some parts of the bed the organic matter amounts to 25 to 28%. The phosphate nodules are black, shining and hard. They are concentrated, sometimes in the upper part of the bed and sometimes in the lower part. The continuity of the nodules with depth has been proved by recently cutting the bed, 114 meters below the outcrop, in Las Cabesses manganese mine near Saint Girons. These new deposits promise to be of much importance.

Tunis.—The Gapa Co., in Tunis, is now completing the railway from Ipax to the mines, about 250 km. It has experienced some trouble in the construction, owing to the nature of the ground, while other difficulties are expected from the inundations of the heavy rains of the spring. Notwithstanding these difficulties the Gapa Co. expects to make its first shipment before the summer of 1899.

PHOSPHORUS.

There are three producers of phosphorus in the United States, namely J. J. Allen's Sons, the Moro Phillips Chemical Co. and Allbright & Wilson, the domestic consumption of phosphorus being controlled practically by the Diamond Match Co. The electrolytic process of making phosphorus has now superseded all others, the process which is employed at the works at Oldbury, England, and Niagara Falls, N. Y., respectively the largest in Great Britain and the United States, being covered by the Readman-Parker patents.

Electrolytic Phosphorus Smelting.—In the Readman-Parker process natural phosphate rock, intimately mixed with carbon and a suitable flux, is heated in an electric furnace, by means of a current passing between carbon poles, the heat being due to the resistance, and not to the arc. The furnaces are small and capable of yielding only about 170 lb. per day. The smelting goes on continuously and recovérs 80 to 90% of the phosphorus contained in the raw material. The works at Niagara Falls use 300 h. p. The Readman-Parker process is now employed in four countries.

A new electrolytic process for the manufacture of phosphorus was patented in 1898 by C. S. Bradley, of Livingston, N. Y., and C. B. Jacobs of East Orange, N. J.* According to this process pulverized phosphate rock is mixed with enough carbon to combine with all the oxygen and also the calcium, so as to form calcium carbide, 300 lb. of calcium phosphate and 200 lb. of coke being recommended as a proper charge, which should yield, it is claimed, about 192 lb. of carbide and 62 lb. of phosphorus. The charge is heated in an electric furnace, the volatilized phosphorus passing from the top of the furnace to a condenser. Some experiments with this process were made in the United States in 1898, but so far as we are aware it has not yet been introduced practically. The idea of producing phosphorus and calcium carbide at one operation is not novel.

Reduction of Phosphorus with Aluminum.—L. Franck has found that sodium metaphosphate mixed with an excess of aluminum powder and heated to dull redness is reduced energetically, the phosphorus being eliminated in part as vapor, which may be condensed in the red amorphous condition, while part remains as aluminum phosphide intermixed with alumina and sodium aluminate, also resulting from the reaction, which may be represented by the following equation:

$$6NaPO_3+15Al=6NaAlO_2+Al_5P_3+2Al_2O_3+3P.$$

Consequently one-half of the phosphorus is completely reduced, while the other half remains as phosphide. The whole of the phosphorus is reduced, however, by smelting with an addition of silica, the reaction being effected quietly and regularly, according to the following equation:

$$6NaPO_3+10Al+3SiO_2=3Na_2SiO_3+5Al_2O_3+6P.$$

With bone ash the reaction is very energetic, phosphorus being partly vola-

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* English Patent No. 10,290, May 5, 1898.

tilized and partly uniting to a complex aluminum calcium phosphide. Phosphorite and fossil bones behave in a similar manner, but with apatite the reaction is less complete. Calcium metaphosphate is, however, completely decomposed. The best results are obtained with bone ash treated with hydrochloric acid, as follows:

$$Ca_3(PO_4)_2+4HCl=2CaCl_2+CaH_4(PO_4)_2.$$

When this was heated to redness with aluminum powder and diatomaceous earth, the phosphorus, to within a very small fraction, distilled off quietly.*

Phosphorus Alloys.—The addition of a certain quantity of phosphorus to copper enables founders to make a sound casting, but in order to mix properly the phosphorus in the copper considerable skill is required, owing to the extreme combustibility of the phosphorus. A founder in New York introduces the phosphorus into a small crucible in the bottom of which several holes have been bored, the top being then covered over and a piece of paper wound around the bottom. When the copper has the right heat the crucible containing the phosphorus is plunged quickly to the bottom of the pot containing the copper, where the paper is burned, permitting the phosphorus to run out and mingle with the molten copper. The metal is then ready for pouring. Phosphorized copper is said to be especially useful for castings for electrical purposes.

L. Dill, of Frankfort-am-Main, proposes to produce alloys of phosphorus with metals by subjecting phosphoric acid or a concentrated solution of an acid phosphate salt to electrolysis with a carbon electrode and an electrode of the metal to be alloyed. By passage of the current the phosphorus separates and combines with the metal of the electrode, which melts down.† This process may be useful for the production of phosphor bronze.

Lucifer Match Manufacture.—The dangers attending the manufacture of lucifer matches have been recently the subject of an investigation in behalf of the Home Department of the British government, the results of which were summarized in an elaborate report on the "Use of Phosphorus in the Manufacture of Lucifer Matches," by Prof. T. E. Thorpe, Prof. Thomas Oliver and Dr. George Cunningham, published in 1899. In response to a reward offered by the French government for a substitute for yellow phosphorus in making matches, MM. Sevene & Cohen submitted a compound of phosphorus sesquisulphide and chlorate of potash. The sesquisulphide is a gray-yellowish substance, which is prepared by heating amorphous (non-poisonous) phosphorus and sulphur. The substance is very stable; Lemoine, who studied it in 1864, kept it for 15 years exposed to the air without noticing any change. Its latent heat is low; it ignites at 95° C. (203° F.) and can be, therefore, lighted by rubbing, like ordinary phosphorus. The mixture burns quietly, but a mixture of amorphous phosphorus and chlorate of potash is explosive. The new matches have a faint smell, but the employees in the works are said not to complain about it. They do not phosphoresce even when rubbed energetically, but are poisonous to a slight degree.

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* *Stahl und Eisen*, Vol. XVIII., 1898, p. 410.

† German Patent No. 99,128, Oct. 29, 1897.

THE MANUFACTURE OF ACID PHOSPHATE OF LIME.

BY PETER S. GILCHRIST.

THE terms "acid phosphate" and "superphosphate" are used synonymously in the United States, but in reality acid phosphate is the product resulting from the treatment of mineral phosphates (tribasic phosphate of lime) with sulphuric acid, whereas superphosphate is the product resulting from the treatment of the same material with phosphoric acid. The manufacture of acid phosphate has grown to gigantic proportions, the world's estimated production being over 4,000,000 tons per annum, and the demand is continually increasing, owing to the impoverishment of lands by continuous cultivation.

The raw materials used in the manufacture are mineral phosphates, bones, and sulphuric acid. The mineral phosphates are obtained chiefly from the United States, France, Algeria and Belgium; the bones from the abattoirs. The United States is the largest producer of mineral phosphates, as will be seen from the statistical tables preceding this paper. Its production is derived chiefly from three States, namely, South Carolina, Florida and Tennessee. The phosphates from these States vary in composition, those from Tennessee being of the highest grade. Rock from the same district is apt to give varying analyses, but the following represents about an average: South Carolina—phosphate of lime, 59·63%; combined iron and alumina, 6·00%; Florida (land rock)—phosphate of lime, 77·20%; combined iron and alumina, 2·75%; Florida (pebble)—phosphate of lime, 62·20%; combined iron and alumina, 2·35%; Tennessee—phosphate of lime, 80·74%; combined iron and alumina, 2·60%.

Tennessee phosphate is now taking a very prominent place in the fertilizer business in the Southern States owing to its high analysis and close proximity of the mines, while moreover the rock from the Mt. Pleasant district is very soft, and consequently easy to grind.

Chemistry.—The process of manufacturing acid phosphate is not very difficult, but it requires some chemical knowledge to obtain satisfactory results, though this requisite would not be so important if the mineral phosphates did not vary in composition, not only in the percentage of phosphate of lime, but also in carbonate of lime, iron, alumina, etc.; hence it becomes necessary that the manufacturer should be acquainted with the rock he is to use. Having determined which is the cheapest source of supply of the raw material a full analysis should be made, so as to calculate the amount of sulphuric acid theoretically required for combination with each base. The total thus obtained is always more than is used in practice, but it is a close guide for the manufacturer. If the theoretical amount of sulphuric acid were used, it would result in a pasty mass, but there would be no insoluble phosphoric acid. A less amount is therefore taken, so as to insure a product that will dry rapidly. This leaves some of the phosphate of lime undecomposed, which is generally termed insoluble.

The purpose in treating the mineral phosphates with sulphuric acid is not to form free phosphoric acid, but to split up the tribasic phosphate of lime, so as to form acid calcium phosphate (acid phosphate), which is soluble in water, and thus readily available for plant life. In doing this it becomes necessary to remove the foreign acids, such as hydrofluoric, silicic and carbonic, and to act on two of

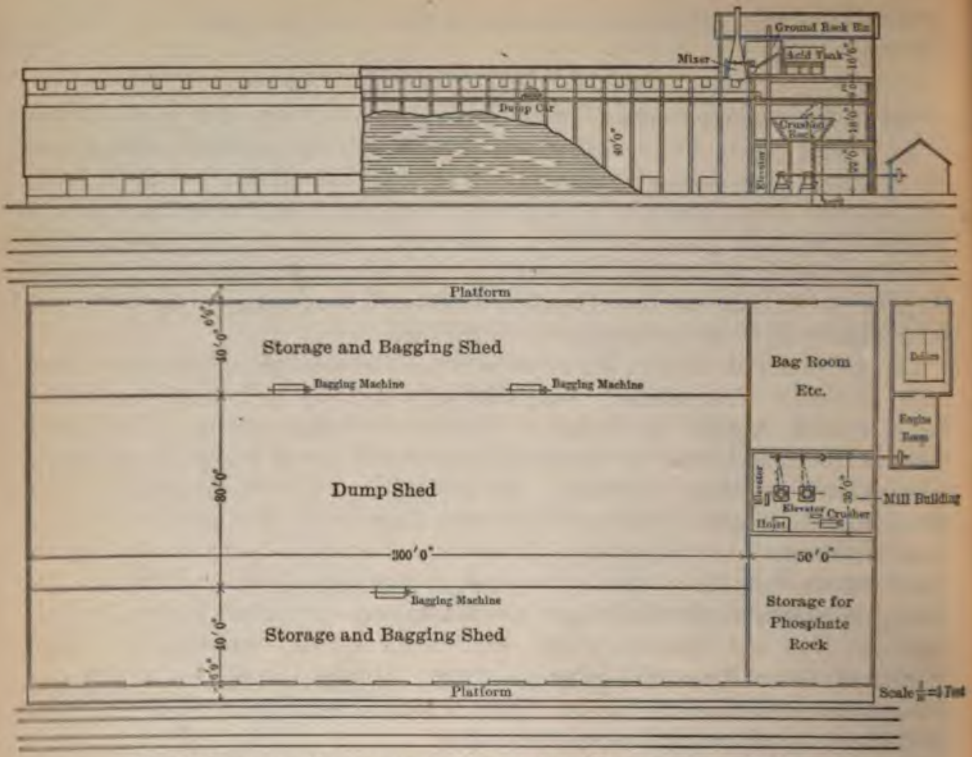


FIG. 1. PLAN AND ELEVATION OF FERTILIZER WORKS, SHOWING TRACKS.

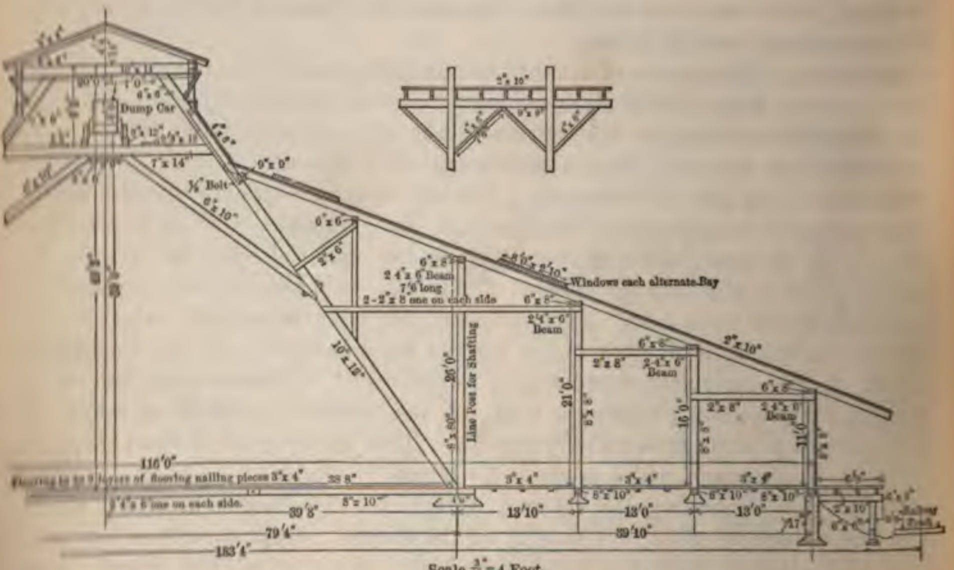


FIG. 2. PARTIAL SECTION OF FERTILIZER BUILDING

The construction and dimensions are the same on each side of the center line.

the three molecules of lime, which are in combination with phosphoric anhydride. The reaction may be stated as follows:

$$Ca_3(PO_4)_2+2\ H_2SO_4+5\ H_2O=CaH_4(PO_4)_2H_2O+2(CaSO_4 2H_2O).$$

If too little sulphuric acid is used, a neutral phosphate of lime is formed, insoluble in water but soluble in neutral citrate of ammonia; if an excess of sulphuric acid is used, all the three molecules of lime are acted upon, forming free phosphoric acid. Some of the mineral phosphates are deficient in carbonate of lime, and this is often overcome by mixing different phosphates, or by adding some natural carbonate. If the carbonate of lime is not in too great excess it is an advantage, since the escaping carbonic acid gas helps to make the mass porous and consequently easier to dry. An excess of iron and alumina is a disadvantage, apart from the excess of sulphuric acid required to neutralize them, since the iron causes the water soluble phosphate to revert to citrate soluble. In the United States this does not make much difference, since both water soluble and citrate soluble phosphate are calculated by the agricultural experimental station chemists as available phosphoric acid, but abroad it is different, reverted phosphoric acid often being not taken into consideration; hence foreign buyers like their mineral phosphates low in iron and alumina.

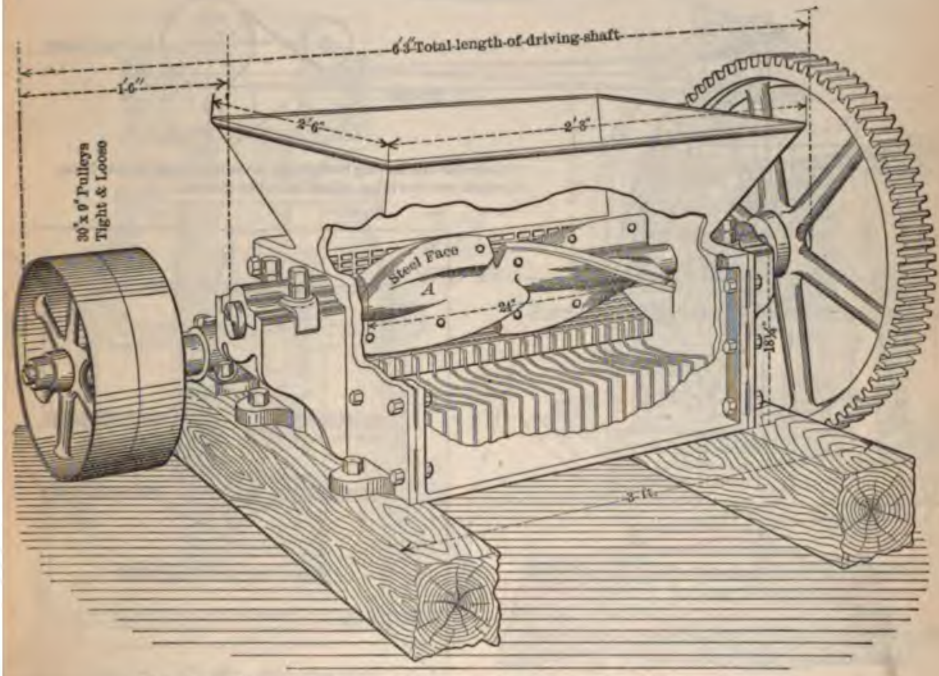
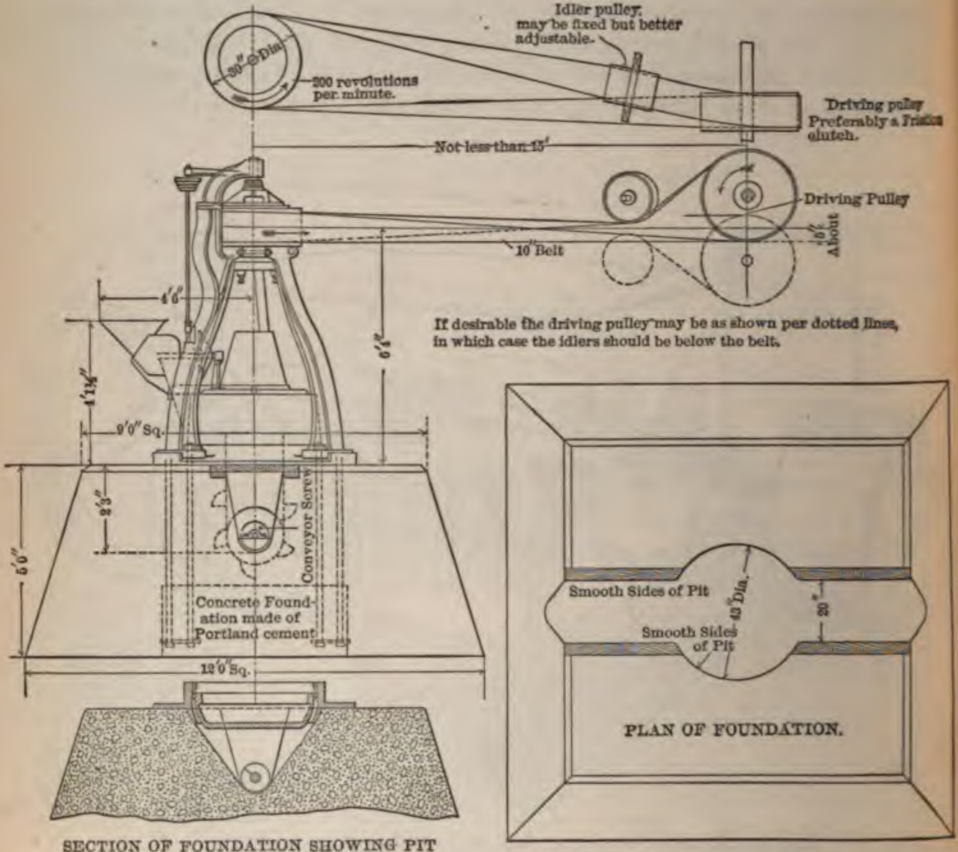


FIG. 3.—ROBERT POOLE & SON CO.'S PHOSPHATE ROCK CRUSHER.

The essential points in the manufacture of acid phosphate are to get as high a percentage of available phosphoric acid as possible, with a minimum of insoluble; at the same time the product must be dry and porous, and made at a minimum cost. In making acid phosphate so much sulphuric acid is used that its cost is a very important item, and it is therefore essential to produce it cheaply. Formerly it was generally made from brimstone, but to-day most plants use pyrites, this being a cheaper source of supply in the United States. Pyrites fines are becoming very generally used, owing to the adoption of the Herreshoff furnace, which handles these ores very successfully.



If desirable the driving pulley may be as shown per dotted lines, in which case the idlers should be below the belt.

SECTION OF FOUNDATION SHOWING PIT FOR RECEIVING GROUND MATERIAL.

FIG. 4.—BRADLEY PULVERIZER CO.'S 30-IN. GRIFFIN DRY MILL.

In setting up the Griffin mill a very strong foundation is necessary. It should be made of concrete, using nothing but fresh Portland cement. The Bradley Pulverizer Co. recommends four parts (by measure) of broken stone, two parts sand, and one part fresh Portland cement. These ingredients should be thoroughly mixed dry, and then just enough water should be added to make a stiff paste, which should be again thoroughly turned over. The concrete should be put in place as soon as mixed and well rammed until the moisture flushes to the top. The stone should be broken so that all will pass through a 2-in. ring, and 25% of the whole should be less than half that size. The sand must be sharp, clean, and free from loam.

Design of Works.—The modern acid phosphate plant is much simpler in construction than its predecessors, owing to the great improvement in the machinery used, especially as regards grinding. It is at the same time less costly to maintain. The plants as now laid out have ample room for operation, and the storage of goods. An arrangement which is much in vogue, especially in the

Southern States, is shown in Fig. 1. It affords ample track room for handling cars, which is important since usually the year's product has to be shipped in two to three months. The plant consists of mill building, where the manipulation of the raw materials takes place, dump shed, bagging sheds, rock storage, bag room, etc.; the engines and boiler rooms are generally situated in a separate building. On the ground floor of the main building are situated the rock crusher and the grinding mill. Fig. 2 shows a section of the dump shed and bagging sheds, with car tracks on each side. This construction is used to obtain a large storage capacity. It allows a pile of acid phosphate to accumulate 40 ft. high. This plant will handle 20,000 tons of acid phosphate, outside of ammoniated goods.

Crushers of the cakes or nodules are generally of the Comet type, or that illustrated in Fig. 3. The object is to get the rock broken as small as possible, so as to lessen the work of the grinding mills.

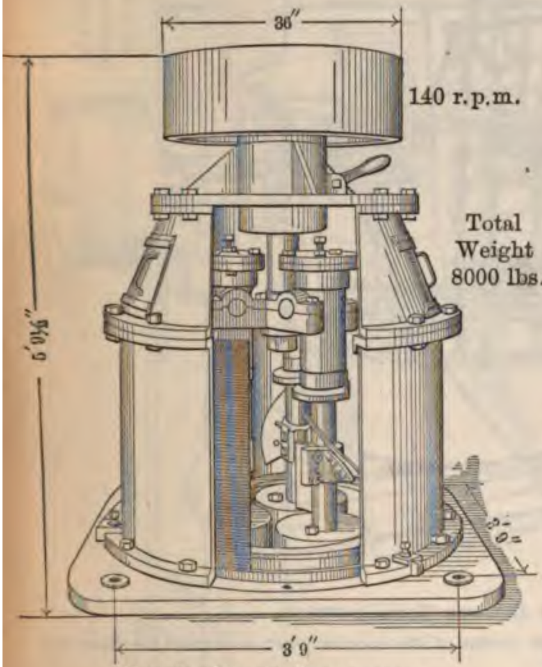


FIG. 5.—NAROD PULVERIZER.

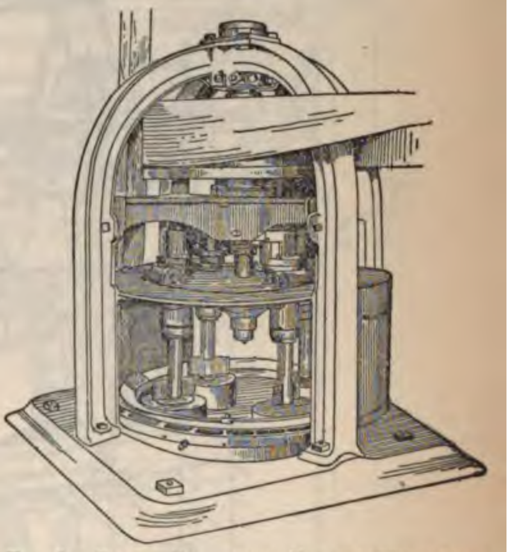


FIG. 6.—CLARK FOUR-ROLL PULVERIZER; WITH SCREEN REMOVED.

Floor space required, 7×7′; height, 6′ 9″; weight 12,000 lb.; driving pulley, 42×14″, 126 r. p. m.; screen surface 30 sq. ft.

The older grinding mills, such as the French buhr, Frisbee-Lucop, etc., are now being replaced by the Griffin, Narod, and Clark, which are giving very satisfactory results. These mills, shown in Figs. 4, 5 and 6, are all of the roller and die type, the principle being the same in each, the difference being only in the number of rolls, and the special manner of driving them. The rolls in revolving exert an enormous crushing pressure, owing to the momentum given them by the speed of the central shaft. The screens are placed above the rolls, so the wear and tear upon them is reduced to a minimum. The screens used are such as to give a product between 70 and 80 mesh. All of these mills are satisfactory, though each style has its own admirers, the capacities being between 20 and 30 tons per day, according to the nature of the rock. The repairs are not heavy, and are easily made.

The engines and boilers are generally installed in a separate building, power being transmitted to the mill building by heavy shafting. On the second floor of the latter there is a large rock bin, for feeding the mills, into which the broken rock is discharged from the crusher by an elevator. On the third floor is the motor for driving the dump car. This is done by an endless cable worked on a drum, which is controlled by the operator. On the fourth floor is the acid phosphate mixer, ground rock bin, and acid tank. The mixer is usually a large revolving pan, 7 or 8 ft. in diameter, capable of holding over a ton to a charge, in which the agitators revolve, the double agitator being preferable to the single, since the charge is more rapidly mixed. Fig. 7 shows a type of double mixer.

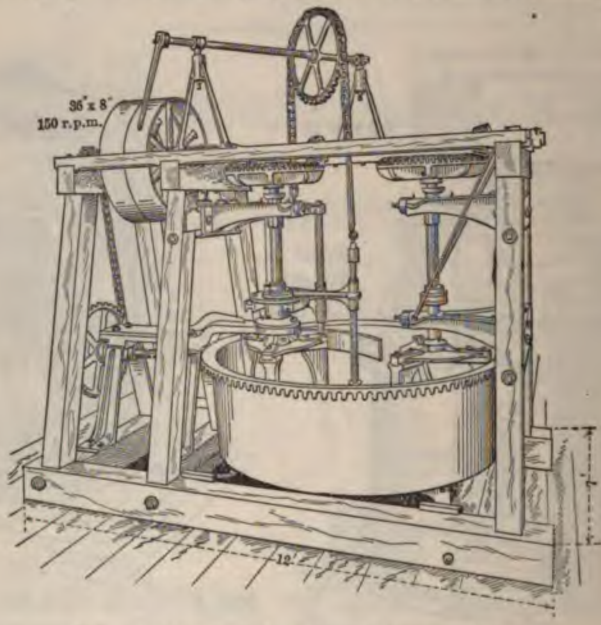


FIG. 7.—WALKER & ELLIOTT DOUBLE MIXER.

The pan is 7' in diameter and 24" deep. The total weight of the machine (which is intended for either wet or dry mixing) is 10,000 lb.

The best position for the mixer is in an extension of the top floor over the dump shed, so that in case of leakage of acid from the pan no damage may result to the machinery, and any leakage will fall into the dump shed, and be absorbed in the phosphate. The mixer has a vent-hood over it, passing through the roof, to carry off the noxious vapors from the operation, and to keep the mixing room free from gas. Sometimes a fan is used to draw the gases off, but this is not necessary if the vent-flue is large enough. Under the mixer there is a track which extends the whole length of the dump shed, on which runs an automatic dump car for removing the fresh acid phosphate from the mixer to any desired point in the dump shed. The car is usually built of heavy wrought iron, to withstand the action of the acid.

In the storage sheds there are bagging machines, frequently termed smitherers, which put the dry phosphate into proper mechanical condition. These machines consist of beaters in a revolving screen, the coarse particles being returned to the feeding elevator, while the finished product is run into bags. They are shown in Fig. 8.

The Operation as Practiced in America.—The mineral phosphates are put into the crusher and broken to ½-in. size, and smaller. The crushed rock is elevated to the storage bin, whence the mills are fed. Sometimes it is screened so as to remove the product already fine enough, the amount thus obtained being often quite considerable, especially when soft phosphates are used. Moreover, the removal of the fine particles at this stage increases the grinding capacity of the mills. In the mills the rock is ground as fine as it can be done with economy, usually to about 80 mesh, but the finer the better. The harder the rock the finer it should be in any case; otherwise chamber acid (50°B.) will have some difficulty

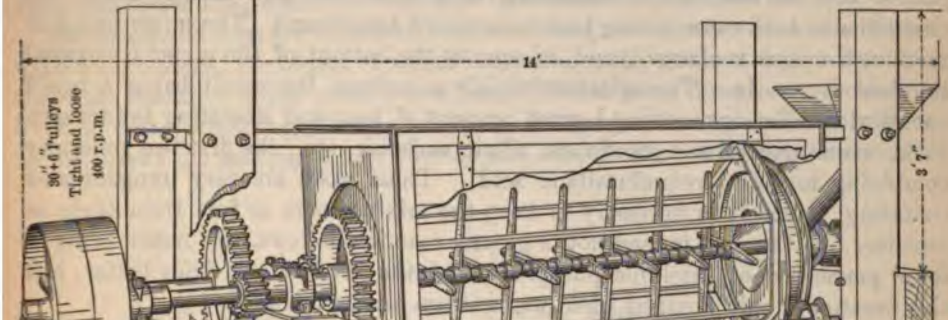


FIG. 8.—ROBERT POOLE & SON CO.'S SMITHERING OR BAGGING MACHINE FOR FERTILIZERS.

in acting upon it. From the mills the ground material is elevated to a storage bin on the top floor, of sufficient size to prevent any interruptions in working. On this floor is also the large acid tank, holding about 40 tons. The strength of the acid used is commonly 50° B., but some manufacturers employ it as low as 48° and some as high as 52° B.

Before charging the mixer a certain amount of ground rock is run out of the storage bin into a receptacle on scales, and weighed, and a certain amount of sulphuric acid is run into a measuring tank from the acid storage tank, the quantities in both cases having been determined beforehand. The mixer being in motion, the rock is charged, and as soon as the bottom of the mixer is covered the acid is run in. The agitators rapidly mix them, the result being a rapid chemical reaction, generating a great amount of heat and liberating the foreign acids, which pass off as hydrofluoric, silicic, carbonic, etc., the first two generally combining to form hydrofluosilicic acid. These gases are very pungent and irritating; hence it is necessary to keep the mixing room as free from them as possible. If the acid is used hot, a greater reaction follows, the result being a lower percentage of insoluble phosphate, and the product also dries better, but this practice is only justified when waste steam can be utilized.

The reaction in the mixer causes the mass to swell, and toward the end it begins to settle down again, after which it is in condition to remove. The center plug which revolves with the pan is then raised, and the seething mass is dropped into the dump car and trammed into the dump shed, the whole operation in the mixer not taking over five minutes, one man mixing as high as 150 tons in a day, and one charge being usually a ton. When conveniently arranged, one man can do everything, the rock and acid being got ready while the reaction in the pan is going on, but usually two men are employed in the mixing room, and one to work the dump car. In dumping acid phosphate, it is advisable to keep it together as much as feasible, so that it may retain the heat, and continue the reaction as long as possible, whereas if spread over much ground it cools rapidly, and the reaction virtually stops; but lying in a large mass, the insoluble phosphate is always found lower after standing for some time.

There is a loss of about 7% on the raw materials charged, part being carried into the atmosphere from the mixer with the steam, the remainder being lost as moisture in drying out while in the dump shed. The amount of sulphuric acid used varies from 850 to 1,000 lb. per 1,000 lb. of rock. Tennessee rock frequently takes equal proportions, while Florida and South Carolina rocks require less. Each manufacturer tries to get as low a percentage of insoluble phosphate as possible, and to this end some have used a little muriatic acid with the sulphuric; the insoluble has thereby been made less, but on the other hand the product is apt to become deliquescent.

Practice in Europe.—In Europe the operations are carried out a little differently. Some of the grinding mills described above are being used with advantage, but the mixing of the ground rock and acid is done in close vessels, and the acid phosphate is dropped into air-tight dens, the dump car not being used.

One of these mixers is shown in Fig. 9. It usually consists of large cast plates, well put together, forming a tight trough, 9 to 10 ft. long, 3 ft. wide, and about 4 ft. deep, through which passes a large shaft with the bearings on the outside.

To this shaft are fastened paddles, which revolve rapidly with the shaft during the mixing operations. The top of the mixer has a tight cover, on which sits the hopper for putting in the ground rock, and a connection with the acid tank; there is also a gas outlet pipe. At the bottom of the mixer there is a discharge valve, connecting the mixer with the dens. The mixer sits over two to four dens, which are built very strongly to resist the pressure following upon the mass cooling. The mixing operations take two to three minutes, when the acid phosphate is discharged in a semi-liquid state into one of the dens. This is continued until the den is full, when the discharge is changed into another one.

The mixer and dens are connected with scrubbers, and these in turn are connected with a high stack, the object of which is to remove the noxious vapors resulting from the reactions going on in the mixer. This is being done very successfully by the use of a settling chamber between the mixer, dens, and the scrubbers, which allows the settling of the fluorine and silicon compounds by the excess of steam present, thus giving greater efficiency to the scrubbers, since it prevents choking. To make this method effective it is absolutely necessary to keep the dens air-tight; otherwise the gases would be so dilute that the bulk would be difficult to handle. Sometimes the water leaving the scrubbers becomes so acidulated that it has to be neutralized with lime before allowing it to pass into the drains, but if sufficient water is used this should not be necessary.

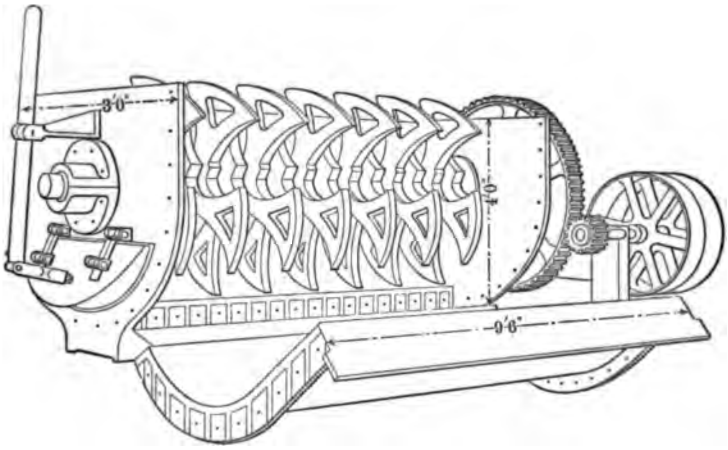


FIG. 9.—ACID PHOSPHATE MIXER, ENGLISH STYLE.

Manufacture of High-grade Superphosphate.—There is little high-grade superphosphate made in the United States, owing to low freights, and the many works scattered throughout the country, but abroad it is beginning to find favor, especially where freights are high. The process for making high-grade superphosphate is more difficult, especially in the presence of much iron and alumina. The phosphates, finely ground, are treated with diluted sulphuric acid, so as to decompose three molecules of lime, thus liberating the phosphoric acid. The liquor is then filtered from the sulphate of lime, and evaporated to about 45° Baumé, from which point the operation goes on the same as in the manufacture of acid phosphate, except that phosphoric acid is used in lieu of sulphuric acid, and at the finish the product has to be dried before it can be put through the bagging machines. Bones are treated the same as mineral phosphates.

Preparation of Acid Phosphate for Shipment.—In the United States the acid phosphate usually lies for several months where dumped, so by the time it is used it becomes very dry, and in good condition for manipulating in the bagging machines, as shown in Fig. 8. Many manufacturers are now using a disintegrator, or cylinder pulverizer, with an elevated shaking screen, in lieu of the bagging machine, its capacity being larger, 150 to 200 tons being put through in a day. Attached to these machines is frequently an automatic weighing machine. If the acid phosphate is damp it balls in manipulation, working into a soft pasty mass, which sticks to the machines, necessitating frequent stops. In the United States the acid phosphate is generally put up in 200-lb. bags, 100-lb. bags being used occasionally. Each bag has to be branded with the guaranteed analysis and the name of the maker, and a State tag has to be attached to each one. A tax of 25c. per ton is levied in some States, independent of the analysis. The composition of acid phosphate from the different rocks is as follows: Tennessee, 17 to 18% available phosphoric acid, (P_2O_5); Florida, 14 to 17%; South Carolina, 13 to 14%.

Many manufacturers add ammoniates and potash in the mixer and thus make complete fertilizers at one process. This is the most satisfactory way, since the ingredients are thoroughly mixed, and it avoids the discrepancies in analysis, but materials such as nitrate of soda should not be added in wet mixtures, as in that case there would be loss of nitrogen.

PLATINUM AND IRIDIUM.

THE domestic production of platinum is exceedingly small, being only that which is derived as a by-product in the refining of gold from certain localities on the Pacific Coast, iridium being obtained in conjunction therewith. The United States Mint is practically the only producer of this domestic product. In 1898 the New York Assay Office sold 290·64 troy oz. of iridium and 749·53 of platinum, which were accumulations since 1894. The San Francisco Mint produced 8·5 oz. of iridium in 1898. In 1897 we reported a production of 200 oz. of platinum and 20·25 oz. of iridium. The production of platinum in California in 1898 was 300 troy oz., against 150 in 1897, 162 in 1896, and 150 in 1895. Most of this has been derived from Trinity, Shasta and Plumas counties.

In 1898 a discovery of platinum ore near Leavenworth, Wash., was reported, but investigation showed that there was no truth in the statements which were widely circulated. A discovery of low-grade platiniferous sand in Alaska was really made, however, and another was reported by W. E. Everette, of Dawson, N. W. T., who states that the black sand at the mouth of Hootalinkwa and Thirty Mile (or Lewes) rivers contains a large amount of platinum, one sample yielding 97 oz. platinum and 2 oz. iridosmine per ton of black sand. A good deal of platinum is brought into the United States in the nickel-copper matte from Sudbury, which contains about 1 troy oz. of platinum to 1,000 lb. avd. of nickel, but there has not yet been any considerable production of platinum from this source. A fact which is not generally known is that this matte contains about 2 oz. of palladium for every 3 oz. of platinum, and the value of its palladium contents is almost double that of the platinum.

Besides the United States Mint there are four concerns in the United States which refine platinum, importing the crude metal from Siberia and Colombia. The returns of the Bureau of Statistics do not show the extent of this business, the imports of platinum being grouped as "unmanufactured and ingots, bars, sheets and wire," but under this classification there was in 1898 an importation of 6,703 lb. avd., valued at $1,179,242, against 5,697 lb., valued at $960,299, in 1897, and a large part of these importations is refined in the United States. The domestic consumption of platinum in 1898 was larger than for many years, which was attributed to the increased activity in the electrical and incandescent gas lighting industries. The average price of bar platinum at New York in 1897 was $12·34 per troy oz.; in 1898 it was $12·79.

PRODUCTION OF PLATINUM IN FOREIGN COUNTRIES.

Platinum is produced commercially only in New South Wales, Colombia and Russia (Siberia), the production of the last being by far the most important. A small amount of platinum is produced annually in British Columbia under the same conditions as in the United States. The existence of platinum in the basin of the Uelle River in the Congo has been recorded by Dr. Schweinfurth.

Colombia.—There are no official statistics of the production of platinum in this republic, but according to our private information there was an output of 11,700 troy oz. in 1897, against 10,320 oz. in 1896. The production in 1898 was probably not materially different from that of the previous year.

New South Wales.—The production of crude platinum at Fifield in 1898 was 2,150 troy oz., against 1,966 in 1897 and 2,900 in 1896. Work in this field is carried on by individual miners or small parties of two or three, and the yield is dependent upon the amount of water available for mining purposes, many claims having been abandoned on this account during the last few years, while others have been worked only half time.

Russia.—The platinum industry in Russia was considerably excited in 1898 on account of the protest of the producers against the refiners' monopoly, the purchase of the principal mines by a Parisian syndicate, and the establishment of an independent refinery near the mines. The condition of the industry up to the summer of 1898 was described by N. P. Steinfeldt of Ekaterinburg, in an article in the *Torgovo Promishlennaia Gazeta* (*Gazette for Trade and Manufactures*) of August 23, 1898. At that time the value of crude platinum at the mines was 9,000 to 10,000 rubles per pood.* Nine years previous the price for platinum was so low that the metal could be produced only by persons of small means who were satisfied with small profits or merely with employment, and carried on the business in a simple and primitive manner. About that time the demand for platinum increased and the representatives of the foreign refiners advanced to many of the platinum miners the necessary capital to carry on the business on a more extensive scale, with the stipulation that they should sell their product at a price of 7,000 rubles per pood for a fixed period which has still five years to run, the miner paying the refiner interest on the money advanced; consequently the refiners have derived all the benefit from the advance in the price. Under ordinary circumstances 7,000 rubles per pood covers the cost of mining; but sometimes it does not, and anyway there is not much chance for profit to the miner. Most of the miners of the Ourals have been suffering from this state of affairs, the few who did not enter into these contracts not having been sufficiently strong to make their own terms. The Gold and Platinum Miners' Assembly of the Oural applied to the government for assistance, praying also for the erection of a government refinery, but the Department of Mining was not able to grant these requests, although arrangements were made for a meeting of the platinum miners at St. Petersburg to discuss measures to improve the condition of the industry.

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* The pood may be taken as equal to 526·6 troy oz.

POTASH AND POTASSIUM.

The world's supply of potash salts, with the exception of saltpeter, continues to be derived from Germany, where the production is still controlled by the Kali-Syndicate, which regulates the output of each mine and fixes the prices for all markets. With December 31, 1898, the agreement between the mines, under which the syndicate operated for 10 years previous, came to an end, and a new contract, extending for three years, was entered into to govern the potash industry. According to the new contract each manufacturer is limited as to the amount of potash salts which he can put on the market, while under the old contract the limitations were placed on the amounts of crude salts mined. The different products are now grouped in four classes according to their percentage of potassium, as follows: (1) Products with more than 48% K, an equivalent of 76·1% KCl or 88·9% K_2SO_4. (2) Products with not more than 48% K, nor less than 18%, equivalents of 76·1 to 28·5% KCl or 88·9 to 33·3% K_2SO_4. (3) Crude salts (not carnallite) with 12·4 to 18% K, equivalents of 19·7 to 28·5% KCl or 23·0% to 33·3% K_2SO_4. (4) Carnallite salts with less than 12·4% K. The percentage of potash is made the basis for determining the amount of salts of each kind which any manufacturer may offer for sale, but no restrictions are put on the output of the crude material so long as it does not come into the market. Class 1 includes mostly refined products; class 2, both refined and crude; classes 3 and 4, only crude. A method for separating classes 3 and 4 was adopted, and will prevent carnallite salts which contain a high percentage of potash being sold under class 3. It is based on the relative solubilities of carnallite and kainite in 96% alcohol, the magnesium chloride in the kainite being insoluble, while it is soluble when occurring in carnallite. Salts which contain more than 6% soluble chlorine are put in class 4.

The potash industry in Brunswick and Hanover is at present experiencing the evil results of wildcat speculation. Over 100 prospecting companies have been operating in this field, and a large number of new works were built and equipped on the strength of their reports. The failure of many of these concerns has led to a slump in prices which affects the whole industry. One bed of salts of supposed enormous thickness, from which four companies expected to draw their supplies, proved to be a thin vertical seam. Some beds pinch out quickly, while mining in others is rendered unprofitable by difficulties met with in handling the water.*

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* Dr. Precht, *Zeitschrift für Angewandte Chemie*, Feb. 21, 1899, pp. 185–186.

The statistics of the Stassfurt salt industry are shown in the following tables, which we owe to the courtesy of the Kali-Syndicate:

SALES OF CRUDE POTASH SALTS. (IN METRIC TONS.)

Year.	Rock Salt.	Carnallite.	Kieserite.	Hartsalz Schoenite.	Kainite.	Sylvinite.	Boracite.
1857	12,797	....	....	....	....	....	....
1858	25,567	....	....	....	....	....	....
1859	15,386	....	....	....	....	....	....
1860	31,863	....	....	....	....	....	....
1861	40,314	2,293	....	....	....	....	....
1862	47,045	19,726	20	....	....	....	....
1863	42,402	58,303	68	....	....	....	....
1864	46,511	115,408	88	....	....	....	4
1865	45,027	87,670	74	....	1,313	....	8
1866	49,128	135,553	413	908	4,900	....	14
1867	56,153	141,604	1,143	624	8,351	....	10
1868	71,945	167,336	1,417	308	10,463	....	18
1869	65,201	211,883	226	....	16,857	....	26
1870	52,018	268,225	70	1,423	18,877	....	16
1871	50,154	335,944	47	3,886	32,695	....	15
1872	55,334	468,537	22	215	17,851	....	25
1873	64,341	441,078	7	6	6,094	....	25
1874	71,072	414,961	16	....	9,751	....	12
1875	77,105	498,736	5	304	23,818	....	11
1876	76,656	563,669	145	161	17,776	....	24
1877	80,525	771,819	151	2,170	33,305	....	44
1878	101,694	735,750	519	1,522	32,480	....	97
1879	107,471	610,427	760	1,379	48,827	....	104
1880	118,170	528,211	892	1,695	137,795	....	107
1881	149,257	744,726	2,081	3,028	155,301	....	116
1882	141,338	1,059,299	4,658	3,719	144,757	....	125
1883	152,746	950,203	11,790	2,216	226,600	....	205
1884	180,818	739,958	12,388	14,102	203,004	....	159
1885	212,082	644,709	11,969	26,458	245,911	....	142
1886	233,543	638,229	13,917	6,775	240,552	....	149
1887	201,962	840,206	14,185	57	237,570	....	150
1888	191,595	849,602	10,753	56,462	319,110	2,220	169
1889	259,286	798,721	9,354	41,916	320,694	28,328	139
1890	302,205	838,525	6,951	43,745	358,125	31,918	164
1891	365,910	818,862	5,815	45,249	467,244	32,661	180
1892	293,247	736,750	5,782	40,689	545,084	32,669	166
1893	264,417	794,659	4,807	41,679	648,315	49,139	187
1894	281,246	851,338	3,864	38,514	690,786	63,494	169
1895	259,424	782,944	3,012	20,388	649,152	76,097	145
1896	277,884	856,223	2,841	3,339	829,686	90,300	195
1897	288,036	851,272	2,619	36,372	975,814	84,105	184
1898	291,591	990,298	2,444	37,139	1,083,476	94,270	253

UTILIZATION OF THE CRUDE POTASH SALTS. (IN METRIC TONS.)

Year.	Kainite.			Carnallite.			Kieserite.		Sylvinite.		
	For Agricultural Purposes.		For Manufacturing Concentrated Salts.	For Agricultural Purposes.		For Manufacturing Concentrated Salts.	For Agricultural Purposes.		For Agricultural Purposes.		For Mfg. Concentrated Salts.
	Germany	Elsewhere.		Germany	Elsewhere.		Germany	Elsewhere.	Germany	Elsewhere.	
1880	23,768	103,749	10,278	3,244		524,967	892		....	....	....
1881	20,372	119,491	15,438	4,820		739,905	2,081		....	....	....
1882	30,413	95,263	19,080	5,590		1,053,709	4,658		....	....	....
1883	48,138	153,199	25,262	5,643		944,559	11,790		....	....	....
1884	48,643	109,655	44,704	6,265		733,693	12,388		....	....	....
1885	50,870	143,517	51,523	7,018		637,691	11,969		....	....	....
1886	65,835	105,050	69,665	8,811		689,417	13,917		....	....	....
1887	84,493	89,293	63,784	16,706		823,500	14,185		....	....	....
1888	105,236	142,060	71,813	20,967		828,580	10,753		....	110	2,110
1889	150,176	100,828	69,689	28,719	55	769,947	9,027	326	165	12,280	15,882
1890	176,748	117,415	63,961	27,923	72	810,529	6,650	300	1,283	9,565	21,065
1891	237,988	163,305	65,949	33,487	141	785,233	5,405	410	2,012	10,201	20,447
1892	363,223	122,370	59,490	40,114	723	695,912	5,252	530	3,437	9,541	19,690
1893	425,366	173,093	49,854	54,987	3,173	732,233	4.477	310	3,524	11,264	34,350
1894	462,461	183,910	44,414	57,098	4,046	790 192	3,794	70	3,746	16,330	43,418
1895	434,460	171,348	43,343	47,536	3,816	731,581	2,992	20	2,298	19,557	54,242
1896	555,066	221,949	52,672	53,750	3,913	798,560	2,791	50	2,471	23,101	64,818
1897	665,758	266,876	43,180	55,949	5,133	790,190	2,595	24	3,207	28,264	52,634
1898	718,605	302,113	62,757	58,461	7.076	925,460	2,331	112	3,510	31,997	58,762

PRODUCTS. (IN METRIC TONS.)

Year,	Potassium Chloride.	Potassium Sulphate.	Sulphate Potassium-Magnesium.		Kieserite in Blocks.	Manure Salt.	38% Potash Manure Salt.
			Crystalliz'd	Calcined.			
1878......	110,760	....	....	....	....	....	....
1879......	92,401	....	....	....	....	....	....
1880	96,831	....	....	....	....	....	....
1881......	115,266	....	....	....	....	....	....
1882......	152,960	....	....	....	....	....	....
1883......	136,000	....	....	....	17,500	....	....
1884......	106,330	3,000	400	8,000	17,800	9,500	....
1885......	104,500	4,000	450	9,000	18,500	8,400	....
1886......	110,200	3,638	472	10,111	19,500	8,161	....
1887......	130,000	10,527	500	6,284	24,018	8,163	....
1888......	132,000	10,916	522	11,380	28,325	14,016	....
1889......	131,592	7,321	671	9,214	31,823	19,138	....
1890......	134,759	13,839	907	10,830	32,004	13,833	....
1891......	143,487	18,980	1,052	11,399	28,559	16,078	....
1892......	121,028	15,465	708	11,842	23,854	16,895	....
1893......	132,528	16,361	739	12,642	24,385	16,809	535
1894......	147,936	15,242	1,779	12,718	30,607	18,425	1,302
1895......	134,538	13,403	898	8,248	25,115	18,685	1,040
1896......	147,680	13,889	1,051	4,622	24,988	16,418	2,836
1897......	142,314	15,403	922	7,415	25,669	18,979	4,063
1898......	155,927	17,781	914	10,535	19,934	19,000	4,781

Prices.—The prices established by the syndicate for the year beginning March 31, 1899, for Boston, New York and Philadelphia per 100 lb. are as follows: Muriate of potash, 80 to 85%, basis 80%, \$1·75; muriate of potash, minimum 95%, basis 80%, \$1·78; sulphate of potash, minimum 90%, basis 90%, \$1·955; sulphate of potash, minimum 96%, basis 90%, \$1·985; double manure salt, 48 to 53%, basis 48%, \$1·005; manure salt, minimum 20%, potash, 64%, manure salt, minimum 30%, potash, 87c. For Norfolk prices, add 1·5c. per hundred to the foregoing quotations except for double manure salt and 30% manure salt, to which add only 1c. For Charleston, Savannah, Wilmington, N. C., and New Orleans add 3c. to New York prices, except 20% manure salt, for which add only 2·5c., and 30% manure salt for which add 2c. For bulk salts, kainite is quoted at \$8·45 per ton of 2,240 lb., invoice weight at shipping port, or \$8·70, actual weight at receiving port; silvinite, 36·5c. per unit of sulphate of potash, invoice weight at shipping port, or 37·5c. per unit, actual weight at receiving point. Kainite prices for Norfolk are 35c. per ton higher than New York, and at Charleston and other Southern ports 50c. higher, while silvinite is respectively 1c. and 1·5c. per unit higher. These prices are for not less than 500 tons of bulk salts or 50 tons of concentrated salts, and are based upon shipment by river from the mines to seaport. For shipments by rail 40 pfennigs per 100 kg. extra is required.

Imports.—The importation of potassium products into the United States is shown in the following table:

	1895.		1896.		1897.		1898.	
	Pounds.	Value.	Pounds.	Value.	Pounds.	Value.	Pounds.	Value.
Potassium nitrate...	11,419,090	\$305,207	20,085,327	\$479,899	16,276,352	\$306,696	11,461,823	\$281,504
Potassium muriate..	81,833,531	1,296,184	88,525,983	1,392,504	108,839,049	1,683,472	104,358,601	1,620,720
Potassium chlorate..	4,917,636	456,342	4,659,793	366,300	6,508,359	461,620	4,806,402	288,965

Wood Potash.—The supply of potash was formerly derived from wood ashes, but the competition of the Stassfurt salts has practically put an end to this industry. However, according to the *Canadian Journal of Commerce*, Canada still produces a considerable quantity of potash from wood, and this product has peculiar advantages which are likely to keep it always in demand for soap-making. A large quantity is employed in manufacturing soap used extensively by the British navy. The following shows the receipts of potash for inspection during the last five years, in barrels:

	1894.	1895.	1896.	1897.	1898.
Potash..............	1,936	1,904	1,964	1,404	1,080
Pearlash............	224	415	295	176	243
	2,160	2,319	2,259	1,580	1,323

The falling off in the exports of potashes and pearlashes in recent years was mainly caused by adulteration and other improper practices. Potash is peculiarly liable to deterioration by any admixture of common salt. So sensitive is it in this respect that potash made from vegetables or trees grown where the water is saline is thereby injuriously affected. Yet some of the producers of Canadian potash practice this adulteration, and from this reputation the trade in the product has suffered severely. Prior to seven or eight years ago the inspection of potash was compulsory, the result being that Canadian potash held the first position in Europe for excellence and regularity of quality, but now the inspection is voluntary, not compulsory, and the effect has been a lowering of the Canadian standard of quality, and consequent loss of trade.

Saltpeter.—The world's consumption of saltpeter or niter, potassium nitrate, is derived mainly from India, where it is produced chiefly in the district of Behar, in the Province of Bengal, and to a less extent in the northwest provinces. In Saran, Behar, alone, there were in 1897, for the refining of saltpeter, 166 works, which employed 3,320 persons. The mineral occurs as a natural efflorescence on the surface of the ground, in which condition it is mixed with several other salts, chiefly potassium and sodium chloride, calcium nitrate and sodium sulphate, the last being usually the most abundant. The saltpeter lies in patches, some parts of the efflorescence being more productive than others. The area over which saltpeter is manufactured in India has been estimated at 232,314 sq. miles. According to the census of 1891 there were 119,558 saltpeter workers and sellers in that year. The crude mineral is refined by solution in water, filtration to separate the clear liquor and evaporation to the point where crystallization will take place upon cooling, the foreign salts being removed by precipitation in the processs of boiling. The crystals of pure saltpeter, having been washed with water or alum water, drained and dried, are ready for market. In 1892-93 the Empire produced 48,800 long tons of crude saltpeter, of which 47,580 were refined, yielding 27,580 tons of crystals. The exports of saltpeter from India since 1882-3 have been as follows:

Fiscal Year.	Long Tons.	Fiscal Year.	Long Tons.	Fiscal Year.	Long Tons.	Fiscal Year.	Long Tons.
1882-83.........	19,978	1886-87.........	19,879	1890-91.......	19,985	1894-95.......	17,649
1883-84.........	24,583	1887-88.........	19,320	1791-92.......	19,459	1895-96.......	21,088
1884-85.........	22,596	1888-89.........	21,025	1892-93.......	22,197	1896-97.......	26,423
1885-86........	20,109	1889-90.........	21,111	1893-94.......	16,097		

The consumption of saltpeter in the United States is supplied chiefly from India. Prices in 1898 did not fluctuate very widely until the second quarter of the year, when it was declared contraband of war. On July 1 crude saltpeter was quoted in New York at 5c. for stock on spot and afloat, while refined commanded 6½@7c. per lb. The visible supply on July 1 was 17,887 bags, or 27,916 bags less than at the corresponding period in 1897. The stocks in hands of importers at New York on July 1 amounted to only 5,000 bags, as against 20,500 bags in 1897. Business in October was limited, but prices were steady at 3⅛@3¼c. per lb. for spot crude, and 3·15c. for shipment. The imports from January 1 to December 31 amounted to 62,449 bags, against 77,137 bags in 1897. The consumption during this period amounted to 75,095 bags, against 81,109 bags in 1897. The stock on hand December 31 was 2,822 bags, against 15,468 bags in 1897.

Potassium Chlorate.—Potassium chlorate was produced electrolytically in the United States in 1898 by the National Electrolytic Co., of Niagara Falls, and the North American Chemical Co., of Bay City, Mich., the latter being the American branch of the United Alkali Co. The Bay City production in 1898 was small, but it is expected that in 1899 it will amount to 35,000 kegs. The works of the National Electrolytic Co. ran continuously throughout the year, using 1,100 h. p. The process employed at its works is the invention of W. T. Gibbs, of Buckingham, Quebec. Potassium chlorate was quoted in New York in January at 8¾@9⅜c. per lb., an increase of 1@1⅜c. from the price a year previous. The war sent prices up to 18c. Importations were stopped for a time, but the scare passed and in July prices fell, and purchases were made at 8¼@8⅜c. per lb. A combination was effected among the English and continental makers to govern 1899 trade, and quotations in July were 9¼@9½c. In October prices were 9¼@10¼c., and December closed at 9⅛@9½c. per lb.

J. B. C. Kershaw described* the electrolytic potassium chlorate works of the Société d'Electro-chimie, at St. Michel, in the southeast of France, where 3,000 to 4,000 h. p. are used in the manufacture of chlorates of potassium and sodium by the Gall & Montlaur process, which in principle is the old process of Watt patented in 1851. A solution of an alkali metal chloride is electrolyzed between insoluble anodes at a moderately high temperature and the less soluble chlorate salt, formed by the reaction between the free chlorine at the anode and the caustic alkali formed at the cathode, is separated by crystallization. The anodes are of an alloy of platinum and iridium, containing 90% of the former, as this stands the action of chlorine better than pure platinum; the cathodes are of an iron-nickel alloy and are covered with an asbestos cloth to carry off the hydrogen, of which 19,000 cu. ft. are produced for each ton of chlorate obtained. The details of the process are kept secret; no external

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* *Electrical Review*, London, Nov. 25, 1898.

heat is required, as the current keeps the mass at a sufficiently high temperature. The presence of metallic oxides must be carefully avoided, as they decompose the product. The latter requires re-crystallization before it is ready for the market. In addition to these works the same company has a chlorate factory at Vallorbes, in Switzerland, where 3,000 h. p. is used, the two works together producing 1,800 tons of chlorate per annum. At first the output per electrical horse-power day was only 1 kg. against 2·69 required by theory, but the yield has now been considerably increased. Kershaw estimates that 1 lb. can be produced for 5·5c.

Two-thirds of the world's production of potassium chlorate is now said to be produced by electrolytic processes, there being works using these processes at St. Michel, Chedde, Vallorbes, Bitterfeld, Leopoldschall and Mansboe on the Continent, and at Niagara Falls, N. Y., and Bay City, Mich., in the United States.

ESTIMATION OF POTASSIUM.—According to Dr. Precht the method of determining potash with perchloric acid as proposed by Dr. Wense has been thoroughly tested in the Stassfurt works, and its value confirmed. In this process the solution of potassium salts, after the sulphuric acid has been removed by precipitation with barium chloride, is treated with an excess of perchloric acid and evaporated to dryness. After dissolving in alcohol containing 0·2% of perchloric acid and filtering, the potassium is determined by the usual platinum method. The advantages in this process are: (1) No great care is needed in precipitating the sulphuric acid, as an excess of barium chloride is not injurious; (2) it is cheaper than the platinum method; (3) larger quantities of the potassium salts may be employed in the analysis.

Mercier described* a method of determining potassium by precipitation and weighing of the metallic potassium in K_2PtCl_4. After obtaining the precipitate in the usual manner, it is dissolved in boiling water, and Hg_2Cl_2 added little by little to the boiling solution, until an excess is deposited on the bottom of the beaker. After five minutes' boiling 1 or 2 c.c. HCl are added and it is boiled again. On cooling and filtering, all the Pt is found in the precipitate, which is washed, ignited and weighed, affording pure Pt. The weight multiplied by 0·4835 gives K_2O present.

C. C. Moore found that the usual procedure is much shortened by the addition of the hydrochlorplatinic acid to the acid solution of the substance, after filtration from any insoluble residue. The platinum compound is added only in sufficient quantity to combine with the potash, and, after evaporation almost to dryness, the double potassium-platinum salt is washed with acidulated alcohol (made by passing dry hydrochloric acid gas into cool 90% alcohol), then with ammonium chloride solution and finally with 85% alcohol. The double salt is then dried and weighed. Ammoniacal salts must first be destroyed.†

A. L. Winton and H. J. Wheeler investigated the Lindo-Gladding method for determining potash, and concluded that the objections made to the process are, so far as practical considerations go, without foundation.‡

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* *Bull. de l'Ass'n Belge des Chemistes*, 403 M. † *London American Chemical Society*, XX., pp. 340-343.

‡ *Idem.*, pp. 597-609.

QUICKSILVER.

The production of quicksilver in California in 1898 showed a large increase, but there was a considerable falling off in the sales, which was due to the lack of water in the gold mining districts of the Pacific Coast, causing a decreased demand from the amalgamating mills. However, the average price for the metal in San Francisco was $1·49 per flask higher than in 1897, and the year was consequently considerably more satisfactory to the quicksilver miners. On March 8, 1899, the stock of quicksilver in San Francisco was only 78 flasks, a smaller amount than ever before in the history of the industry. The production of the important mines in 1898 is shown in the following table:

Month.	New Almaden.	Napa Con.	Mirable.	Ætna.	Great West'n.	Great Eastern.	New Idria.	Altoona.	Abbott.	Redington.	Knox.	Totals.
January	330	600	40	300	70	181	500	306	21	90	37	2,375
February	336	600	68	300	30	135	500	352	20	128	32	2,501
March	450	600	......	200	86	120	400	264	18	70	57	2,265
April	544	600	......	230	81	133	400	422	......	135	3	2,548
May	550	400	......	310	100	170	400	391	24	95	58	2,498
June	600	395	......	360	40	121	400	454	10	132	75	2,587
July	575	655	......	300	202	148	400	404	43	......	52	2,779
August	450	600	......	155	146	140	400	400	18	51	77	2,437
September	500	600	......	250	110	110	400	322	11	......	146	2,449
October	530	600	......	325	83	140	400	247	24	......	45	2,394
November	550	600	......	350	82	136	400	226	......	46	65	2,455
December	460	600	......	370	98	170	400	247	......	243	143	2,728
Total, 1898	5,875	6,850	108	3,450	1,128	1,704	5,000	4,032	189	990	790	30,116
Total, 1897	4,700	6,200	391	3,000	2,445	1,501	3,605	686	773	1,488	*Nil.*	25,479
Total, 1896	6,200	5,000	1,125	3,800	2,165	1,126	1,335	4,203	1,189	1,443	*Nil.*	(*a*)29,271

(*a*) Includes 1,685 flasks produced by the Sulphur Bank mine, which made no output in 1897 or 1898.

In addition to the above production various small mines made an output of 224 flasks, bringing the total for California up to 30,340 flasks, against 26,079 flasks in 1897, which was also the total production of the United States in that year. In 1898 there was a production of 153 flasks in Oregon and Texas, making the total for the United States 30,493 flasks.

California.—The Ætna, Napa, and New Idria companies, all of which are controlled in Boston, were dividend payers in 1898. The Empire Quicksilver Mining Co., near Williams, began the construction of a new 40-ton furnace at the old Abbott mine, which it is working under lease. A new furnace has also been erected at the Buckeye mine S.W. of Sulphur Creek, in Colusa County. The receipts of quicksilver in San Francisco in 1898 were 23,156 flasks, an increase of 5,148 over 1897. Shipments by water were as follows: To British Columbia, 107 flasks; Mexico, 4,427; Central America, 1,486; South America, 10; New Zealand, 30; New York, 135—total, 6,196, an increase of 757 over 1897.

The Napa Consolidated Quicksilver Mining Co. smelted 32,489 tons of ore in 1898, an increase of 3,839 over the previous year. The production of quicksilver was 6,850 flasks against 6,200 in 1897. The average yield of the ore smelted in 1898 was 0·8%. The average receipts per flask were $35·89 and the average cost $21·33. The total receipts were $245,865; expenses, including repairs and new construction, $146,099. Development work on the lower levels showed good bodies of ore, and improvements in the hoisting work will decrease the cost of production in 1899.

The New Idria Quicksilver Mining Co. smelted 18,627 tons of ore in 1896 an increase of 3,317 over 1897 and produced 5,000 flasks of mercury against 3,605 in the previous year. The average yield per ton of ore was 1·03%. The average receipts per flask of quicksilver were $35·86 and average expense $17·48. Total receipts were $184,299; expenses, including repairs and construction, $87,424. A new reduction works has led to a decrease in the cost of production, the automatic handling of the ore contributing largely to this result. Large reserves of ore were opened in the lower levels of the mine during the year, and adits are now being driven at the 500-ft. and 700-ft. levels. Development work is to be undertaken on the San Carlos claims in 1899.

The Ætna Consolidated Quicksilver Mining Co. smelted 18,394 tons of ore in 1898, an increase of 856 tons over 1897. The yield of quicksilver was 3,450 flasks against 3,600 in the previous year. The average yield of the ore treated was 0·72%, a decrease from 1897. The average receipts per flask were $36·03 and the average cost $26·55. The total receipts for the year were $125,228; expenses, $91,609, new construction etc. being included; the expenses were higher than usual owing to the large amount of new work done, especially at the Washington shaft, all of which was charged directly to operation. Extensive explorations were made on the tenth level of the old workings and the Washington shaft opened a body of ore of good grade. Most of the ore produced in 1898 was got from the eighth and ninth levels.

Oregon.—The Black Butte Quicksilver Mining Co. opened a mine in a spur of the Cascade range, about 17 miles from Cottage Grove, in Lane County. A 40-ton reduction furnace of modern design was erected and put in operation in October, but after a short run it was discovered that the metal was being condensed imperfectly, and consequently the plant was closed down for alterations. The Black Butte mine is said to have a large quantity of low-grade ore which may average 0·5% mercury. The quicksilver mines on Evans Creek, which were first opened 20 years ago, were sold to persons in Port Townsend, who proposed to develop them.

Texas.—Lindheim & Co. produced a small amount of quicksilver in a crude furnace near Agua Frio, Brewster County. These mines are said to be prom ising, but at present they are in the hands of persons who have no money to develop them. The ore, it is thought, will average 5% mercury. There has been some of higher grade, but this has been already won. The mines are situated 22 miles from Agua Frio and 90 miles from Rio, on the Southern Pacific Railway. The smelting furnace is 30 miles from Agua Frio and 8 miles from the mine.

QUICKSILVER STATISTICS OF THE UNITED STATES. (*a*)

Year.	Production.			Exports.			Year.	Production.			Exports.		
	Flasks.	Met. Tons.	Value.	Flasks.	Met. Tons.	Value.		Flasks.	Met. Tons.	Value.	Flasks.	Met. Tons.	Value.
1893	30,164	1,046	$1,108,527	16,631	575	$539,385	1896	29,863	1,036	$1,104,997	19,944	692	$618,437
1894	30,440	1,056	1,095,840	14,408	500	397,528	1897	26,079	965	910,418	13,173	475	394,549
1895	33,978	1,179	1,313,589	15,542	539	482,085	1898	30,493	1,058	1,109,945	12,830	445	440,587

(*a*) In 1893 there were imports of 395 flasks, and in 1894 only 7 lb. In 1895, 1896 and 1897 there were no importations.

RANGE OF QUICKSILVER PRICES PER FLASK AT SAN FRANCISCO IN 1898.

Month.	Domestic.		Export.		Month.	Domestic.		Export.	
	Highest.	Lowest.	Highest.	Lowest.		Highest.	Lowest.	Highest	Lowest.
January	$39·00	$39·00	$36·00	$35·00	July	$43·00	$42·50	$38·00	$37·50
February	39·00	39·00	36·00	35·00	August	43·00	42·50	38·00	37·50
March	40·00	40·00	37·00	36·00	September	43·00	42·50	38·00	38·00
April	40·00	40·00	37·00	36·00	October	41·50	40·00	37·50	36·50
May	42·50	40·00	37·50	37·50	November	42·00	41·00	37·50	37·00
June	42 50	42·50	38·00	38·00	December	42·00	41·00	38·00	37·00

GENERAL QUICKSILVER STATISTICS. (*a*)

	1891.	1892.	1893.	1894.	1895.	1896.	1897.	1898.
	Flasks. (*b*)	Flasks. *b*)	Flasks. (*b*)	Flasks. (*b*)	Flasks. (*b*)	Flasks. (*b*)	Flasks. (*b*)	Flasks. (*b*)
Shipments from Spain to London	47,993	47,321	44,670	42,414	40,409	40.949	46,577	46,367
Shipments from Italy to London.	10,440	6,765	6,680	8,700	5,775	3,800	4,450	5,650
Total	58,433	54,086	51,350	51,114	46,184	44,799	51,027	52,017
Shipments from London	63,143	46,055	42.265	41,046	37,937	31,273	31,734	32,288
Maximum price of Spanish.	£9 0s.	£7 15s.	£6 17s. 6d	£6 15s,	£7 7s. 6d.	£7 5s.	£7 7s. 6d	£7 15s.
Minimum price of Spanish	7 5s.	6 1s.	6 2s. 6d	5 10s.	6 7s. 6d.	6 8s. 6d	6 12s. 6d	7 0s.

(*a*) From W. Sargant & Co.'s Annual Metal Circular. (*b*) American, Russian and Italian flask, 76·5 lb. (34·7 kg.); Mexican flask, 75 lb. (34·03 kg.); Spanish flask, 76 lb. (34·5 kg.).

QUICKSILVER PRODUCTION OF THE WORLD. (IN METRIC TONS.) (*a*)

Year.	Austria.	Canada.	Italy.	Mexico.	Russia.	Spain.	United States.	Total.
1893	512	(*b*)	273	286	201	1,666	1,046	3,984
1894	519	(*b*)	258	300	196	1,610	1.056	3.939
1895	535	2·5	199	213	434	1,506	1,179	4,069
1896	564	2·0	186	218	491	1,524	1,036	4,021
1897	514	0·3	192	294	617	1,728	965	4,310
1898	494	(*b*)	170	353	362	1,681	1,058	4,118

(*a*) From official reports of the respective governments and direct reports of the producers to THE MINERAL INDUSTRY. The figures for Austria and Italy for 1898 are due to V. Spirek, that for Russia to the sole producer and that for Spain to the *Revista Minera*. (*b*) None.

Italy.—(By V. Spirek): "The production of quicksilver in Italy in 1898 was 170 metric tons. The production of quicksilver by the Siele and Monte Amiata mines is shown in the subjoined table, which gives the official figures of the Italian government, presented recently at the Turin Exposition. These show the concentration of the ore by washing, the amount of quicksilver contained in the ore, and the production; consequently the faults of the metallurgical processes during the eight years last elapsed may be compared. The loss of metal is about 8% It is expected that this will be reduced by the use of the Spirek furnace.

Year.	Siele.					Total of Monte Amiata.		
	Ore.	Concentrates.	Hg in Ore.	Hg.	Hg Produced.	Ore.	Hg.	Hg.
	Tons.	Tons.	Tons.	%	Tons.	Tons.	%	Tons.
1890	1,989	1,668	334	2·0	287			449
1891	2,955	2,643	261	9·8	244			330
1892	3,654	3,654	296	8·1	258			325
1893	4,800	4,590	259	5·6	235	19,450	1·9	273
1894	5,372	4,773	229	4·8	216	15,022	1·7	258
1895	7,758		170	2·14	158	10,504	1·9	199
1896	9,265		158	1·6	149	14,305	1·4	186
1897	12,000		150	1·2	138	20,659	1·0	192
1898								170

"An increasing proportion of low-grade ores has been worked in recent years. The production will be increased in 1899 by the output of the works of San Salvadore, which are to begin operations in March. There are three works in Monte Amiata for the treatment of quicksilver ores, namely, Siele, with one large, one medium, and one small-size Cermak-Spirek furnace, three shaft furnaces, and one muffle furnace; Cornacchino, with one large, one small Cermak-Spirek furnace, and one shaft furnace; and San Salvadore, with two large and two small Cermak-Spirek furnaces. The output in 1899 will probably be 45,000 tons of ore, which will yield 450 tons of quicksilver."

Mexico.—The production of quicksilver in Mexico continues to be made by a few mines in the State of San Luis Potosi and the Huitzuco mines in the State of Guerrero. There are numerous small mines in these districts exploited by natives, from which it is impossible to obtain statistics, although their production has probably been increasing, owing to the higher price of quicksilver in London and the existing high rate of exchange in Mexico, which increases proportionately the value of the product in Mexican currency. El Nuevo Potosi mine in San Luis Potosi was practically idle in 1898 on account of lack of working capital.

The quicksilver deposits of Huitzuco were discovered about 28 years ago, and their exploitation was commenced at the beginning of 1874. They are situated at 18° 21′ north latitude and 0° 5′ west of the meridian of the City of Mexico.[*] They are owned and operated by the estate of Señor Romero Rubio and for many years have been the largest producers of quicksilver in Mexico,

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[*] "El Mineral de Huitzuco, Estado de Guerrero," by Carlos Sellerier. *El Minero Mexicano*, No. 26, Dec. 29, 1898. For notes as to the history, mineralogy and geology of these deposits reference should be made to that paper.

although they have not been worked to their maximum capacity. At least Señor Sellerier, who has lately described these deposits, is of the opinion that by the introduction of modern methods of mining and reduction the output of Huitzuco could be largely increased, the reserves of low-grade ore being large and the cost of mining and reduction so cheap that ore with as little as 0·3% quicksilver can be worked at a profit.

The production of the Huitzuco mines since 1885 is shown in the following table:

Year.	Ore. "Cargas." (a)	(Mercury.) Kilograms.	Flasks of 75 Lb.	Average Grade of Ore.	Average Price per Quintal of Mercury. (b)
				%	
1886..........	26,016½	61,443	1,780	1·01	$62·00
1887..........	81,626¼	76,942	2,229		62·00
1888..........	72,642¼	86,538	2,507		70·00
1889..........	74,804	86,158	2,496		80·00
1890..........	97,154¾	60,856	1,763	0·91	78·00 65·00
1891..........	86,021¼	107,801	3,123		
1892..........	157,345½	165,896	4,806		
1893..........	85,955¾	84,087	2,436	0·62	65·00 to 91·00
1894..........	111,237	70.797	2,051		
1895..........	134,963½	96,652	2,800		
1896..........	146,555¾	95,938	2,779		

(a) 300 lb. Spanish. (b) Mexican currency.

New Zealand.—A. McKay, Government Geologist, investigated a cinnabar lode in the Kauaeranga Valley, about 6 miles from where the river enters the Firth of Thames at Shortland, and considered the outlook to be sufficiently promising to warrant the development of the property. There is a small amount of rich ore and a considerable quantity of ore of medium grade. The New Zealand government offers a bonus of 4d. per lb. for the production of 100,000 lb. of quicksilver from any New Zealand mine.

Russia.—The production of quicksilver by Auerbach & Co., who continue to operate the only quicksilver mines in the empire, in 1898 was 22,122 poods (362,358 kg.), against 37,600 poods (617,000 kg.) in 1897. Owing to the decreased output no dividends were paid in 1898, but the prospects for 1899 are thought to be better. The production and average yield of the ore are shown in the following table:

	1888.	1889.	1890.	1891.	1892.	1893.	1894.	1895.	1896.	1897.	1898.
Average yield, %...	0·76	0·51	0·73	0·80	0·65	0·46	0·46	0·69	0·71	(a) 0·79	(b)
Production, kg....	164,815	167,109	292,137	323,865	342,768	200,999	195,857	434,070	491,465	617,000	362,358

(a) To Oct. 1, 1897. (b) Not yet reported.

Spain.—According to Señor Adriano Contreras, in the annual statistics published in *Revista Minera*, the production of quicksilver in Spain in 1898 was 48,722 flasks, of which 1,450 flasks came from El Porvenir mine, and 46,211 from Almaden, the remainder being produced by various small mines. The monthly production of the mines of Almaden was as follows: January, 8,167; February, 7,155; March, 7,350; April, 3,492; May, 220; October, 3,575; November, 7,161; December, 9,091—total, 46,211 flasks.

PRESENT PRACTICE IN THE METALLURGY OF QUICKSILVER IN CALIFORNIA.

BY R. B. SYMINGTON.

ALL the quicksilver mines in the United States are found in the Coast Range in California at intervals in a belt extending about 250 miles both north and south of San Francisco; in fact, cinnabar has been discovered in excavating for the foundations of buildings in that city. Practically all the ore is in the form of cinnabar, other varieties such as metacinnabarite and native quicksilver being of rare occurrence and of no commercial importance.

Geology.—Generally the veins are mineralizations along faults occurring near the contact of metamorphic and unaltered rocks. A variety of serpentine, locally called "quicksilver rock," is the usual metamorphic rock, while sandstone and shale constitute the unaltered strata.

Occasionally the ferruginous shales are interlaminated with bands of jasper which often has a bold outcrop, and looking like cinnabar is sometimes mistaken for the latter, but the real veins seldom show a bold outcrop. When a porous rock like sandstone forms the wall of a vein, it is often impregnated with cinnabar for a considerable distance from the fissure, and the true vein matter may be comparatively barren; but if clay or any impervious rock forms the wall, no ore will be found beyond that line. In some of the smaller mines beds of impregnated sandstone are worked as the vein, but further development will doubtless connect these with the fissures from which the ore originated.

Mining.—The departure of the cinnabar from the fault fissure renders it necessary to thoroughly examine the adjacent ground by intelligent cross-cutting, and the neglect of this has been the cause of many failures, while the success which has followed the reopening of some of the previously abandoned mines has been due to attending to this matter. Formerly, too, it was the custom to hunt for only the higher grade ore, overlooking the poorer parts of the deposit. Now, however, everything containing over 0·5% of quicksilver is worked, resulting in a much reduced cost per ton for mining. More than once the waste-dumps of quicksilver mines have been turned over with excellent results.

Cinnabar has been found in Oregon in the same Coast Range as in California, but of too low grade to be worked profitably. Concentration by mechanical washing has been suggested as a means of making these large low-grade deposits available, but there are two reasons why this is not feasible: (1) Because the cost would generally be nearly as much as that of treatment in the furnace; and (2) because, notwithstanding the high specific gravity of cinnabar, it is difficult to save it when the ore is crushed mechanically, since it is so soft that it is floured by the treatment and floats off. Fairly successful concentration can be effected with naturally decomposed rock, but the first reason remains sufficient to prevent the method from being profitably adopted.

The New Almaden mine in Santa Clara County is worked to a depth of over 2,000 ft. Two or three others have reached a depth of 1,000 ft., but in all the remaining mines the workings are comparatively shallow. One per cent. quicksilver is not far from an average of the grade of the ore. It is impossible to state definitely the average cost per ton of mining, owing to the extremely variable nature of the prospecting and dead work always accompanying the operation, but it may be said generally that the sandstones are usually soft and easily worked, while serpentine is tougher and more expensive to mine. The ore chambers are apt to be large and irregular in shape, making any average estimate of the cost of working them difficult. Three dollars per ton might be given as an approximation of the mining costs when no special difficulties are encountered.

Metallurgy.—The reduction of the ore is a simple process, yet requiring careful attention in the details. The earliest practice was to retort the ore; then reverberatory furnaces, worked intermittently, were used; then continuous shaft furnaces suitable only for coarse ore, the fine ore having to be made into briquettes and the whole being mixed with the necessary amount of fuel; and then the Smith fine-ore furnace, which did away with the necessity of making briquettes, using the shaft furnace only for the coarse ore. The best modern practice is to use fine-ore furnaces exclusively, putting the coarse ore through a rock-breaker to reduce it to the proper size, while the experience of the last few years has shown that the fine ore need not be as small as was formerly thought necessary, and that furnaces of a larger capacity are advantageous.

The reduction process can be best illustrated by a description of the 50-ton (daily capacity) fine-ore furnace last erected at Oat Hill by B. M. Newcomb, superintendent of the Napa, Ætna and New Idria quicksilver mines, which embodies the experience of his many years' successful management of these properties. He kindly permitted me to make and use for this paper the accompanying drawings and details of operation. The drawings are in skeleton and do not attempt to show details which would only be confusing in plans drawn to so small a scale. Fig. 1 is a general ground plan of the reduction works; Fig. 2 is a vertical section through condenser and furnace on line *AB* of Fig. 1; Fig. 3 is a vertical section of furnace on line *CD* of Fig. 1 and elevation showing condensers; Fig. 4 is a horizontal section of furnace on line *EF* of Fig. 3. The reference lettering and scale are the same in all figures.

The ore is passed over a 1·5-in. grating at the mine. What does not go through the grating is run through a rock-breaker set to break to 1·5-in. size, and all is dumped into the ore bin from the cars on the tramway passing over it. There are eight sliding gates, worked by levers, opening from the ore bin (25 ft. long) to the drying platform (25 ft. square), sloping at an angle of 30° with four lines of foot-boards (shown on Fig. 2, but not on Fig. 1 or Fig. 3) for the attendant to stand upon and spread the ore evenly over the platform. Beneath the platform is a coil of 1·5-in. steam-pipe laid so as to be about 6 in. apart, through which steam is passed to dry the ore when necessary, but since two-thirds of the platform is directly above and close to the first and hottest condenser, this is not always needed, and the moist ore also helps to cool the condenser. A series of gates similar to those from the ore bin open to spouts below which a tramway passes, and the ore is run into cars as required for charging the furnace, which is reached by a turntable and tramway at right angles to the other track. On the top of the furnace is a hopper with two slots extending the full length of the roasting chamber, which are opened and closed by sliding iron plates operated by levers (not shown on the diagrams). The hopper is kept full of ore, which serves both to heat the ore without volatilizing it, and to act as a seal when the charge is being introduced into the furnace. The ore is fed at intervals of one hour, fully two tons to each charge.

The furnace is a brick structure 22 ft. long, 16 ft. wide and 37 ft. high, braced every five feet of its height by a timber clamp-frame 16 in. wide by 7 in. deep, having heavy iron angle-socket pieces at the four corners. Each of these socket pieces is connected to the next one horizontally by a 1¼-in. curved iron tension rod, having turnbuckle screws, and bearing on struts, which in their turn bear against the timber at three intermediate points on each side of the building. These braces are not shown on the diagrams and are merely precautionary against any bulging that might take place.

The fire-grate and fire-box are shown at *a*, Figs. 2 and 4, extending across the furnace and fed from both ends. The fuel required for roasting the ore is 2½ cords of wood every 24 hours. The roasting chambers shown by *i*, *i*, *i*, *i*, Fig. 4, by the zigzag shelves in Fig. 3, and by the horizontal divisions in Fig. 2, are the most important parts of the structure. They consist of four compartments alike in every respect, divided by the brick partitions, in which a zigzag series of sloping shelves is formed by fire-clay tiles 3 ft. long, 16 in. wide and 3 in. thick, resting at the upper end on suitably shaped projecting impost courses, and set into the partition about 2 in., the lower end being partly supported by a firebrick (not shown on diagram) set on the tile of the next lower shelf. Three of these tiles set end to end form a shelf 9 ft. long.

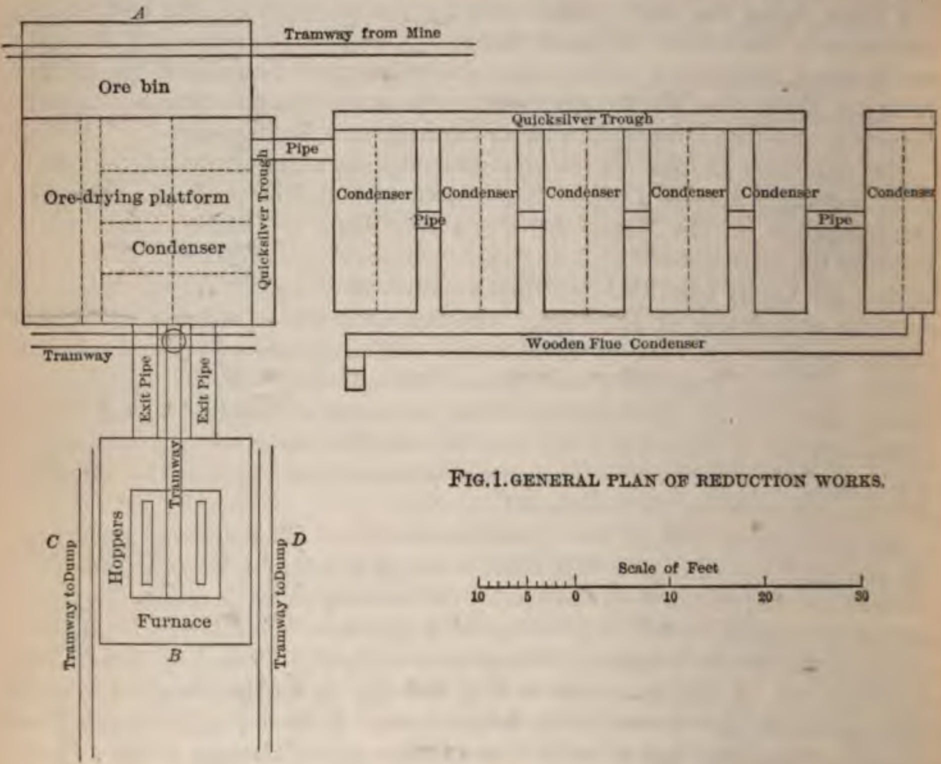

FIG. 1. GENERAL PLAN OF REDUCTION WORKS.

The capacity of the furnace is regulated by the width of the space left between the lower end of one shelf and the upper surface of the next shelf below it. In this case 6 in. is the gauge. There are 24 shelves in each of the four compartments. While the furnace is charged every hour, the roasted ore is withdrawn every 15 minutes from the discharging ports *f*, *f*, *f*, Figs. 2 and 3, by hand rakes into cars leading to the waste-dump. As the point of discharge is about 5 ft. below the level of the fire-grate the spent ore has cooled considerably before it is withdrawn and the floor slopes toward the car, rendering the task quite easy. As the spent ore is removed its place is taken by the material on the shelf above it, and so on up to the top, the whole operation being continuous and 50 tons (2,000 lb.) being treated every 24 hours. The furnace when full holds about 40 tons, so that the ore remains under treatment about 20 hours. This is in very marked contrast with the two and three hours stated as the practice in Italy by Vincente Spirek in his article in THE MINERAL INDUSTRY, Vol. VI., p. 577.

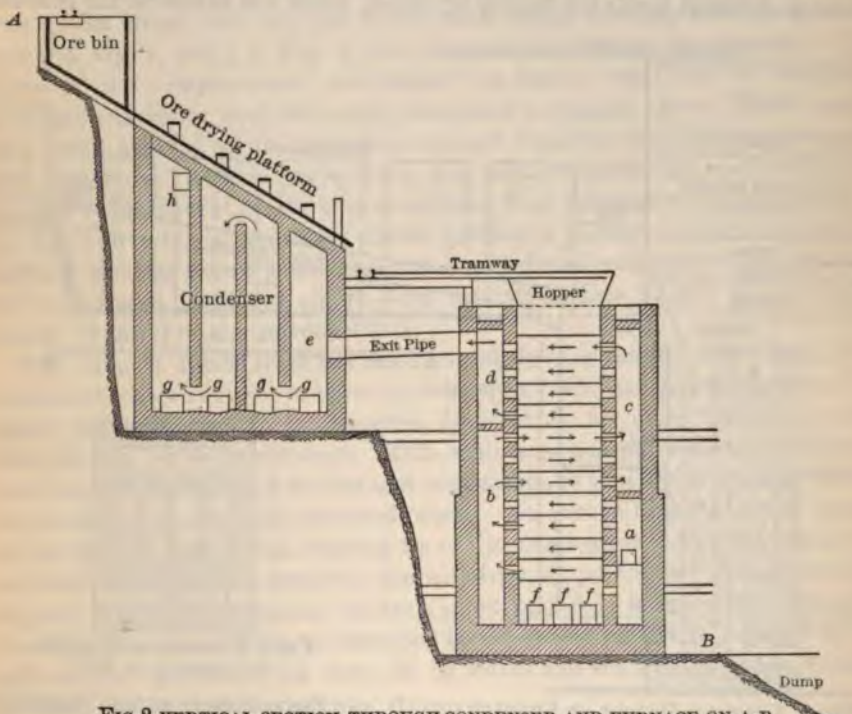

FIG. 2. VERTICAL SECTION THROUGH CONDENSER AND FURNACE ON A-B.

At the end of the furnace opposite the fire-grate is the dust chamber *b*, Figs. 2 and 4. Above the fire-box, and separated from it by fire-clay tiling or arching, is the chamber *c*, while above *b* and separated from it in the same way is the chamber *d*. The flame and hot air from the fire-box *a* pass through pigeon-holes in the cross partition wall at the end of the roasting chamber (Figs. 2 and 4), one hole 8 in. high by 6 in. wide being placed so as to be just under the upper end of each sloping shelf. The charge of ore on the shelves fills and seals the 6-in. space between each zigzag pair of tiles, so that the upper surface of the ore, the partition wall and the under surface of the upper tile form a cell or flue through which the flame and hot air pass, roasting the ore, and passing by similar pigeon-holes into *b*, as indicated by the arrows in Fig. 2. Continuing in the same manner the course of the current is reversed, passing from the upper portion of *b*, over the ore to the lower portion of *c*, then over the upper tiers of shelves to *d*, when the volatilization is complete, a current of flame and hot air constantly coursing over each shelf, and every particle of the ore passing over every shelf of a compartment on its way down.

The temperature aimed at is a cherry red heat. Quicksilver volatilizes at 680° F., and it is obviously most desirable, both for economy of fuel and to facilitate condensation, not to use a higher heat than necessary. Peep-holes in line with every pigeon-hole are made through the outside end walls of the furnace so that the temperature can be observed and any obstruction removed by inserting an iron rod. A light, open shed-roof (not shown on the diagrams), with flooring stages at different levels for facility of access, cover and surround the furnace.

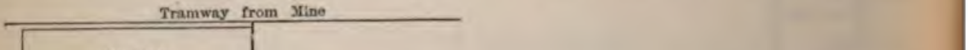

FIG. 4. HORIZONTAL SECTION OF FURNACE ON E-F.

FIG. 3. VERTICAL SECTION OF FURNACE ON C-D. AND ELEVATION SHOWING CONDENSERS.

The ore in passing through the furnace loses about 10% in weight, even in the case of 1% ore. In some localities there is enough pyrites in the ore to diminish the amount of fuel required for roasting, so that a cord of wood suffices for 40 tons of ore, and occasionally there may be ore with sufficient pyrites to enable it to be roasted without further fuel.

The volatilized quicksilver and other products of combustion pass from the furnace into the first condenser through two cast-iron pipes or boxes each 30 in. square and 12 ft. long, sloping slightly downward from the furnace and fitted with man-holes. At Oat Hill the condensers are brick structures with solid masonry or brick floors, divided by partition walls into cells, the partitions being alternately open at top and bottom, causing a continuous current as indicated by the arrows in Fig. 2, where the fumes enter the first condenser at *e*, and leave at *h*, this condenser being divided into eight cells. The partitions are shown by dotted lines in Figs. 1 and 3. There are six other condensers, each connected by one pipe similar to the exit from the furnace. These have each one partition, making two compartments, except the fifth of this series which, owing to the topography of the site, has only one compartment. From the last brick condenser a wooden flue 24 in. deep by 30 in. wide (Figs. 1 and 3) slopes gently downward for 60 ft., then vertically downward for 30 ft., then, after two square bends, vertically upward about 40 ft. to the exhaust fan, from which a similar flue about 20 ft. high discharges the cooled fumes into the open air.

The floors of the condensing cells are covered with cement, are slightly concave, and slope gently to the discharging points *g*, *g*, *g*, *g*, Fig. 2, which are light iron plates luted with clay, the precipitated metal running into the quicksilver troughs, Fig. 1, and *j*, *j*, Fig. 3, each compartment having one opening. Two or three of the compartments are cleaned up nearly every day in rotation, the arrangement being such that each condenser is cleaned about once a week. A very small amount of quicksilver is obtained from the first compartment of the first condenser, a little more from the next compartment, increasing up to the last compartment of the first condenser, then gradually diminishing until in the last two or three condensers almost nothing is yielded. At the bottom of the vertical wooden flues a small quantity is sometimes collected. The capacity of the condensers is 12,400 cu. ft., the cooling surface 12,200 sq. ft., and the length of travel of the current 400 ft.

The exhaust steam from the small engine driving the fan is led into the first condenser where it is found to be advantageous. The exhaust fan is a small four-blade paddlewheel 5 ft. in diameter, from which far more than the necessary draught can be easily obtained. This maintains a steady current from the fire-box through the roasting shelves and condensers, so that the attendants are never salivated in the ordinary course of work. No change is made in the operating of the furnace during the cleaning up of the condensers. The fan prevents any outward escape of the fumes at the withdrawing points and the admission of a slightly increased volume of air for a short period is immaterial. This differs from the practice in Italy, as described in the article previously referred to, where half of the condensers are shut off by valves and the charge in the furnace is lessened during the cleaning up.

Together with the quicksilver a considerable quantity of soot is raked out at every clean-up. This soot is impregnated with quicksilver, which is extracted by stirring and rubbing in a small 3-ft. iron pan with a concave bottom. The pan is charged with ten or twelve 3-gal. buckets of soot, and after about an hour's stirring and rubbing the quicksilver collects and is withdrawn from the bottom. An ordinary yield of each run of the pan is about one flask or 76½ lb.

About 30% of the total yield is got from the soot, this, too, differing widely from the 70 to 80% reported to be obtained in Italy from a similar grade of ore. The soot-pan and rock-breaker of the works herein described are situated at the old furnace, which is similar to the new one, but the shelving tiles are gauged to a 4-in. space. The volatilization effected by the new furnace is very complete, hardly a trace of cinnabar being found in the material discharged. The condensers are also efficient, but an amount estimated at about 5% escapes. The summer temperature is often over 90° F., but no material difference in the operation of the furnace is noted during the summer season.

The number of men required is two attendants above, for spreading and drying the ore, feeding the furnace and firing the boiler which makes steam for the fan engine, etc.; and two men below for firing the furnace and withdrawing the spent ore, on both day and night shifts; besides one man for the soot-pan for the day shift only. The fuel used for raising steam to dry the ore, for running the rock-breaker, exhaust-fan, and soot-pan is 2¾ cords per day, the rock-breaker and soot-pan sufficing for both furnaces. The cost of such a furnace as this, treating 50 tons every 24 hours, is about $35,000.

There are no known extensive unworked sources of supply of cinnabar in the United States at the present time, but systematic exploration will doubtless keep up the present rate of production for a long time.

THE ASSAY OF QUICKSILVER ORES.

R. E. CHISM described an improvement on the crucible and gold cover method of assaying quicksilver ores by using silver foil instead of gold, the advantage being in the greater economy.* A Battersea annealing cup, size C, is used as the retort, heat being applied to the bottom by means of a Bunsen burner or alcohol lamp. The upper part of the crucible is protected from the heat by a circular shield of tin plate 13 cm. in diameter, with a 3·5 cm. hole in the center. The annealing cup passes almost through the hole, but remains firmly fixed therein with about 1 cm. of its upper part protruding. The shield serves also to suspend the crucible from the ring of a lamp stand. The crucible is covered by a piece of silver foil 0·02 mm. thick and 5 cm. square, weighing about 0·6 g., above which is placed a shallow silver dish, which is filled with water in order to cool the silver foil.

In operation 0·5 to 1 g. of ore is mixed with 5 g. of iron filings, which should be as fine as possible, filings which will pass a 60-mesh sieve being desirable. The charge is heated in the crucible, covered as described above, for 10 to 15 minutes. Ten minutes is too short for most ores, and more than 15 minutes is apt to lead to loss of mercury. The water in the cooling dish should be renewed once or twice during the assay. When the heating is finished the crucible is allowed to cool at least five minutes, after which the foil is weighed, the increase in weight showing the amount of mercury absorbed. The amalgamated silver foil is then replaced on the crucible, which is heated again for about 10 minutes, after which the foil is weighed again. If the weight is constant, or if there is a slight decrease, the amount of mercury obtained by the first weighing may be considered correct. If, however, there is an increase in weight the determination is repeated with a new charge, which is heated for a longer time, say 5 to 10 minutes more than at the first determination. The silver foil is prepared for use again by heating over an open flame, when the mercurial stain quickly disappears.

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* American Institute of Mining Engineers, Buffalo Meeting, October, 1898.

RARE ELEMENTS.

Last year was remarkable for the number of new elements of which the discovery was reported, the list comprising krypton, neon, metargon, xenon, coronium, monium, polonium and etherion, the existence of all, with the exception of the last, being regarded by chemists as probable, if not well established. The existence of an alleged new element called bythium, which was claimed to have been obtained by the decomposition of sulphur, was disproved by the investigations of independent chemists. Two new tabulations of the atomic weights were presented in 1898, one by Prof. T. W. Richards, and the other by the commission appointed by the German Chemical Society. Both of these were based upon the principle of accepting most probable figures, while the table of F. W. Clarke gives precedence to precision in the analytical methods, which is subject to the criticism that even the most expert chemical manipulation is likely to continue certain constant errors. However, the difference among the values assigned by the three tables is not great, the most important variation being in the cases of antimony, cadmium, platinum, tungsten and some of the rare elements.

Calcium.—M. Henri Moissan has succeeded in obtaining crystalline calcium free from impurities by reducing calcium iodide by means of an excess of sodium in an iron crucible at a red glow for one hour, making use of the fact that metallic calcium is soluble in liquid sodium at this temperature. On cooling, the calcium crystallizes out, and on removing the sodium by means of absolute alcohol, silver white, brilliant hexagonal crystals of pure calcium are obtained. The operation is performed by charging into an iron crucible of 1,000 c.c. capacity 600 g. of crystallized anhydrous calcium iodide and 240 g. of sodium, *i.e.*, three times the sodium required by the equation $CaI_2+2Na=Ca+2NaI$. The crucible is closed with a screw top and placed in a furnace and maintained at a dull red heat for about an hour, being occasionally shaken. The crucible after cooling contains a blue mass of the melted salt upon which rests a cake of metallic sodium. This cake is broken into fragments and thrown into a liter flask containing 500 c.c. of absolute alcohol cooled by ice. When the evolution of hydrogen has ceased, the liquid is decanted and an equal quantity of absolute alcohol is added. This washing is continued until the alcohol shows no residue on evaporation. The brilliant powder left behind is treated with absolute ether and then dried in a stream of dry carbon dioxide or hydrogen. The yield is about 50% of the theoretical quantity, about 40 g. being obtained in each operation. Crystallized metallic calcium may also be obtained by electrolyzing fused calcium iodide at a red heat, the negative electrode being pure nickel and the positive a graphite cylinder placed in a porous cup. On analysis the metal thus obtained gave 98·9, 99·1 and 99·3% of calcium in three samples.*

Pure calcium prepared by the first method has a melting point of about 760° C. On solidifying the molten metal a non-crystalline and soft form of calcium results which can be cut with a knife but is less malleable than sodium or potassium. The density of the crystalline form is 1·85, while non-crystalline calcium sinks in propyl iodide of density 1·78, and floats in allyl iodide of density 1·87. The non-crystalline form scratches lead, but not chalk. It can be forged, molded, and drawn to wire. Calcium crystallizes in hexagonal plates or derived rhombohedra. With hydrogen the metal forms a crystalline hydride, CaH_2. Calcium is not attacked by chlorine in the cold, but at 400° C. the chloride is formed. It burns in oxygen at 300° with a brilliant light, and evolution of sufficient heat to volatilize part of the lime which is formed. At a dull red heat, in a current of air, both the elements of the atmosphere are fixed. Water attacks calcium at ordinary temperatures, slaked lime being formed.† The decomposition is more rapid with water containing sugar in solution. It is attacked very slowly by fuming nitric acid, but rapidly upon the addition of a little water. Sulphuric acid, fuming and the monohydrate, chlorhydric and acetic acids act upon it in the cold. Crystallized calcium combines with the carbon of smoke at below red heat, forming a carbide, CaC_2, with a great disengagement of heat. At below red heat it takes fire in NO_2 with a dazzling light and the formation of fused lime. With liquid ammonia it forms a solid alloy at 40° C., represented by the formula, $Ca(NH_3)_4$, which takes fire in the air at ordinary temperatures.

On gently heating metallic calcium in an atmosphere of nitrogen, a combination between the two takes place. At a dark-red heat the reaction becomes violent, raising the whole mass to incandescence. M. Moissan finds that the yellow color usually associated with metallic calcium is due to a surface coating of this nitride, the pure metal being quite white. The composition of the nitride appears to be N_2Ca_3; its sp. gr. is 2·63 at 62° F., and its melting point in the neighborhood of 2500° F. Its most remarkable property appears to consist in a reaction with water, resulting in the production of ammonia, quite analogous to the production of acetylene from calcium carbide. This reaction may ultimately prove of industrial importance, provided the difficulties of cheaply preparing metallic calcium by means of the electric furnace are successfully overcome. Calcium heated to redness in an atmosphere of hydrogen unites with the latter, forming calcium hydride, CaH_2, a white fused mass, of crystalline fracture, 1·7 sp. gr., and stable at red heat, though it takes fire in the blowpipe flame. It decomposes water, forming calcium hydrate and free hydrogen.

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* *Comptes Rendus*, CXXVI., 1753-7, June, 1898.

† *Idem*, CXXVII., No. 17, Oct. 24, 1898.

Columbium.—According to Dr. F. R. Carpenter, columbite is abundant at the Bob Ingersoll mine in the Black Hills. W. P. Blake is authority for the statement that a mass of it weighing 1 ton was once found there.

Coronium.—Prof. Nasini claims to have discovered in some volcanic gases at Pozzuoli the previously hypothetical element coronium, supposed to cause the bright line 5,316·9 in the spectrum of the sun's corona. The hypothetical elements aurorium, with a characteristic line at 5,570.7, and nebulum, having two bright lines at 5,007·05 and 4,959·02, still await discovery. These investigations are still in the domain of spectroscopy. Analogy points to coronium being lighter and more diffusible than hydrogen.

Crypton is a new gas isolated by Prof. Ramsay from the atmosphere by evaporating large quantities of liquid air. It is transparent, heavier than argon, and, like that element, inactive. Crypton is present in the atmosphere in the proportion of 1: 20,000. The chief lines of the spectrum are green and yellow, the yellow being nearly coincident with the helium yellow line D 3. According to M. Berthelot, its wave length is 5,566·3, and its density approximately 22·5, that of oxygen being 16.

Etherion, of which the discovery is claimed by Prof. Chas. F. Brush, is a gas which exists in the atmosphere and is remarkable for its enormous heat conductivity at low pressure, this being something like 100 times that of hydrogen. The existence of this gas, however, is disputed by chemists and physicists. Sir William Crookes pointed out the probability of its being simply a residue of water vapor.

Glucinum.—The high temperature required for the reduction of glucinum, its volatility, and the ease with which its carbide is formed, have prevented the preparation of this metal. The reaction generally employed heretofore consists in making potassium or sodium react on glucinum chloride or on the double fluoride of glucinum and potassium. A few attempts have been made to obtain the metal by electrolysis, but it is doubtful if any of the products have been pure. P. Lebeau finds* that pure glucinum may be obtained by the electrolysis of glucinum-sodium fluoride, BeF_2NaF, which may be prepared by mixing concentrated solutions of glucinum fluoride and sodium fluoride and allowing them to crystallize, or by dissolving pure glucinum hydrate and sodium carbonate in the exact calculated proportions in fluorhydric acid. The solution is evaporated and the residue melted in a platinum crucible. The double salt must be kept in a stoppered glass bottle. If the quantity of glucinum fluoride present is less than that which corresponds to the formula BeF_2NaF the product will become deliquescent. In the production of metallic glucinum the salt is melted in a nickel crucible, and when fused (care being taken that the temperature does not exceed a very dull red) it is subjected to a current of 6 to 7 ampères and 35 to 40 volts. A rod of graphitic carbon serves as positive electrode, while the nickel crucible itself acts as the negative pole. In using a crucible capable of containing 100 g. of the salt a non-adherent felted crystalline mass was obtained after about 45 minutes. This was isolated and treated with boiling water. After prolonged washings a

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* *Comptes Rendus*, CXXVI., No. 10.

powder formed entirely of rather irregular crystals, which under the microscope has a brilliant white metallic appearance, was collected. This was pure glucinum, entirely free from nickel and iron.

L. Liebmann has patented[*] a process for the preparation of glucinum from beryl. The mineral is pulverized and thoroughly mixed with a quantity of fluorspar, and after melting is subjected to electrolysis with carbon electrodes. The calcium fluoride dissolves the silica from the beryl and forms lime or lime alumina silicates. The glucinum compounds having been freed from silica are again subjected to the electric current, which sets glucinum free on the cathode.

In attempting the reduction of glucinum oxide, several alloys have been prepared.[†] An intimate mixture of carbon with the oxides of glucinum and copper, best prepared by calcination of the nitrates, heated for five minutes in the electric furnace, gives a rosy red non-homogeneous ingot, from which a true alloy of a yellow color is separated by fusion, containing from 5 to 10% glucinum. The 5% alloy is malleable, readily filed and polished, unaffected by air, but slightly tarnished by sulphuretted hydrogen. It is easily soluble in HNO_3. The glucinum copper alloy with only 1·3% is very sonorous, easily worked and of a golden yellow color. Alloys of glucinum with other metals have been similarly prepared.

Helium is similar to argon in its physical properties and resembles it also in chemical inertness. Its atomic weight has not been determined exactly, but is probably about 4·28. It was discovered originally by the spectroscope in the light of the chromosphere during the solar eclipse in 1868, but it is only recently that it has been identified on this earth, where it occurs in certain rare minerals and in some mineral waters, associated with argon. It is not a normal constituent of the atmosphere, although its existence in the air has been reported by several scientists, among them Sir William Crookes, who is also authority for the statement that a sample of the helium separated by Prof. Dewar from Bath gas undoubtedly contained neon. Prof. Dewar in 1898 succeeded in liquefying helium.

Hydrogen.—On May 10, 1898, Prof. Dewar reported that he had succeeded in liquefying hydrogen. Static hydrogen boils at a temperature of 238° C. at ordinary pressure, and 250° C. in a vacuum, the latter being only 23° C. less than absolute zero. It collects well, drops easily, and with proper arrangements may be manipulated as easily as liquid air. Prof. Dewar produced about half a glassful in five minutes. When pure it is a colorless, transparent liquid, possessing considerable refractive power. It has a well-defined meniscus, but no appearance of being a metal, as Faraday thought it might prove to be. It is not a conductor of electricity. Its density is only one-fourteenth that of water, or six times less than that of the lightest liquid hitherto known. One liter weighs only 70 g. It instantly freezes and solidifies air and oxygen and hence enables the density of those two solids to be ascertained. A little ball cooled in it

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[*] German Patent No. 101,374, Feb. 9, 1898; English Patent No. 3,479, of 1898; *Zeits. f. Angew. Chem*, Jan. 17, 1899.

[†] P. Lebeau, *Comptes Rendus*, 1897, CXXV. (26), pp. 1172–1174.

and exposed to the air was first covered with a coating of solid air and then began to drip liquid air. A piece of cotton wool soaked in it appeared magnetic. But the liquid itself is not magnetic, so that this phenomenon must be due to the cotton wool being immediately filled with solid oxygen, which is highly magnetic. A vacuum of very high tenuity can be easily and simply obtained by immersing a closed tube in the liquid. With suitable arrangements, according to Sir William Crookes, a vacuum with a pressure of only one ten-millionth of an atmosphere can be produced in this manner.

Lithium.—H. Moissan has found that metallic lithium forms with liquid ammonia a solid alloy at 40° C., which on coming in contact with the air at ordinary temperature takes fire.

A small amount of lepidolite, or lithium mica, was produced in 1898 by the California Lithia Mining Co., near San Diego, Cal. According to C. R. Orcutt the deposit of lepidolite at this place was discovered in 1888, and in 1892 about 20 tons of mineral interspersed with crystals of rubellite was mined and sold for specimens. The mine is situated about a mile from the Pala Mission, on the San Luis del Rey River, in the northern part of San Diego County, in a hill of dark hornblende and feldspar diabase. About 300 ft. above the valley, on the east side of the hill, there is a pegmatite vein, underlying which is a bed of light-colored, fine-grained lepidolite, containing crystals of rubellite, which are often found in radiations. The developments already made show a body of mineral about 2,000 ft. in depth, 20 ft. in thickness, and 500 ft. in length, which is estimated to contain 1,500,000 tons. The percentage of lithia ranges from 3 to 5%. A recent analysis showed 49·5% silica, 1·1% manganese oxide, 27·9% alumina, 3·5% lithia, and 17·6% potash. The deposit is believed to have much industrial value as an ore of lithium.

Lepidolite has been found to be useful in making a glaze for agateware and pottery, brick and tiles, giving a beautiful white enamel. The rubellite contained in it also fuses to a white enamel, but with more difficulty. It has been tested as a material for glass making. At a low red heat it melts to a white enamel, but at white heat for over three hours it is said to become clear glass without the addition of any flux. Combined with lime it is said to make a superior cement. The chief value of the mineral, however, will be probably for the manufacture of lithium salts, for which there is a large demand, especially for the manufacture of lithia water. The consumption of lithium carbonate in the United States is variously estimated at from 20 to 100 tons per annum (entirely obtained from Germany), the value of this salt at wholesale in 1898 having been $4·22 per lb.

Lepidolite has also been found in the United States at Mount Mica, Me., where there has been a small production in previous years, and in Connecticut, where there is a large body of low grade mineral, but in neither of these localities is the quantity or quality equal to that in California.

Spodumene, which is also valuable for its lithia contents, and is employed to a slight extent in the manufacture of certain kinds of porcelain, is found in South Dakota in connection with the tin mines around the base of Harney Peak, especially at the Etta mine, where the crystals are said by Dr. F. R.

Carpenter to sometimes attain a length of 35 ft. An experimental lot of 500 lb. of spodumene was shipped from South Dakota to Paris in 1898, and there were inquiries for the mineral from New York and Philadelphia.

Metargon is a new gas, isolated from the atmosphere in 1898. It is not changed by sparking with oxygen in the presence of caustic potash, and has a spectrum which shows a strange resemblance to that of carbon monoxide. The density of metargon is the same as that of argon, viz., 20 or 22, reckoning oxygen as 8.

Monium.—Sir William Crookes in his presidential address at the annual meeting of the British Association in September, 1898, announced the discovery of this new element belonging to the yttrium group. Its atomic weight is probably in the neighborhood of 118.

Neon.—Argon has been the subject of further study by Prof. William Ramsay and his associates, with the result that after repeated liquefaction and distillation of the liquified gases, a product of lower density was obtained which was no longer liquifiable under the existing conditions, namely, that of the temperature of air boiling under a pressure of about 10 mm. This gas was neon mixed with minute quantities of argon and nitrogen. Making allowance for these admixtures, its density would be 9·6. Neon is monatomic, its refractivity is low (0·3071 or probably lower), and it has a characteristic spectrum. Eighteen liters of argon contain only 50 c.c. of neon; consequently its proportion in the air is only about one part in 40,000. The existence in the atmosphere of new gases with densities higher than 8 or thereabouts had been foreshadowed by Dr. Johnston Stoney, and it is interesting to observe that neon fulfills the conditions foretold for one of these elements.

Osmium.—Dr. Auer von Welsbach proposes to use osmium for the filaments of incandescent lamps, and furthermore to coat the naked filament with a refractory oxide like thoria. Osmium is practically infusible, and resists heat under which platinum and iridium are vaporized. By heating osmium in a vacuum with an electrical current strong enough to volatilize platinum, it attains a luminosity hitherto almost unknown, and emits a white light of agreeable quality, color, and great intensity. These properties are increased by coating with a refractory oxide which adds to the resistance of the filaments. Osmium for this purpose must be free from all impurities except platinum, a small admixture of which renders the osmium more elastic without sensibly diminishing its infusibility.

Palladium.—S. O. Cowper-Coles produces palladium on a large scale by electrolysis of a solution of palladium chloride and ammonia, with addition of ammonium chloride. The electrolyte contains 0·6 to 0·7% palladium chloride and 1% ammonium chloride. The anode consists of retort carbon. The current used has a strength of 0·025 to 0·03 ampères per sq. meter and a tension of 4 to 5 volts.* For mention of the possibility of Canadian nickel-copper matte becoming a source of palladium refer to the paper on "Platinum and Iridium," elsewhere in this volume.

Polonium.—This new element, discovered by M. and Mme. Curie, exists as

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* *L'Ind. Electro-Chim.*, 1898, 2, 31.

a heretofore unknown constituent of pitchblende. It possesses in a four hundred fold degree uranium's mysterious power of emitting a form of energy capable of impressing a photographic plate and of discharging electricity by rendering air a conductor. The radiant activity of the new element needs neither the excitation of light nor the stimulus of electricity; like uranium it draws its energy from some constantly regenerating and hitherto unsuspected store. Chemically the new element resembles bismuth. It is precipitated from solution by H_2S and by ammonia. Its salts are soluble in acids and are precipitated by dilution with water. The metal is isolated by precipitation with bismuth sulphide and partly separated by heating in vacuum to 700° C., a sublimate being obtained. The spectrum of polonium emits no characteristic lines. The name was suggested from the country where the pitchblende was found.

Radium.—In continuation of their experiments with pitchblende M. and Mme. Curie and M. Bèmont claim to have discovered a new element, which they call radium, in addition to polonium previously reported. Radium is said to have a remarkable effect on sensitive plates, and some of its chlorides have a photographic power 90 times higher than that of uranium. In its chemical reactions it resembles barium, and has a similar spectrum. Both radium and polonium are more active in rendering air a conductor of electricity than any other known media. The rays emitted give good photographs with an exposure of half a minute, while those of uranium require half an hour. While not so strong as the Roentgen rays, those from both polonium and radium make barium-platinocyanide fluorescent. The rays also impress the photographic plate in the dark. So far radium has not been obtained free from barium, but there are strong reasons for believing it to be a distinct element. It is not precipitated either by sulphuretted hydrogen or by ammonium sulphide, or by ammonia; its sulphate is insoluble in water and acids; the carbonate is insoluble in water; the chloride is very soluble in water, but insoluble in concentrated hydrochloric acid and in alcohol.*

Selenium has recently been employed for producing colored glass. Rose-tinted glass is made by adding selenium directly to the ingredients in the melting pot, the depth of tint depending entirely on the quantity used, and also to some extent upon the character of the glass—whether it be hard or soft. An orange-red color is produced by mixing cadmium sulphide with the selenium before adding to the contents of the pot. The intensity of the yellow constituent in this case depends directly upon the proportion of cadmium sulphide made use of. A practical advantage claimed for this process is that it is not necessary to reheat the articles after being manufactured and to dip them in a coloring mixture, as in the ordinary process of making red glass.†

Silicon.—(By G. L. J. de Chalmot): "Silicon is produced commercially by the Willson Aluminum Co., at Holcomb's Rock, Va., the process invented by G. L. J. de Chalmot being employed. This process consists in the reduction of silica in the form of sand, quartz powder, or silicates, in an electric furnace in the presence of a metal or a metallic oxide, the metal reduced from

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* *Comptes Rendus*, Vol. CXXVII., p. 1215, Dec. 26, 1898.

† *Engineering and Mining Journal*, Dec. 18, 1897.

which is able to dissolve the silicon. Such metals are preferred as do not chemically combine with silicon, but simply dissolve it. When the mass cools and solidifies the silicon crystallizes out, and is separated from the metal by treating the alloy with acids which dissolve the metal and leave the crystalline silicon unattacked. Such part of the silicon as is chemically combined with the metal is lost in this operation, because it is transformed into silica, which has to be separated from the remaining silicon by means of hydrofluoric acid. Although it is therefore better not to use such metals as combine with the silicon, they may be used: for example, copper.

"The alloy of copper and silicon contains, besides free copper and free silicon, also a combination of silicon and copper. If this alloy is ground finely and mixed with a proper amount of sulphur and heated in absence of air at a temperature not exceeding 300° C., the sulphur replaces the silicon in combination with copper, and the silicon which is liberated can to a large extent be recovered together with the original crystalline silicon. This process is described in United States Patent No. 602,632; the original process was described in United States Patent No. 589,415. So far as we know no experiments with any other process have been made on a commercial scale, either in Europe or in the United States.

"As to the possible uses of silicon, it may be said that this substance alloyed with different metals hardens them considerably, even if it is used in a very small amount. Silver, for example, might be a better material for coin, if hardened by silicon. It is further well known that silicon is a powerful reducing agent, and some uses in that direction may be found. Silicon has been repeatedly tried as a material for electric heating apparatus, for which it is especially adapted, since it is very hard to fuse, and although it conducts electricity, its resistance is so great that it is very economically heated by an electric current. Several inventors have tried to utilize silicon for the above purpose, but their inventions have not been practicable, mainly on account of their not being able to obtain a sufficient amount of raw material. Being a conductor of electricity, and not being attacked by acid oxidizing agents, it is one of the few materials outside of platinum that is fit for electrodes in electrolytic processes where the solution has to be kept quite clean and where carbon electrodes are not suitable. A well-conducting metallic electrode covered with a thin compact layer of silicon will be the ideal electrode, both as to efficiency and cheapness."

The Willson Aluminum Co. has been making silicon for the last two years, and is prepared to sell crystalline silicon, 95% pure, for $10 per lb. in lots of 500 lb. or more. So far the company has made silicon only for experimental purposes, a few pounds at a time, but it will begin the manufacture on a commercial scale as soon as a suitable market can be found. According to the London *Electrical Review* carbide of silicon is now manufactured in the form of filaments for incandescent lamps at Huyton, in England, by the Premier Electric Lamp Syndicate. It is claimed that this substance is economical and lasts longer than any other filament known.

M. Ferdinand Le Roy, in a paper in *Comptes Rendus*, suggested the substitution of resistances of crystallized silicon in place of metallic resistances when electrical heating is required. Its resistance, specific heat, and emissive power are high. It is easy to prepare sticks of pure agglomerated silicon, 10 cm. long and 40 square mm. in sectional area, with a resistance which varies with the physical condition of the silicon, but may be as much as 200 ohms, while the resistance of similar rods of carbon is only 0·15 ohm, and of German silver only 0·00085 ohm. It follows that, for a given resistance, the rods of carbon must be much smaller than those of silicon, and the rods of German silver very much smaller still. The resistance of silicon diminishes as the temperature rises, and at 800° C. is only about 0·6 of its value at the ordinary temperature.

Tellurium.—An improved method of estimating tellurium was described by James F. Norris and Henry Fay, depending upon the fact that in the presence of a large excess of alkali, tellurous acid is oxidized by potassium permanganate to telluric acid.* The addition of potassium iodide and sulphuric acid to the solution results in the liberation of free iodine, which can be estimated by sodium thiosulphate in the ordinary manner. In estimating tellurium by this method a portion of the substance, about 0·150 g., is treated with 20 c.c. of a 10% solution of sodium hydroxide, and potassium permanganate is added until the meniscus of the brown solution shows a deep pink. The solution is then diluted to 400 c.c. with ice-water, and there are added 10 c.c. of potassium iodide solution containing 2 g. of the salt, and dilute sulphuric acid until the solution becomes clear. The liberated iodine is titrated with sodium thiosulphate. The value of the permanganate solution is found in the same way by liberating iodine from potassium iodide and titrating with thiosulphate. The deficit in sodium thiosulphate represents the tellurium dioxide present. The method is rapid, gives accurate results, and can be used in the presence of halogen acids.

Thallium.—According to F. Foerster the crude thallium chloride which is recovered from the dust from pyrites roasting is converted into sulphate and dissolved by the addition of a small excess of sulphuric acid. This solution is subjected to electrolysis between a platinum anode and a copper cathode with a current of 1·3 to 1·5 ampères per sq. meter and a tension of 3·5 volts, whereby the metal is precipitated in the form of scales or shining needles. This precipitate is collected, washed, dried, and then melted under a cover of potassium cyanide.†

Titanium.—A. J. Rossi,‡ of New York, proposes to produce a useful alloy of titanium and iron, containing more than 5% titanium, and not less than 10% Fe, together with some carbon, by supporting in a bath of iron, at an intense heat, a mixture of carbon and titanic acid, titaniferous iron ore, or other suitable compound, or a rich slag of titanium. The charge is heated in a crucible, packed in coke, in a small electric furnace. In one charge, 50 to 100 lb. of an alloy may be obtained containing about 43·5% titanium and 49·50% iron.

Uranium.—There was a production of 33 tons of 2,000 lb. of uranium ore in Colorado in 1898, according to H. A. Lee, Commissioner of Mines for that

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* *American Chemical Journal*, XX., pp. 278–283.

† *L'Ind. Electro-Chim.*, 1898, 2, 31.

‡ United States Patent No. 609,467, Aug. 23, 1898; English Patent No. 18,127, Aug. 23, 1898.

State. Of this 23 tons came from the Wood mine in Gilpin County and 10 tons from the Copper Prince mine in Montrose County. The entire product was exported to France. The average price paid was $10 per unit. The product probably averaged 50% uranium oxide. The production in 1897 was 17 tons ($9,010). The Copper Prince mine is situated on Rock Creek in the S. W. part of Montrose County on the east slope of the La Sal Mountains. The vein, which was located originally for gold and silver and copper values, is said to occur in limestone and to have been opened for 1,500 ft. along its strike, showing uranium ore at various places. Specimens of uranium ores were received in 1898 at the Colorado Bureau of Mines from El Paso, Routt, Montrose, Fremont, San Miguel, Dolores, Montezuma, Ouray and La Plata counties, but nothing was made known as to the commercial importance of any of these discoveries.

Uranium mining in Gilpin County, Colo., in 1898, was trifling and spasmodic. The Kirk mine, which was productive in 1897, was idle, but a few men were at work on the Wood mine nearby. The latter mine was operated by the American Uranium Mining Co., of New York and Central City. There was nothing of importance done in the uranium mines of the Black Hills in 1898.

According to H. Borntraeger the Patera method of determining uranium, recommended by C. Winkler, by which the ore is treated with nitric acid, the solution neutralized with an excess of sodium carbonate, and the uranium in the filtrate precipitated by caustic soda and weighed as sodium uranate, $Na_2U_2O_7$, gives tolerably accurate results with high grade ores such as pitchblende, but with low-grade ore there is danger of error through silicon going into solution and being weighed as uranate. Borntraeger recommends that the precipitate of sodium uranate be dissolved in nitric acid after glowing and filtered off from the silica, after which uranium is again precipitated with soda or with ammonia, being weighed in the latter case as uranous uranic oxide.*

Vanadium.—S. O. Cowper-Coles has succeeded in obtaining electrolytically brilliant metallic deposits of vanadium, the color being almost as white as that of silver, from a solution prepared as follows: Vanadic anhydride, V_2O_5, was boiled with an excess of caustic soda, the sodium vanadate thus formed being decomposed by an excess of hydrochloric acid. The proportions were as follows: 1·75 parts of vanadic anhydride (equal to 1 oz. per gal. of solution) were dissolved in 2 parts of caustic soda and 160 parts of water, to which 32 parts of hydrochloric acid were afterward added. The best results were obtained with a current density of 18 to 20 ampères per sq. ft., the e. m. f. at the terminals of the electrolyzing cell being 1·88. The solution was worked at a temperature of about 180° F. If higher or lower current densities were used red-brown oxide was deposited along with the metallic vanadium. A reddish brown non-adherent powder was deposited from cold solutions at the same current density. The solution when first made up was of a light greenish-yellow color. After boiling with a carbon anode for some time it became darker, and after passing a current it turned a rich dark green; after a good

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* *Zeits. Anal. Chem.*, 1898, XXXVII., p. 436.

deal of metal had been taken from the solution it appeared bluish. When a solution containing 2 oz. of vanadium to the gallon was electrolyzed the deposit was not so white as that obtained from a solution containing 1 oz., the deposit having a steely appearance.*

Xenon is a gas also isolated from the atmosphere in 1898 by Prof. Ramsay, in connection with neon and metargon. It has a higher boiling point than its companion gases, is denser than argon, and is therefore isolated with comparative ease. Its spectrum resembles that of argon in general character, the lines differing in position, however. The ordinary spectrum of xenon shows three red and five blue lines and the gas glow is blue. The jar and spark gap spectrum has a green glow and is characterized by four brilliant green lines which are intermediate between the argon lines.

Yttrium.—G. Urbain has arrived at the following conclusion in regard to the nature of yttrium: As no one has been able to separate yttrium (atomic weight 89) into earths of higher or lower atomic weight, there is at present no plausible reason for believing that yttrium should be considered a mixture.†

Zirconium.—The large deposit of zircon in New Zealand, mentioned in the last volume of THE MINERAL INDUSTRY, is said to have a thickness of 200 mm., and a superficial area of 105 acres. It lies only 250 mm. below the surface and rests on a bed of blue clay 600 mm. thick, below which there is a bed of sand. The zircon bed is made up almost entirely of that mineral.

F. P. Venable and A. W. Belden state‡ that ignited zirconia is practically insoluble in all acids except hydrofluoric, and is unattacked by heating with sodium carbonate.

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* *Institution of Mining and Metallurgy*, March 15, 1899.

† *Chemisches Repertorium*, Nov. 26, 1898.

‡ *Journal American Chemical Society*, XX., pp. 275-276.

SALT.

The domestic production of salt is obtained chiefly from New York, the Ohio Valley, Michigan, Kansas, Louisiana, Utah and California. Comparatively small amounts are obtained from Illinois, Nevada, Virginia, Massachusetts and some other States. In Massachusetts and California some salt is produced by evaporation of sea water in lagoons. Rock salt is mined in Louisiana, New York, Kansas and Utah, but by far the greater part of the American production is produced by solution in water, pumping from wells, and evaporation by artificial heat.

PRODUCTION OF SALT IN THE UNITED STATES.
(In barrels of 280 lb.)

Year.	California	Illinois.	Kansas.	Louisiana.	Michigan (c)	Nevada.	New York. (c)	Ohio and W. Virginia.	Utah.	Other States. (c)	Total Barrels.	Total Short Tons.
1893........	312,850	65,000	1,607,000	172,500	a 3,514,485	7,988	4,413,181	b 925,620	438,002	182,435	11,639,061	1,629,469
1894........	324,623	65,000	907,000	185,000	3,485,428	5,100	6,529,694	b 1,342,400	821,565	473,821	14,039,631	1,965,548
1895........	365,280	68,000	1,168,951	159,775	3,929,342	8,428	5,919,155	b 1,704,000	411,784	541,139	14,275,854	1,998,619
1896	462,450	55,660	1,624,121	173,117	4,321,144	7,818	6,714,394	b 1,829,771	178,393	343,440	15,707,908	2,199,107
1897........	484,650	58,764	1,224,980	209,392	5,470,621	5,540	6,172,631	b 1,010,300	657,340	528,705	15,822,923	2,215,209
1898........	667,293	57,796	1,810,809	120,300	6,728,005	5,325	7,625,809	b 1,044,036	121,429	515,592	18,756,394	2,625,895

(*a*) Amount inspected. (*b*) Includes the production of Pennsylvania. (*c*) Includes salt used for the manufacture of alkali.

California.—About 50,000 tons of salt were sold in California in 1898. This was produced chiefly around the Bay of San Francisco. Prices declined from 12 to 30%, and the trade was reported as never before in such a deplorable position. The reason of this situation is attributed to the change in the conditions under which the industry is conducted. Formerly the entire production was made by manufacturers under leases and contracts with the owners of the properties. These leases and contracts having expired, everybody who owns salt ponds has gone into the business, and consequently the entire market has been demoralized. The salt-making season in California begins about January 1, earlier or later, according to climatic conditions, and lasts until November 1. Refined salt (dairy and table) sold in 1898 at $4·25 per ton at the works. Coarse salt, or crude, as it comes from the evaporation ponds, was worth $1·25@$1·85.

Illinois.—The production of this State was about the same in 1898 as in 1897. Prices were 10 to 12% lower. There were no new producers.

Kansas.—The production of salt in the Hutchinson district increased in 1898. Prices ranged between 47c. and 60c. per bbl., including cooperage, which amounts to 25c., the average for the year having been 52c. including cooperage, and 27c. not counting it.

Louisiana.—The Iberia mines were operated in 1898 on about the same scale as previously. Plans were being made for a new mining plant with double the capacity of the present one. The entire product of these mines is rock salt. The Avery Rock Salt Mining Co., in which the Retsof Mining Co. of New York is interested, was organized early in 1899 to sink a new shaft 600 ft. deep at the Avery Island mine, which has been idle several years.

Massachusetts.—The production in 1898 was small, work being interrupted by rain, fog and mosquitoes. The product sold at wholesale at 28c. per bush.

Michigan.—There was no material change in the salt business in Michigan from 1897 to 1898, but, if anything, the product of 1898 moved a little more slowly than that of 1897. Some manufacturers reported a falling off in business, owing to increased competition and lower prices. The net price of salt at Saginaw from January 1 to April 1 was 52c. per bbl. In May it fell to 47c., and in June to 42c.; in July it rose to 43c. These prices include cooperage, which amounts to 15c.

New York.—The production of salt in the Warsaw district was less in 1898 than in 1897. In the Onondaga district, apart from the wells of the Solvay Co., whose product is used for the manufacture of soda ash, the production is no longer of any consequence in the trade. In the Warsaw district agricultural salt sold at $1·50 per 2,000 lb.; common, fine and coarse at $2·30; dairy and table salt at $2·90. These prices are about 50c. per ton better than in 1897. Elsewhere in western New York common fine salt, which sold at 55c. per bbl. January 1, advanced to 60c. in February, and later to 70c., which remained the current price throughout the rest of the year. These prices include the cost of cooperage, which varies from 20c. to 25c. per bbl. The output of most of the salt plants of New York in 1898 was bought in advance by the National Salt Co., of New York City. The production of salt in the Onondaga Reservation in 1898 was 2,473,381 bush. of 56 lb. The year was more than usually remunerative to the manufacturers, and if it had not been for the unfavorable weather in August and September, which in some instances nearly ruined whole crops of solar salt, it would have been still more so.

Ohio Valley.—There was little change in 1898 in the salt business in this district, which comprises parts of Ohio, Pennsylvania and West Virginia, except an advance in prices, owing to a better demand. Prices, which in 1897 ranged from 50c. to 65c., in 1898 settled at 65c. a bbl. including cooperage.

Utah.—The salt industry in Utah was not active in 1898, owing to the large amount of solar evaporated crude salt in stock January 1, 1898, about 85,000 tons being thus held. There was a falling off in the demand for salt from the metallurgical works of Montana, Idaho and eastern Nevada, and consequently there was little incentive to new production, the stock on hand being drawn upon to meet consumption. The amount marketed in 1898 was as follows: Refined salt, artificially evaporated, 600 short tons; refined salt from solar product, 9,500 tons; crude salt, 14,500 tons; artificial rock salt, 400 tons; rock salt mined, 1,000 tons—total, 26,000 tons. Crude salt ranged from 75c. per ton in bulk for metallurgical purposes to $3 for stock needs. Refined salt averaged $10 per ton, including packages. Natural rock salt sold at $1@$1·50 per ton, while the artificial rock salt brought $8·50. The total value of the product marketed in 1898 was $120,000. The refinery of the Intermountain Salt Co. was destroyed by fire early in the year. Afterward the Intermountain and Inland Crystal Salt companies consolidated, the new company retaining the latter name. The Intermountain Salt Co. held a lease of the plant of the Nebo Salt Manufacturing Co. of Nephi, and the new consolidation now practically controls the refined salt industry of Utah. The eastern Colorado market continued unsatisfactory in 1898, on account of the low freight rates from Hutchinson, Kan., and the ability of the refiners in the latter district to produce at a lower cost than in Utah. The market for Utah salt embraces western Nebraska, part of Colorado, Wyoming, Utah, Idaho, central and western Montana, Oregon, Washington, central and northern California, and Nevada.

SALT PRODUCTION OF THE CHIEF COUNTRIES OF THE WORLD. (a)
(In metric tons and dollars.)

Year.	Algeria.		Austria.		Canada.		France.		Germany.	
1893	19,008	$93,416	305,586	$9,811,404	56,525	$195,926	1,114,327	$3,317,471	1,173,565	$4,290,250
1894	17,830	74,763	311,597	11,655,048	51,890	170,687	890,607	2,287,624	1,257,527	4,339,578
1895	25,758	102,990	278,875	11,472,855	47,515	160,455	871,312	2,406,155	1,212,336	4,340,821
1896	19,658	85,121	308,932	9,309,068	39,880	169,698	1,042,614	2,473,568	1,303,319	4,416,952
1897	23,222	78,058	321,084	10,357,496	46,584	225,730	948,000	2,239,823	1,306,684	3,834,436

Year.	Greece.		Hungary.		India.		Italy.	
1893	18,329	$293,264	167,209	$7,010,400	853,180	$1,594,430	25,392	$96,721
1894	21,310	332,436	169,282	7,153,440	1,318,198	(b)	30,793	114,835
1895	22,238	335,800	169,395	6,456,480	1,120,832	(b)	29,315	110,190
1896	22,800	344,280	180,133	6,191,861	1,043,172	1,097,909	29,274	113,322
1897	20,421	308,357	171,711	5,375,788	937,888	975,260	31,526	117,304

Year.	Russia.		Spain.		United Kingdom.		United States.	
1893	1,351,056	$2,224,234	151,464	$85,052	1,954,813	$3,676,110	1,478,245	5,617,331
1894	1,354,218	1,723,200	186,121	228,374	2,271,687	3,818,145	1,783,133	7,578,315
1895	1,540,194	2,383,743	326,320	952,098	2,208,025	3,548,755	1,813,135	6,472,982
1896	1,347,852	3,026,000	521,751	1,153,880	2,054,715	3,333,065	1,995,017	5,328,290
1897	(c)	(c)	508,606	1,159,294	1,934,039	3,104,490	2,009,625	3,898,556

(a) From the official reports of the respective countries. For Austria, Hungary, Russia, Spain, and the United States, the production of all kinds of salt is given; Germany, rock salt and common salt; Greece, sea salt; France, rock and sea salt; Algeria, sea and rock salt; Italy, rock and salt from brine; United Kingdom, rock and brine salt; India, salt which is liable to British salt tax only, and does not include salt made in certain native States. (b) Not reported in the official statistics. (c) Statistics not yet published.

England.—The shipments in 1898 from the Cheshire district were 559,742 long tons of white salt and 48,178 tons rock salt, making a total of 607,920 tons, a slight increase over 1897. In boring for coal at Whitehaven, on the Isle of Man, in 1898, large beds of rock salt were discovered, and steps have been taken by a Liverpool syndicate to establish a salt industry there.

Peru.—The production of salt in Peru is a government monopoly. The output in 1896 was 168,945 metric tons, of which 83,000 tons were exported.

Russia.—United States Consul-General Holloway, in a communication to the

Department of State, January 10, 1899, described the salt resources of Russia. In addition to the well-known mines of the Crimea, one of the richest beds of rock salt in the Bakhmut district of the Donetz basin is exploited regularly, and the high quality of its product enables it to compete with all the other salt mines of Russia. Its production in 1897 was 306,451 tons. The salt wells near Bakhmut and Slaviansk produced 80,645 tons in 1897. In the northern Caucasus salt is produced in small quantities from small lakes. There are a few salt lakes in the government of Baku, the annual product of which amounts to 8,064 tons. In the Erivan government and Karsk region rock salt is produced only from large beds which run along the Russo-Turkish-Persian frontier, 398 miles along the Russian territory. At present the production of salt in that region does not exceed 32,258 tons per year, but there is reason to expect that after the construction of the Kars Railroad, with a branch to Erivan, the Trans-Caucasian salt will find its way to the Russian markets, and the productiveness of the local mines will increase. The Trans-Caspian region produced 24,194 tons of rock salt, and a like amount of lake salt comes from Turkestan annually. In Poland salt is produced only in the Tsekhotsensk works, which belong to the government and are near the Prussian frontier. At present these works are leased by the heirs of General Glinka-Mavrin and produce but 4,839 tons yearly, whereas 94,839 tons are required to supply the demands of the Polish governments. In Siberia the production of salt is comparatively small —from 32,258 to 48,387 tons a year. This is principally due to the fact that the salt-bearing regions are far from the markets, and to the absence of convenient ways of communication. At present salt is shipped to Siberia partly from European Russia (through Odessa) and partly from America. In western Siberia salt is produced from salt lakes, and in eastern Siberia from wells. There are in the Yakutsk region well-known rock-salt deposits, but until now they have not been worked, as they are far from inhabited localities. The prices of salt in Siberia are exceedingly high, amounting to $57·47 per ton; whereas in European Russia the price fluctuates between $12·77 and $3·19 per ton. The Russian salt industry began to develop as an important industry in November, 1880, when by an imperial ukase the excise duty was abolished. With the increase in production the importation of foreign salt has decreased greatly, amounting at present to but 8,065 tons per annum, which goes principally to the Baltic region and Poland. During recent years salt from the Donetz basin has found its way to the above regions, and therefore it is to be expected that the import of foreign salt will become still smaller.

SILICA.

THERE is probably no mineral which has so manifold uses in commerce and industry as silica in its various forms. It is used extensively in the forms of sand, sandstone, and quartzite; as rock crystal, granular quartz, amorphous quartz, flint, and diatomaceous earth; while novaculite, or whetstone, pumice stone, and tripoli are silicious rocks in which silica is the chief constituent, and which are properly considered under this general caption. It is perhaps futile to attempt a complete enumeration of all the purposes for which silica is employed, and still less a consideration of them in a paper of limited length, while the collection of statistics to represent completely the industry is practically impossible, since a large part of the product is consumed for purposes of which no record is kept by either the miners or consumers. Quartz sand is used extensively in the manufacture of glass and silica brick and in the preparation of cement and mortar. Large amounts are used also for molding sands in iron foundries, for casting pig iron in iron smelteries, and in the preparation of furnace bottoms. Comparatively smaller quantities are used in the manufacture of sandpaper, as a scouring agent with sand blast apparatus, and as a filtering medium, while certain amounts are still used for spreading on floors of dwellings, etc. The chief use of sandstone, which is sand cemented by some foreign substance and consolidated by heat and pressure, is as a building stone, for which purpose it is one of the most valuable. Quartzite, which is harder and more compact than sandstone, and is usually white in color, is also employed as a building stone, but in some places is crushed for use by the glass makers and manufacturers of silica brick.

Crystalline and amorphous quartz is used in pottery, as a filler for acid towers in the chemical industry, as a smelting flux, and for lining Bessemer copper converters, and after pulverization is utilized in certain pigments as a wood filler, and in the manufacture of sapolio and other varieties of scouring soap. The silica used in pottery is commonly known as flint, probably because the true flint or chert, which owes its origin to chalk formations, was originally employed for this purpose. A good deal of such flint is even now imported by the potters of Trenton from the coasts of England and France, but practically all of the domestic production consumed by them under this name is crystalline, amorphous or granular quartz of dike or vein origin.

Pumice stone, tripoli, diatomaceous earth, and novaculite or whetstone are employed chiefly as abrasives, but tripoli is also used for the manufacture of filters; diatomaceous earth is employed extensively as an absorbent for liquids and as a nonconductor of heat; while novaculite is used in the United States by one company at least in the preparation of a special paving material.

Diatomaceous Earth, often improperly called infusorial earth, is a silicious material, generally white in color and frequently pulverulent, which is found in many places widely distributed over the earth. Commercially it is most commonly known by its German name, "kiegelguhr." Pure diatomaceous earth consists of silica and water, the former in the colloidal condition and the latter partly combined and partly absorbed. Diatomaceous earth is seldom found pure, however, being contaminated commonly with carbonates of lime and magnesia, clay, etc., the percentage of silica being generally between 80 and 90. Diatomaceous earth is not affected either by extreme heat or by acids, but its silica, being in the colloidal form, is readily acted upon by alkaline solutions. Its specific gravity is less than 0·25, while with this low density is correlated excessive porosity. This great porosity is due to the method of formation of the material, which is the silicious membrane secreted by microscopical confervoid algæ, the diatomacæ or bacillariæ which lived or live in both salt and fresh water; so minute are these organisms that Ehrenberg estimated that 1 cu. in. of diatomaceous earth contains about 41,000,000,000 diatoms. It follows from the nature of this material that it comprises a vast number of air cavities capable of being occupied by any liquid absorbed. An average sample of diatomaceous earth will absorb from four to five times its own weight of liquid. When saturated, the density of the material increases to about 1·24. After calcination the specific gravity is increased about 50%.

Analyses of diatomaceous earth from various localities are shown in the following table:

	Germany.	Hanover.	Hanover. (Hermannsberg.)	Laugschiefer.	Oberohe.	Scotland.	Scotland (Aberdeen).	Tuscany (Santa Fiora).	Auvergne (Ceyssat).	Nevada (Storey County).	Maryland (Pope's Creek).	New Jersey (Morris County).	Virginia (Richmond).	Canada (New Brunswick, Pollett Lake).
SiO_2	68·01	86·4	84·15	80·30	96·85	92·0	95·34	79·0	87·2	81·08	81·53	80·06	75·85	*b* 80·487
Fe_2O_3	6·82	1·5	0·70	5·40	Tr..	2·5	3·65	3·0	......	......	3·33	......	2·92	0·951
Al_2O_3	7·13	*d* 1·6	1.40		Tr..			5·0	2·0		3·43	3·84	9·88	3·146
CaO	......	1·3	1·75	0·44	......	......	......	......	......	......	2·61	0·58	0·29	0·342
MgO	......	......	*e* 1·10	0·43	......	......	......	......	......	......	*f* 5·63	......	*f* 1·63	0·283
H_2O	8·45	6·9	10·40	10·90	3·15	5·5	0·03	12·0	10·0	18·44	3·47	14·01	8·37	13·332
(*c*)	8·17	2·3	......	1·30	......		0·32	......	......		....		....	
	98·58	100·00	99·50	98·77	100·00	100·00	99·34	99·00	99·2	99·52	100·00	99·09	98·95	98·548

(*b*) 92 per cent. of the air-dried material was soluble in caustic potash. (*c*) Other volatile matter. (*d*) Clay (*e*) Magnesium carbonate. (*f*) Including K_2O and Na_2O.

Diatomaceous earth is used as an abrasive, as an absorbent, as a nonconductor of heat, for the manufacture of building and refractory brick, and in the preparation of water glass. Its value as an abrasive is due to the hardness, sharpness, and minute size of the individual particles of which it consists. As a polishing powder it is sometimes used crude and sometimes after levigation. Its value as an absorbent is of course due to its porosity. Formerly it was used largely in the manufacture of dynamite, but recently it has been replaced by wood pulp and other cheaper substances. In the manufacure of water glass 92% of certain diatomaceous earths has been found to be soluble in boiling caustic soda. Owing to the lightness of diatomaceous earth, brick can be made of it to weigh about one-fourth as much as ordinary brick. The use of these brick has been recommended in certain structures where lightness is required and in places whither carriage is difficult and costly, but of course the brick themselves are too expensive for general application. However, the Emperor Justinian (A.D. 522) directed his architect to use such brick in building the dome of Hagia Sofia at Constantinople. As a nonconductor of heat the value of diatomaceous earth is of course due to the almost innumerable air cavities which it contains, the heat waves having to pass repeatedly from air to silica and silica to air and suffering a corresponding loss in energy. Kieselguhr fire brick has been manufactured by several concerns. Formerly the diatomaceous earth was used exclusively as an absorbent for nitroglycerine in the manufacture of "giant powder." This use still continues, though numerous substitutes are employed, and there is an additional demand for the mineral as a material for steam-pipe covering.

United States.—Many deposits of diatomaceous earth are known in the United States, some of them being of extraordinary thickness. So far only a few of them have been worked. The American beds are commonly found in swamps representing filled-up lakes of the Glacial period. The most important producing district is Pope's Creek, Md., where the beds are worked by the New York Silicite Co. A large deposit was recently discovered in Florida. At Richmond, Va., there is an immense bed which is said to extend from north to south for more than a hundred miles, being 30 ft. thick in places. The earth of this bed is yellow and claylike in appearance. It is of Eocene age. The deposit in Storey County, Nev., is still worked intermittently, partly for "giant powder," partly for "electro-silicon." Deposits of diatomaceous earth have been reported from two localities in New York, viz., in White Lake, town of Wilmurt, Herkimer County, and on the shore of Cold Spring Harbor, Long Island. The latter is a fossil deposit in beds probably of Tertiary age. The White Lake deposit is the only one in use commercially at present. The material is dug from the bottom of the lake, which covers about four acres, and has a thickness of 2 to 30 ft., being covered by about 4 ft. of water. It is washed and run through strainers and pipes to settling vats, where it stands for 24 hours. The water is then drawn off and the material shovelled into the press, wherein it is made into cakes 4 ft. square and 4 in. thick. These are subdivided into cakes 1 ft. square and piled under sheds to dry. The product assays 12·120% water and volatile matter; 86·515% SiO_2; 0·449% Al_2O_3; 0·374% Fe_2O_3; 0·120% CaO; and 0·422% undetermined. Near Monterey, Cal., there is a Tertiary bed which attains a thickness of 50 ft. There are some huge deposits in Nevada, one of which is worked by the Electro Silicon Co.

The production of diatomaceous earth in the United States in 1898 was 1,392 short tons ($11,002), against 3,000 short tons ($30,400) in the previous year. The decrease was due to the suspension of operations at a deposit in Maryland and a falling off in the output of the producers in Nevada.

Canada.—Numerous lake deposit of diatomaceous earth occur in Nova Scotia and New Brunswick, being especially numerous in the marshy districts of Nova Scotia. One of the largest occupies the bed and shores of Folly Lake, on the Inter-Colonial Railway. Other occurrences are at Bass River, near Truro, and at Folly River Lake, in New Brunswick.

Europe.—In Europe the most important deposits of diatomaceous earth exist at Bilin, Bohemia, where it occurs above gneiss and a gypseous clay, protected by a capping of phonolite. At Lausitz it occurs above and below brown coal, and is covered by basaltic tuffs and lavas. At Naterleuss, in Hanover, there is a bed of kieselguhr, of which the quality is very pure and the specific gravity very low. It attains a thickness of 150 ft. and is covered by thin alluvial beds. This material is contaminated with more or less sand, and a purification by washing is required. The lowest stratum of the deposit is of a green color, owing to the presence of as much as 30% organic matter. Upon calcination the material turns red, on account of the ferric oxide which it contains. At Franzenbad, in Bohemia, there is a deposit from 6 to 8 ft. thick. At Altenschlirf and Steinfurth, in the Vogelsberg, there are beds from 18 to 24 ft. thick. Berlin is built partially on a deposit of diatomaceous earth. Deposits of diatomaceous earth have recently been discovered near the village of Akholtsikh, and in the vicinity of the villages of Kissatybe and Tskhordza, in the Caucasus, of which the thickness reaches 10 meters. Analyses show from 84·95 to 92·88% silica, the remainder being chiefly water, together with a small percentage of iron and lime.

Australia.—In New Zealand there are numerous deposits of diatomaceous earth, the best known being that at Oamaru. In Victoria the occurrence of these beds is fairly widespread. In New South Wales there is a deposit at Cooma which appears to be of greater extent and purity than any other in the colony. There are also important deposits on the Richmond River at Wyrallah, and in the Warrumbungle Mountains. A deposit at Barraba has also been described. For particulars as to the Australian deposits of diatomaceous earth reference should be made to the records of the *Geological Survey of New South Wales*, Vol. V., Part III, 1897, p. 128, where they are described by G. W. Card and W. S. Dun.

FLINT AND QUARTZ.—Cretaceous flints are at present largely imported from Dieppe, France, selling for $1·75@$2 per long ton at Trenton, N. J., where they are used in grinding clay, being placed in revolving pans together with water and clay. They are also ground and calcined to whiteness, to mix with china clay to make certain grades of porcelain. Vein quartz is largely substituted for flint in china making in the United Sates; but the true flint is said to be preferable for the finer products. There are large amounts of such flint in certain chalk formations in the United States, especially in central Texas, near the city of Austin, and at various places in Florida, but these are too inaccessible to give any commercial value to the product.

Amorphous and crystalline quartz is produced in the United States, especially in Connecticut, Maryland, Pennsylvania and Alabama, usually in connection with feldspar, with the exception of Alabama, where a white granular quartz is mined at Riverton. In New York quartz is quarried for pottery at Bedford, Westchester County. White quartz of Potsdam age has been quarried for use as a wood filler at Port Ann, Washington County, N. Y. A considerable amount of quartz is also quarried for this purpose in Connecticut. A large ledge of quartz assaying 99·39% silica was opened in 1898 on the Saugatuck River, two miles from Westport, Conn. A plant for grinding the material was begun, and the ledge, which is said to be very favorably situated for exploitation, is expected to be productive in 1898. Quartz for acid towers is quarried in North Carolina. The production of vein and dike quartz reported to us by eleven producers in the United States was 35,593 short tons, valued at $73,313, in 1898, against 28,407 short tons, valued at $55,817, in 1897. Although these statistics are probably incomplete, it is believed that they represent the more part of the domestic production.

GRINDSTONES.—A considerable amount of quartzite, or a very hard sandstone, is quarried in Ohio and Michigan, especially in the former State, for the manufacture of grindstones, the production of which in 1898 amounted to 38,859 short tons, valued at $438,675, against 36,502 short tons ($342,186) in 1897, 31,301 ($294,338) in 1896, 36,389 ($290,378) in 1895, and 29,989 ($257,596) in 1894. The large increase in the production in 1898 was due to the general revival of business, particularly in the iron industries, and also to the large grain crop of 1897. A variety of stone produced in the United States for the first time in 1898 was "pulpstone."

"Pulpstone" is a special kind of grindstone used for grinding wood pulp for paper making. There are two kinds of pulp employed in the manufacture of paper, namely, "ground" and "chemical." The ground is produced by grinding blocks of wood pressed against the stone, either by steam or water cylinders placed on the castings surrounding the stone. For this purpose a stone is required which will stand a pressure of 25 to 75 lb. per sq. in. of cylinder while running at a speed of 250 to 300 r. p. m. A good stone will produce from 3 to 6 tons of pulp per 24 hours. Previous to 1897 the only place where pulpstone was produced in the United States was Peninsula, Cuyahoga County, Ohio, but these stones would not successfully withstand the pressure, and the main supply of American and Canadian paper manufacturers (who use about 2,000 stones per year) has been obtained from New Castle, England, under the names "Lombard" or "New Castle" stone. The Tippecanoe Pulp and Grindstone Co. opened a quarry at Tippecanoe, Harrison County, Ohio, in June, 1896, and from that date up to February 20, 1899, had furnished 338 stones. This stone is claimed to show a crushing strength of 4,230 lb. per sq. in., against 4,238 for the Lombard stone and 4,000 for the Peninsula. The composition of the Tippecanoe stone, with the corresponding figures for Lombard stone following in parentheses, according to analyses by

Rawlins, of Wheeling, W. Va., is as follows: 95·17% SiO_2 (94·07), 0·39% Al_2O_3 (0·73), 3·43% Fe_2O_3 (2·54), 0·31% CaO (0·36), loss on ignition 1·75% (1·17) —total 99·95% (99·96). The Tippecanoe stone absorbed 6·08% water against 4·62% for the Lombard.

MOLDING SAND.—According to D. H. Truesdale four qualifications are absolutely necessary in molding sand. These are, refractoriness, porosity, fineness and adhesiveness. The sand should consist, therefore, essentially of silica and alumina. The presence of lime, magnesia and metallic oxides is detrimental, since they increase its fusibility. Owing to the difficulty in obtaining sand with silica and alumina in precisely the correct proportion, it is best to make an artificial mixture, using a pure silica sand and a sand containing at least 60 to 70% of alumina. The proper composition of molding sand is silica 90% and alumina 10%. The object is to obtain castings with a smooth surface and sharp lines. Porosity is important in molding sand, owing to its assistance in venting the casting. With the use of good porous silica sand large and intricate castings that have been considered hard to make, owing to the difficulty in venting them, have been made with complete success without the use of a vent wire. As to fineness it is well known that smoother castings can be made with fine sand than when coarse sand is used. Alumina plays the part of bond in molding sand, but no more should be present than is necessary to hold the particles together, since any increase over the proper percentage makes bad and rough work by its melting and adhering to the castings, causing extra expense in cleaning them and danger of loss through blow holes, owing to the increased density of the mold which prevents the rapid escape of gases. In making cores a mixture of silica and oil gives the best results.*

PUMICE.—There was a production of 144 short tons of pumice stone, valued nominally at $720, against 1,700 tons ($8,500) in 1897; the entire product each year being obtained in Utah. The ash beds in Nebraska were worked on a small scale for local consumption, but their output was insignificant. The Chicago Pumice Co., which exploits the deposits of Utah, had some financial difficulties, which probably accounts for the falling off in the production, although it is doubtful if this product has been received favorably in the trade. The receiver appointed for the Chicago Pumice Co. has now been discharged and the company reorganized with a capital of $500,000.

The world's supply of pumice stone is still obtained chiefly from the Lipari Islands, near Sicily. The principal market for the product is Leghorn, but Messina is said to be taking its place to some extent.

Granulated pumice stone is now used in the preparation of the lead plates of storage batteries. The powder is mixed in with the molten lead the heat of which expands the air contained in the pores of the pumice stone and in this way produces a multitude of cells throughout the mass of lead. The porosity thus obtained is said to be so great that a plate 9×7·5 in. will absorb 5·5 oz. of water. About 10% of pumice stone is used, and it being electrically inert, its presence does not affect the efficiency of the plate.

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* Abstract of a paper read at the Cincinnati meeting of the Western Foundrymen's Association, Oct. 18, 1897.

TRIPOLI.—There was a production of 1,865 short tons, valued at $3,367, in 1898, against 1,631 short tons, worth $5,475, in 1897. In both years the output was made chiefly in Newton County, Mo. It will be observed that there was a large decrease in the average value of the product in 1898.

WHETSTONES.—The statistics of the production of whetstones, scythestones and oilstones in the United States are summarized in the following table:

PRODUCTION OF WHETSTONES AND SCYTHESTONES IN THE UNITED STATES.

Kind of Stone.	1895.		1896.		1897.		1898.	
	(a) Quantity.	Value.	(a) Quantity.	Value.	(a) Quantity.	Value.	(a) Quantity.	Value.
Hindostan (finished product)	300,000	$6,250	275,000	$10,000	250,000	$5,625	275,000	$8,225
Indiana sandstone (finished prod'ct)	100,000	7,500			200,000	4,000	100,000	2,000
Arkansas stone ("rough rock")	460,000	10,800	b340,000	50,000	100,000	4,000	40,000	1,600
Washita stone ("rough rock")	203,200	10,614			350,000	4,375	350,000	4,575
Miscellaneous oilstones (c)	....	....	....	....	1,677	167	....	2,500
New Hampshire (Indian Pond, etc.)	12,000	33,000	15,000	35,000	12,000	33,000	13,000	35,750
Vermont (Lamoille, etc.)	3,500	12,250	907	3,176	4,200	14,700	5,000	17,500
Ohio and Michigan grit	4,814	12,889	4,129	7,025	18,114	31,362	15,696	36,100
Total	....	$93,303	....	$105,201	....	$97,229	....	$107,900

(a) Quantities given for Hindostan, Indiana, Arkansas and Washita stones and miscellaneous oilstones are pounds; for New Hampshire and Vermont scythestones, and Ohio and Michigan grit the quantities are number of gross. (b) Finished products in 1896. The product in 1895 was rough rock. (c) Including Queer Creek, Chocolate, etc., estimated.

It will be observed from the above table that there was little change in 1898 in the output of whetstones, by which term is meant all stones manufactured from natural rock for sharpening purposes, including oilstones, scythestones, etc. The amount of Arkansas rough rock quarried was considerably less than in 1897, owing to an overproduction during the last three years. The selling prices for oilstones remained about the same as in the previous year, although the export demand, particularly for the European market, increased a little. The output of Eastern grit scythestones was about 10% larger than in 1897, due probably to their having replaced Western grits to a certain extent in the domestic market, and to a slight increase in the export demand. Prices for the domestic trade were a trifle stiffer than during three or four years previous, owing to the cessation of severe competition between two Western producers. Export prices remained unchanged.

The general condition of the whetstone business in the United States at the present time is good, with a promising outlook; at the same time, there is some prospect of lower prices from the fact that several parties have made recent attempts to put new stones on the market, encouraged doubtless by the general improvement in industrial circles. In the oilstone field, particularly, such competition is likely to result in loss to the present manufacturers, with doubtful benefit to the trade, inasmuch as the oilstone business is one which requires very long experience, and facilities which are only to be acquired by time, in order to give the consumer an article of reliable and tested qualities. The experience of the past has shown that new manufacturers have inevitably flooded the market with inferior goods, that have in the end resulted in loss to themselves and their customers.

SLATE.

There was a large increase in the production of roofing slate in the United States in 1898, which was due especially to the active demand for export. The domestic business was not so satisfactory, owing to the large production, severe competition among the producers, and more or less demoralization of prices. The increase in production was not really so large as appears in the subjoined table, owing to a change in the method of computing the production of New York and Vermont, which in 1898 is based on the shipments from that region. Since stocks on hand were drawn upon largely, the actual production was not so large as indicated.

PRODUCTION OF SLATE IN THE UNITED STATES (*a*.)

States.	1897.				1898.			
	Roofing Slate. Squares.	Value.	Mfrs. Value.	Total Value.	Roofing Slate. Squares.	Value.	Mfrs. Value.	Total Value.
California	(*b*) 400	(*b*) $2,800	(*c*)	(*b*) $2,800	(*b*) 400	(*b*) $2,800	(*c*)	(*b*) $2,800
Georgia	(*c*)	(*c*)	(*c*)	(*c*)	3,845	14,419	(*c*)	14,419
Maine	45,306	190,161	$59,960	250,121	26,389	112,945	$66,484	179,429
Pennsylvania and Maryland	558,220	1,716,700	353,400	2,070,100	688,698	1,981,322	(*d*) 390,000	2,371,322
Vermont and New York	285,040	775,800	165,870	941,670	377,752	717,729	62,812	780,541
Virginia	42,158	140,065	14,875	154,940	38,548	125,281	8,560	133,841
Other States	(*e*) 1,000	(*e*) 4,000	(*e*) 1,000	(*e*) 5,000	(*e*) 1,000	(*e*) 4,000	(*e*) 1,000	(*e*) 5,000
Totals	932,124	$2,829,526	$595,105	$3,238,576	1,136,632	$2,958,496	$528,856	$3,400,796

(*a*) Not including school slates and slate pigment, (*b*) As reported by the California State Mineralogist. (*c*) None. (*d*) Partly estimated. (*e*) Estimated.

Note.—The production of slate pigment, including Baraga graphite and various kinds of mineral black, in 1898 was 7,886 short tons, valued after grinding at $70,671, against 6,857 ($57,863) in 1897.

Export Trade.—The active demand for slate for export was the redeeming feature of the industry in 1898. The conditions governing this trade are somewhat peculiar. The foreign dealers in slate, who would take the Welsh product in preference to foreign, other things being equal, are unable to secure prompt delivery from the Welsh quarries, and consequently bid for the American product the same price as for the Welsh, feeling sure that stocks can be obtained whenever wanted. The profit of the American exporter, however, is dependent largely upon the steamship rates. When the latter are 10 to 12s. per ton slate can be exported at a profit. These figures prevailed in 1896 and 1897 and during the early part of 1898, but later in the year they rose.

The average price at the quarries of the roofing slate exported from Pennsylvania is about \$2·50 per square, or \$8·75 per 2,240 lb. The freight to New York comes to \$2·75, ocean freight (12s. 6d.) \$3, and unloading, storage, etc., \$1·44, a total of \$15·94, or 65s. 7d.=18s. 9d. per square At the end of 1898 Penrhyn second quality, 20×10-in. roofing slate sold at the shipping point in Wales at 142s. a thousand (about 7 squares, American measure) which is about 10s. per 1,000 more than the cost of American slate in London.

The steamship companies now permit the shipment of roofing slate in bulk (formerly it was required to be crated). During the first quarter of 1898 the rate from New York to London rose to 12s. 6d. About the beginning of April there was a further increase and in June 15s. 6d. was asked, but in July room was obtainable at 10s. 6d. In August 15s. was again asked. In November the quotation was 15 to 20s. The rates to Liverpool, Newcastle and Bremen varied between 10s. and 20s. according to the demand for room in vessels destined to those ports.

Slate Manufactures.—The business in mill stock was generally quiet throughout 1898, and prices were not satisfactory. However, there was an improvement in the demand for slate for electrical purposes. The production of blackboard material in Pennsylvania in 1898 is estimated at 1,250,000 sq. ft., against 1,400,000 in 1897. Early in the year blackboards were quoted at 10c. per sq. ft. for 3-ft. width, 11c. for 3·5 ft., and 12c. for 4 ft.; in September the quotations were 1c. higher respectively. Prices continued unsteady, however, and about the end of the year the manufacturers formed an agreement as to the business in 1899, establishing 10, 11 and 12c. as the minimum prices. School slates were in demand in 1898 for the export trade, but the domestic business was rather dull. An increased export trade was done also in marbleized slate. The production of slate pigment increased somewhat in 1898. In our statistics of this substance we include all mineral blacks made from slate, graphitic shale, etc., including such material as the so-called Baraga graphite. The production of slate pigment proper is made especially in the New York-Vermont region, where various colors, including green, red, olive, slate and dark-brown are prepared. The value of the crude slate is about \$1 per ton of 2,000 lb. at the works of the grinders. The ground pigment sells at \$6 to \$10 per ton according to quality. In Pennsylvania refuse slate is now being employed experimentally in the manufacture of brick.

California.—There was only a small production of slate in this State in 1898, but several new companies were organized to exploit various deposits, and an increasing output may be looked for. The Shasta Slate Co. undertook the development of deposits at Slatonis, in Shasta County. Work was started on the Andrews & Sperry slate properties, and the manufacture of roofing slate was begun. The quarries in Eldorado County, which had been closed down for some time, were reopened on a larger scale. Slate deposits in Mariposa County are also to be opened.

Maine.—Nearly all the producers reported decreased outputs in 1898.

Maryland.—The producers in the Peach Bottom district, which extends over the line from Pennsylvania, made a small output in 1898.

Minnesota.—Levi Dietz made a new discovery of blue and blue-black slate of good quality about 3 miles from Cloquet. The ledge is said to be of considerable width and longitudinal extent. The slate splits evenly and is suitable for the manufacture of roofing plates, structural forms and school slates. It is proposed to open a quarry. St. Paul and Minneapolis are now supplied by slate from Pennsylvania and Vermont, whence the freight is $3 per square, while from Cloquet it would be only $2·50. The St. Paul & Duluth and Great Northern railways run over the ledge.

Pennsylvania.—The slate industry in this State was affected unfavorably in 1898 by the low scale of prices, which compelled many of the quarries to reduce wages, leading to labor difficulties and in the case of two large producers—the Old Bangor and the Bangor Excelsior—the suspension of work at their quarries. The ruling price for roofing slate was so near the cost of production that profits were insignificant and in many instances absent. Notwithstanding this condition of affairs the slate workers at Bangor formed a union to maintain wages. A noteworthy event of the year was the disruption of the Bangor Roofing Slate Manufacturers' Association, which was formed in 1895 by twelve of the leading producers in the Bangor region, to guarantee the genuineness of roofing slate sold under the name of "Bangor." The sales of this association have been as follows, in squares:

	1895.	1896.	1897.	1898.
Domestic	102,700	136,700	139,600	150,000
Exports	2,870	7,330	50,820	75,000
Totals	105,570	145,030	190,420	225,000

There was considerable prospecting for new quarries in 1898, especially in the vicinity of Bangor and Pen Argyl, and several promising beds were opened. There was also much activity in the Lehigh region. Some beds of slate were opened near Carlisle, and their exploitation will be undertaken in 1899. The well known Hower quarry at Danielsville was purchased in January, 1898, by English capitalists, who are operating under the name of the Hower Slate Co. During 1898 over 90% of the output of this quarry was shipped to Great Britain and Germany. The Peach Bottom and Chapman Quarries regions, which are comparatively unimportant, showed small increases in 1898.

The Peach Bottom roofing slate is said to be the most expensive of any in the United States to manufacture. One reason for this is that the deposits stand perpendicularly and consequently the cost of mining increases rapidly as they are followed downward, the flow of water increasing with depth, although on the other hand the quality of the slate seems to improve. The slate is exceedingly strong and tough, and the cost of breaking it is very high. Peach Bottom slate is estimated to be about six times as strong as the best marble. In manufacturing the roofing slate the blocks to be split must be sawed to the length or size of the roofing slate, from 12 to 24 in. in length, and split from end to end, since they cannot be split from the side to the right thickness. Some other slates will split from the side, or any place where a chisel is driven, into the proper thickness of roofing slate, 10 ft. in length. A quarrier can split more roofing slate in most of the Lehigh and Northampton County quarries in one day than he can split in Peach Bottom in two.

Tennessee.—The Tennessee Slate Co., which has been the only producer of slate in this State, ceased operations in January, 1897, on account of transportation difficulties, the only means of shipping the product being a river that is very low most of the time. The nearest railway is 16 miles distant, and it does not pay to freight the product in wagons. The slate is of excellent quality and exists in large quantity. Dark-blue roofing slate and a light purple slab slate are the kinds produced. The quarries are at Chilhowee.

Vermont and New York.—The producers in this region suffered from the same competition and demoralization of prices as those in Pennsylvania. In August the Vermont Slate Co., known as the "Seagreen Trust," had to dissolve, owing to dissension among its members, and with this restraint removed prices became unsettled immediately, and cuts of as much as 60c. were made on seagreen slate. Efforts to form a new association failed, and the competition continuing, many operators reduced wages and some closed down their quarries. At the end of the year this grade of slate was still offered at very low rates. Red slate and "unfading green" held fairly firm during the year. New York is the only State where red slate is quarried, and the producers maintain an understanding among themselves as to the amount which it is politic to put on the market.

Slate is being quarried near Hamburg, N. Y., by the Hamburg Slate Co., which is said to approach the Welsh in quality and color. The color is black. After heating to redness the edges show only slight flaking when cold. The deposit is very regular, and it is said that larger sheets can be quarried and put on the market than elsewhere. Specimens 12×6 ft.×1 in. thick, entirely free from pebbles or ribbons, have been exhibited; also specimens 4×5 ft. and ⅛ in. thick. The product of the Hamburg Slate Co., which began operations in 1898, is used for structural purposes, billiard tables, blackboards, etc. It is expected that other slate quarries will be opened in the vicinity of Hamburg.

M. D. Smith, secretary of the Board of Trade, Northfield, Vt., thinks that the slate quarrying industry of that place is capable of a good deal of development. The deposits are of great extent and lie only 5 or 6 ft. below the surface. The quality of the slate is very high. It is durable, holds its color, and splits into thin sheets of great strength with little or no waste. A vein of slate has been found in the east part of the township of Poultney, Vt., this being the first discovery of slate in that vicinity.

Virginia.—The small decrease in the slate production of this State in 1898 was due chiefly to the suspension of operations by one of the producers.

SODA AND SODIUM.*

THE production of soda, caustic, carbonate and bicarbonate, reduced to a common basis of 58% ash, in the United States in 1898 was 340,622 metric tons, against 277,072 in 1897. The amount actually marketed in 1898 as soda ash was 273,000 tons; as bicarbonate, 34,000; and as caustic, 48,000. Soda and soda products were manufactured in 1898 by the Michigan Alkali Co., of Wyandotte, Mich., the Mathieson Alkali Works, of Saltville, Va., and Niagara Falls, N. Y., the Solvay Process Co., of Syracuse, N. Y., and Detroit, Mich., S. D. Warren & Co., of Cumberland Mills, Me., and the Electro-Chemical Co., of Rumford Falls, Me. The last was in operation only a short time and its production was small. The Cumberland Mills, Rumford Falls, and Niagara Falls Works employ electrolytic processes. The Mathieson Alkali Works made chiefly caustic soda and bleaching powder, the product at Niagara Falls having been about 10 tons of caustic soda per day and 22 to 23 tons of bleach, using 2,000 h. p. The entire product of caustic soda and bleach is said to have cost a little over $20 per ton, against a cost of about $16 per ton in England. The Michigan Alkali Co. made all kinds of soda products, but caustic soda was the most important. The Solvay Process Co. made caustic soda at its Syracuse works and soda ash at Detroit. The North American Chemical Co., at Bay City, Mich., made potassium chlorate; it has a caustic soda plant, but this was not in operation in 1898. Several new soda works were planned. The Pennsylvania Salt Manufacturing Co. purchased property in Michigan, where the erection of works to use the ammonia soda process is contemplated. It was said, also, that the Pittsburg Plate Glass Co. intended to erect a plant to supply its factories with soda ash, but it is doubtful if this intention will be carried into effect, since the requirements of the company are comparatively small, and can undoubtedly be supplied cheapest by the large soda producers.

The American soda industry has naturally concentrated in localities where an abundant supply of cheap salt is available. This raw material is said to cost the alkali manufacturers of Michigan only 2·5c. per metric ton, while at Syracuse, N. Y., it comes to 25c.

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* Under this caption are considered also the manufacture of bleaching powder and some other substance which are not derivatives of sodium, but are recovered as by-products in soda manufacture, and consequently cannot be so well discussed elsewhere.

IMPORTS OF SODA PRODUCTS INTO THE UNITED STATES.

	1894.		1895.		1896.		1897.		1898.	
	Pounds.	Value.	Pounds.	Value.	Pounds.	Value.	Pounds.	Value.	Pounds.	Valu
		$		$		$		$		$
Soda, nitrate..	219,578,240	3,185,356	265,388,480	3,778,360	258,728,960	3,566,744	212,721,600	2,810,187	330,386,560	2,298,29
Soda, bicarb...	4,479,524	63,625	8,862,028	123,425	4,343,786	60,878	965,669	13,982	280,988	5,79
Soda, caustic..	44,772,512	911,942	72,019,114	1,211,090	47,827,247	859,809	57,742,392	988,318	24,981,873	354,29
Soda ash (a)...	321,300,874	2,665,836	307,026,094	2,321,612	191,780,537	1,444,417	154,656,060	1,122,867	73,064,707	447,11
Oth'r soda salts	13,238,190	68,760	1,081,887	31,581	1,279,285	40,039	7,927,145	100,989	23,354,295	256,66
Lime, chloride.	96,256,251	1,697,038	104,317,968	1,628,877	99,491,335	1,441,600	109,176,451	1,460,799	108,462,828	1,229,97

(a) Including sal soda.

Market Conditions.—The trade in soda ash was not so remunerative in 1898 as in the previous year, especially with respect to the domestic make. The average monthly price of domestic 58% ash per 100 lb. in lots of 50 tons or more, f. o. b. works, is shown in the following table:

Year.	Jan.	Feb.	March.	April.	May.	June.	July.	Aug.	Sept.	Oct.	Nov.	Dec.	Aver'ge
1897..	72·50	72·50	63·12	63·13	60·00	60·00	60·00	60·00	60·00	65·00	66·25	66·25	64·06
1898..	53·75	51·25	53·75	50·50	57·50	57·50	51·50	52·50	50·00	59·00	57·50	57·50	54·35

Caustic soda at the beginning of 1898 suffered from a heavy production and cutting of prices in consequence thereof. High test caustic soda, which at the end of 1897 sold as high as $1·85 per 100 lb., fell to $1·80 during the first week of January and then dropped to $1·65, while one large sale was reported at $1·45. Foreign caustic soda sold at $1·80@$1·95. Early in February the domestic product fell to $1·55 and the foreign to $1·70, and in March the former declined to $1·40, while the latter remained at $1·70. During April and May the price was $1·50@$1·55 for domestic and $1·65@$1·80 for foreign. From June to August the domestic product fluctuated between $1·40 and $1·50, the foreign selling about 20c. higher. At the end of August contracts for 1899 delivery were made by the American producers at $1·25@$1·30, spot goods being quoted at $1·40 and foreign at $1·60. During the last quarter of the year the domestic product ranged between $1·35 and $1·50 f. o. b. works and foreign at $1·60@$1·70 delivered.

Sal soda was in good demand during 1898, the domestic product gradually superseding the imported. In January the domestic sold at 55c. per 100 lb. and the foreign at 60@65c. In March the former fell 5c., and during the remainder of the year the quotation was 50@55c. for domestic and 62·5@67·5c. for foreign. Concentrated sal soda sold at $1·40@$1·50 per 100 lb. during the first quarter of the year, and later at $1·25@$1·75, according to make.

Domestic bicarbonate of soda sold in January, 1898, at $1·25@$1·50 for ordinary, and $3·25@$3·50 per 100 lb. for extra grades, while foreign sold at $2·25@$2·50. In the last quarter of the year ordinary domestic fell to $1@$1·25, while the finer grades ruled firm at $3·25@$3·50.

English prime brands of bleaching powder sold at $1·80@$1·85 per 100 lb. early in the year, while Continental makes were quoted at $1·70@$1·90. In March the former fell to $1·60 and the latter to $1·65, but with the opening of April the quotations were $1·75@$2 and $1·60@$1·80 respectively, and with few exceptions these prices were maintained until mid-year, though some sales of French bleaching powder were reported at $1·35. Early in July the American makers, being sold out for some time ahead, raised the price to $1·70@$1·80 f. o. b. works, while foreign makes were offered at $1·40@$1·65. In August the United Alkali Co.'s brands were obtainable at $1·375@$1·40 and other makes at corresponding prices. In September higher prices were asked, and at the close of the year prime English brands were quoted at $1·50@$1·625 for 1899 delivery, while spot business was done at $1·65@$1·75 for English and $1·40@$1·65 for other kinds.

The English chemical trade was under a cloud throughout 1898, and there was scarcely an article upon which prices yielded a fair profit. The exportation of alkali fell off from 225,124 long tons in 1897 to 163,509 in 1898, the United States taking only 39,305 in 1898, against 90,638 in the previous year. The exportation of bleaching powder fell off but slightly, having been 51,038 long tons in 1898 against 53,380 in 1897. The United States took 33,647 tons in 1898 and 34,075 tons in 1897. At the annual meeting of the United Alkali Co., Ltd., at Liverpool, March 17, 1899, the increasing competition of the American makers was again discussed. Attention was called to the large falling off in the exports to the United States, both of soda ash and caustic soda, and it was remarked that the American manufacturers have increased their production so that now they not only supply the home demand, but have a surplus for export. Attention was also called to the operations of the syndicate of German manufacturers which in 1897 bought up the entire production of electrolytic bleaching powder in the United Kingdom for a period of three years at a fixed price, and offered the product in 1898 at a lower price than was paid for it, causing a reduction of 20@30s. per ton in the market quotations. This contract does not expire until late in 1900. The net profits of the United Alkali Co. in 1898 were £198,263 against £213,364 in 1897, interest on bonds and all other charges being deducted.

According to the *Chemical Trade Journal* it is still doubtful as to the commercial results of the electrolytic processes of making bleaching powder. This product in Europe is being subsidized at present to the extent of nearly £2 per ton by the syndicate above mentioned, which will probably have to handle about 160 long tons per week in 1899. Early in 1898 bleach was sold at something over £6 per ton, but the price soon declined to £4 15s. for home sales, although export bleach was considerably higher. At this time salt cake was selling at 17@17s. 6d. per ton, a price which caused at least 8s. 6d. per ton loss to the most economical manufacturer. The next move was to bull salt cake and bear bleach. The electrolytic processes not being makers of salt cake, had a distinct advantage over those processes which made salt cake and sold it at a loss; but since the selling price of salt cake has risen more nearly to the cost price, and even so as to yield a small profit in some cases, only those processes can survive which yield *per se* the cheapest ton of bleaching powder. It has been proved that bleaching powder can be made by the improved manganese process for about 50s. per ton, f. o. b. cars, and it is doubtful if the electrolytic processes, or even the Deacon process, can make it so cheaply as that.

ELECTROLYTIC PROCESSES FOR THE PRODUCTION OF ALKALI.

One new electrolytic alkali process was brought out in 1898, namely that of James Greenwood, with which the Salt Union experimented at Winsford, England. The Rhodin cell was declared an infringement of the Kellner patents. According to J. B. C. Kershaw,* of the three English processes, the Castner-Kellner is being developed the most rapidly, the works in England originally designed for 4,000 h. p. being nearly completed at the end of 1898. The Richardson-Holland process at St. Helens has not been so successful as the promoters expected. The Hargreaves-Bird process continued to work satisfactorily on a small scale at Farnworth. The company which is being promoted to work this process upon a large scale in Cheshire has not yet been floated. The Hulin process was put in operation at Gavet-Clavaux about the end of the year, while the Hargreaves-Bird process was employed at St. Gobain, near Paris. Most of the remaining electrolytic works on the Continent use either the process of the Elektron Co., first operated at Greisheim, near Frankfort, or that of the Bitterfeld Electrochemische Werke. Numerous new electrolytic works were put in operation or planned in 1898, chiefly developments of the Greisheim and Bitterfeld companies.

No correct estimate of the total output of the electrolytic alkali works is yet possible. The Castner-Kellner works at Weston Point were producing at the rate of about 2,500 long tons of caustic soda and 5,500 tons of bleaching powder per annum in the latter part of 1898 and it is expected that this output will be doubled when the works are completed. The two plants of the Bitterfeld Electrochemische Werke are producing about 14,000 metric tons of bleaching powder per annum. Borchers in 1897 estimated that electrolytic alkali works in Germany were decomposing between 15,000 and 20,000 tons of potassium chloride per annum at that time, and this amount was undoubtedly exceeded in 1898.

At the beginning of 1899 electrolytic alkali works were in operation at the following places in Europe: Runcorn, St. Helens, Farnworth, Greisheim, Bitterfeld (2), Rheinfelden, Westeregein, Ludwigshafen, St. Gobain, and Hallein; in the United States works were in operation at Saltville, Va., Niagara Falls, N. Y., Rumford Falls, Me., and Cumberland Mills, Me. New works were under construction at Osternienberg, Geneva, Zombkowie, Clavaux, Montiers, Giffre and Livet; also one each at unnamed places on the Ebro in Spain, in Poland, and in France. In the United States a new works at Bay City, Mich., was nearly ready for operation.

The Lesueur Process.—The works at Rumford Falls, Me., employ the Lesueur process, the basic principle of which is the insuring that alkali shall not move from the cathode compartment to the anode compartment, which is accomplished by maintaining a higher level of the liquid in the anode compartment than in the cathode compartment. The inventor of this process claims that no electrolytic apparatus for this purpose can be commercially successful without making use of this principle, which is being infringed generally except in

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* *Electrical Review*, London, Jan. 13, 1899.

the mercury cathode process. In the Lesueur process, as in similar processes, platinum alloyed with 10% iridium is used for the anode.

The Carmichael Process.—At the extensive paper mills of S. D. Warren & Co. at Cumberland Mills, Me., the Carmichael process is employed. This is a continuous process which is claimed to be highly efficient, and it is said that no difficulty is experienced from use of diaphragms or from diffusion as in other processes, and no mercury is found necessary as in the Castner-Kellner cell.

The Solvay Process.—The new Solvay process of making soda electrolytically employs anodes of platinum with which about 10% of iridum is alloyed, this being to obviate the corrosion which takes place when pure platinum is used. According to Solvay & Co., the recombination of products separated by the electrical current is prevented by a liquid diaphragm. The liquid in the electrolytic apparatus is composed of two distinct strata. The lower, a solution saturated, or nearly saturated; over this a solution which is much poorer in salts. In the latter are the anodes. In the electrolysis of the chlorides by means of quicksilver cathodes these two strata may be maintained, as there is no gas released on the quicksilver. The lower and heavier layer fulfills the purposes of a diaphragm without its disadvantages. It remains free of chlorine, since it is constantly in contact with the quicksilver and does not mix with the solution surrounding the anodes, which is saturated with chlorine. The reproduction of alkaline chlorides on the cathode is consequently out of the question, and the full effects of the current are attained.*

The Hulin Process was put in operation in 1898 by the Société des Soudieres Electrolytiques at Gavet-Clavaux, in the valley of Bourg d'Oisans, near Grenoble. The motive force is obtained from the river Romanche. The Hulin process consists in the electrolysis of a fused mixture of lead and sodium chlorides by means of a carbon anode and a molten lead cathode. The chlorine is liberated as a gas at the anode; the sodium ions migrate to the molten cathode, which, being heavier than the chlorides, lies on the bottom of the vessel, and unite with it to form a sodium-lead alloy. From this alloy the sodium is recovered as sodium hydrate, and the lead as spongy lead by treatment with water at a subsequent stage of the process. The molten lead thus plays the part of the mercury in the Castner-Kellner process. It unites with the ions of sodium as they arrive at its surface to discharge their electricity, and prevents any loss by removing them at once from secondary action of the current or of the chlorine.

The presence of lead chloride in the electrolyte is necessary in order to insure the regular working of the process. Vautin found, when electrolyzing fused sodium chloride alone, that the surface of the lead cathode speedily became crusted with the lighter alloy; and that a high e. m. f. and loss of sodium resulted from this tendency. Hulin was met by the same difficulties at an early stage in his experiments, but according to Kershaw, by the modification described above he overcame it. The lead chloride undergoes electrolysis concurrently with the sodium chloride; and ions of lead and of sodium arrive at the cathode surface together. Under these circumstances a regular

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* *Zeitschrift für Angewandte Chemie*, Jan. 17, 1899.

and constant diffusion of the alloy into the molten lead takes place. In order to provide for a renewal of the lead chloride in the electrolyte, Hulin uses in each decomposing crucible an anode of lead in addition to one of carbon. A definite proportion of the total current is passed through this secondary anode; the chlorine liberated at its surface attacks it and forms lead chloride, which diffuses into the molten mixture. It has been found that if 12% of the current is diverted in this way, sufficient lead separates at the cathode to keep the process in regular operation. The practical details of this process were worked out by Hulin at the works of Messrs. Matussiere & Forest, at Modane in Savoy, in the years 1894–97. The only figures available relating to efficiency are those based on these trials. The power used was supplied by a 120 h. p. engine, and gave an electric current of 2,000 ampères at 32 volts. This current was passed through four of the specially designed crucibles containing the fused mixture of lead and sodium chlorides. Each crucible required an e. m. f. of 7 volts, when working with a current density of 700 ampères per sq. ft. The yield per electrical horse power per hour is stated to have been 81 g. chlorine and 54 g. sodium, or 1·85 kg. chlorine and 1·24 kg. sodium per day of 23 hours. In order that these results may be compared with those obtained by the wet processes Kershaw reduced them all to a common form as follows:

Process.	E. M. F. Required Volts.	Actual Yield in Grams.				Efficiency. %	
		Per Amp. Hour.		Per KW. Hour.			
		NaOH.	Cl.	NaOH.	Cl.	Current.	Energy.
Hargreaves-Bird	3·4	1,196	1,057	351	310	80	54
Castner-Kellner	0·4	1,363	1,136	340	284	91	52·3
Richardson & Holland	0·6	....	....	....	....	97·5	57·3
Richardson & Holland	0·4	....	....	....	....	97·5	56
Theoretical figures	2·3	1,495	1,322	650	574	100	100
Dry Processes.							
Hulin	7·0	1,052	907	156	129	63·3	41·5
Theoretical figures	4·2	1,495	1,322	356	314	100	100

The above figures are based on the results of experimental trials, with the exception of those for the Richardson & Holland process. These are mere estimates based on oral statements made to the late Dr. John Hopkinson and included in his report, and are evidently not very trustworthy. All the above processes are, or have been, in practical operation.

The electrical energy required to produce one ton of 70% caustic soda and 21 tons of 35% bleaching powder with the Hargreaves-Bird process is estimated at 2,609 kw. hours, costing $10 (steam power); with the Castner-Kellner 2,694 kw. hours, costing $10·36 (steam power); and with the Hulin, 6,106 kw. hours, costing $16·66 (water power). The Hulin process is thus seen to be less efficient, both as regards energy and current, than the various wet processes now operating upon an industrial scale. The yield of NaOH per kw. hour by the Hulin process is only 150 g., as compared with 340 g. by the Castner-Kellner process. As a set-off to this increased consumption of electrical energy there are, however, several advantages gained by the Hulin process. In the first place, the high current density which can be safely used with fused electrolytes makes it possible to obtain a comparatively large production from a small decomposing plant. The output for a given cathode area is 30 times greater by the dry process than by the wet ones. The second advantage is that by-products of considerable commercial value are obtained on lixiviating the lead-sodium alloy. Both spongy lead and lead peroxide are in great demand at present for forming the electrodes of secondary batteries, and either of these can be produced simply and cheaply by the Hulin process. The third advantage, which will help to compensate for the increased consumption of energy, is that solutions of sodium hydrate of high strength, and absolutely free from sodium chloride, can be obtained by simple lixiviation of the lead-sodium alloy without any evaporation. Solutions of 1·54 sp. gr. have been obtained by Hulin by repeated leaching of the cathode material; and it is evident that if such a strength can be regularly obtained in actual work, an enormous saving in fuel in the further operations will result. This advantage is one, however, in which the Castner-Kellner process also shares. The Hulin apparatus is protected by English patent No. 23,117 of 1894.

No figures showing the cost of the process have yet been published, but data touching upon this subject has been given above. The difference between the cost of energy by the wet and dry processes ($6·34 per 21 tons of bleach and one ton of 70% caustic soda) is in favor of the former, while the greater output per unit of cathode area and the production of valuable by-products are compensating advantages in favor of the latter. The operations of the Gavet-Clavaux works, which were started at the end of 1898, have so far shown results confirming those of the earlier trials at Modane.*

NATURAL SODIUM CARBONATE AND SULPHATE.

Natural soda was produced in 1898 in California, Nevada and Wyoming. The production amounted to 7,150 short tons, valued at $94,650, against 6,350 ($95,400) in 1897, the statistics being reduced to a common basis of 58% ash. The Occidental Alkali Co. of Nevada was idle during the year but is said to be contemplating improvements to its plant and resumption of operations. The producers in California were the Inyo Development Co. and the works at Ragtown. The former sold its product in 1897 as crude (unfurnaced) for borax manufacture, but in 1898 sold it as ash. The product of the Ragtown works both years was soda ash. Soda ash at Inyo and Ragtown was worth about $15 per short ton.

Operations were resumed at certain of the soda deposits of Wyoming, and some shipments were made from Laramie during the autumn. A large company, called the American Soda, Glass and Chemical Co., was incorporated to take over and operate the Union Pacific Alkali Works, which were built in 1895, and it is expected that this company will be a producer in 1899. The soda deposits are situated 13 miles distant from the plant, with which they are connected by railway. It is said to be the intention of the new company to erect works at Laramie for the manufacture of glass and chemicals. The American Development Co. and American Native Soda, Glass and Chemical Co. were also incorporated in 1898 to work soda deposits in Wyoming.

Deposits of sodium sulphate were discovered near Ophir, Wash., several years ago, but the Indian reservation, in which they occur, not then being open,

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* J. B. C. Kershaw, *Engineering and Mining Journal*, April 29, 1899.

locations were not made until last summer (1898). The deposit is situated in Okanogan County, about 14 miles from the mouth of the Okanogan River and 60 miles from Republic. It is only 3 miles from a boat landing, the boat connecting with the Great Northern Railway at Wenatchee. The deposits occur in small potholes in a lake bottom, which is 40 to 60 acres in extent. The potholes are from 3 to 20 ft. in diameter, and the soda is from 0·5 to 2·5 ft. thick. Between 10,000 and 50,000 tons of 95% mineral are estimated to be in sight. A deposit of sodium carbonate was discovered in 1898 in Sonora, Mexico, 2 miles inland from Adair Bay, 100 miles south of the mouth of the Colorado River. The deposit, which is 1 to 3 ft. in thickness, extends over 70 acres. A trial shipment of the product to San Francisco is said to have yielded good returns.

SODIUM NITRATE—CHILE SALTPETER.*

The production of nitrate of soda in Chile increased in 1898, but the prices realized for it were a little lower than in 1897, and the producers experienced another unprofitable and unsatisfactory year. The total exports from all the nitrate ports in 1898 amounted to about 1,254,000 long tons, against 1,060,000 in 1897, 1,088,000 in 1896, 1,218,000 in 1895, and 1,079,000 in 1894. Production in 1898 exceeded shipments by about 30,000 to 40,000 tons, and consequently larger stocks remained in the producers' hands. There was no combined restriction of production, notwithstanding great efforts in the early part of the year by many of the producers to reorganize the combination, but out of 80 existing *oficinas* about 30 were idle during the greater part of the year on account of the unprofitable prices. With a view of arranging a combination to regulate production for the next five years, a commission of experts assessed the producing power of all existing works at over 2,250,000 tons per annum. Many producers, however, being dissatisfied with their individual assessments, an immediate agreement to restrict on this basis is considered doubtful.

Influence of Perchlorates in Nitrates.—A good deal of effort was made in 1898 by partisans of ammonium sulphate to discredit Chilean nitrate as a fertilizer, owing to the presence of potassium perchlorate, which, it is claimed, is highly injurious to plant life. It is also objectionable in the manufacture of gunpowder. Attention having been thus directed to the question, it has been investigated carefully by the chemists of the nitrate *oficinas* and others. The latest information from Chile states that it has been found that the perchlorate is always present in certain of the liquors (*aguas viejas*), but only enters the saltpeter when the liquor in the pans, where the sodium nitrate (together with some other salts) separates out, remains standing a long time upon the saltpeter and evaporates to a certain extent. The cause of the difficulty having been discovered, care is now taken to obviate it, while at one *oficina* (the Santa Fé) at least a process has been developed for the separation of the potassium perchlorate and its recovery in a pure form. On December 6, 1897, a shipment of this material was made to Europe for the first time.

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* With respect to potassium nitrate, or niter, or saltpeter, refer to "potash," elsewhere in this volume.

The results of experiments at Darmstadt showed that perchlorate occurs in Chile nitrate of soda to the extent of 0·14 to 1·65%. Contrary to the conclusions of Sjöllema, who decided that a tenor of 0·5% would be injurious in agriculture, P. Wagner ascertained that ordinarily nitrate of soda containing as much as 1·65% is harmless.*

The necessity for determining perchlorate in nitrate of soda led to the devising of numerous new methods of estimation applicable in technology. Space permits mention of only a few of these. Chemists interested in the subject may refer to the descriptions of O. Foerster, *Chem. Ztg.*, 1898, p. 357; *Chem. News*, January 23, 1898; F. Freytag, *Zts. f. öffentl. Chem.*, IV., pp. 321-323; *Chem. Centr.*, 1898, I. (23), 1203; *Journal Soc. Chem. Ind.*, August 31, 1898; N. Blattner and J. Brasseur, *Bull. Soc. Chim.*, 1898, 19 (12), 539; *Journal Soc. Chem. Ind.*, August 31, 1898; and Mennicke, *Landw. Vesrsuchstat.*, 1898, 50, 230.

Prices.—According to the annual report of Thomson Aikman, Jr., of London, the lowest point touched during 1898 was 7s. 1·5d. per cwt. (112 lb.) in March and April, and the highest about 7s. 9d. in the autumn. The average spot price during the first half of the year was 7s. 4·5d., against 7s. 10·5d. in the corresponding semester of 1897; the average during the second half of 1898 was 7s. 6d., against 7s. 4·5d. in 1897. Refined nitrate in the summer months commanded a premium of 3d. to 6d. per cwt., owing to an export demand for the United States during the Spanish-American war, in addition to a good European demand. Of the visible supply a much larger proportion than usual passed into the hands of dealers, and at the end of the year very little remained in speculators' hands. Freights ranged from 23s. 9d. per ton in the early months to 34s. in November, closing at about 27s. 6d. In 1897 the range was from 16s. in May to 31s. 3d. in November and 25s. in December; while, during the three previous years, 16s. to 27s. 6d. was the range. Exchange during the early months ruled at 17·5d. to 17·75d., but, owing to the financial crisis in Chile and the return to paper currency, it fell to 12d. in August, and subsequently ranged between 12·5d. and 13·5d., closing at 12·5d. The price of nitrate in Chile ranged from 4s. 6d. per quintal in spring to 5s. in autumn, the closing value being about 4s. 8d. to 4s. 9d. In 1897 the range was 5s. 10·5d. in January to 4s. 9d. in December, while during the three preceding years 5s. 7·5d. to 6s. per quintal were the average rates. The shipment duty of 2s. 4d. and railway freight to shipping ports of 5d. to 7d. per quintal remained as before. The result of the year's working to producers was unsatisfactory in the extreme. At the price quoted for nitrate in London at the end of 1898 its nitrogen contents were 10% cheaper than those of ammonium sulphate, whereas two years previous it was 25% dearer. The combined action of the ammonia producers to popularize their product continued.

In New York the most interesting feature of the year was the attempt of eight out of the ten importers to create a "corner" at the time nitrate of soda was declared contraband during the Spanish-American war. In May $3@$3·75

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* *Landw. Presse*, 1897, Nos. 18 and 19.

per 100 lb. was quoted for spot goods. The corner was broken, however, by the arrival of cargoes about June 1, and by July 1 the quotation had fallen to \$1·70. In August \$1·425 was quoted, but in September there was an improvement, and in December the price was \$1·65. The average price of nitrate in New York in 1898 was \$1·77, against \$1·765 in 1897, \$1·775 in 1896, and \$1·78 in 1895.

Statistics.—The statistics of the nitrate industry are given in the following table:

NITRATE OF SODA SHIPMENTS, CONSUMPTION, STOCKS AND PRICES FROM 1891 TO 1898. (*a*)

	1891.	1892.	1893.	1894.	1895.	1896.	1897.	1898.
Shipments from South American ports to all parts......	759,000	784,000	929,000	1,079,000	1,218,000	1,088,000	1,060,000	1,254,000
Afloat for Europe Dec. 31....	321,000	215,000	358,000	393,000	506,000	351,000	394,000	565,000
Stocks in United Kingdom...	23,000	20,000	9,100	17,300	21,500	24,000	21,000	16,000
Stocks in Continental ports..	191,000	185,000	92,900	101,700	134,500	324,000	224,000	116,000
Consumption in U. K.........	120,000	117,000	100,000	114,500	117,500	105,000	108,500	132,000
Consumption in Continent...	703,000	685,000	682,000	749,500	789,500	805,000	867,500	900,000
Consumption in U. S.........	100,000	100,000	104,000	100,000	110,000	105,000	110,000	142,000
Consumption in other countries......................	....	....	....	....	7,000	8,000	14,000	12,000
Consumption in the World..	923,000	902,000	886,000	964,000	1,024,000	1,023,000	1,100,000	1,186,000
Visible supply Dec. 31.......	535,000	420,000	460,000	512,000	662,000	699,000	639,000	697,000
Price Dec. 31, per cwt. (112 lb.)	9s. 1½d.	9s. 1½d.	9s.	8s. 10½d.	7s. 7½d.	7s. 10½d.	7s. 6d.	7s. 7½d.

(*a*) In long tons; from W. Montgomery & Co.'s Circular.

METALLIC SODIUM.

Metallic sodium is produced by two works in the United States, but it is impossible to obtain statistics as to their output. In England the Aluminium Co., Ltd., of Oldbury, near Birmingham, is the only producer. Its output in 1897 was 85 long tons, valued at £12,750. Dr. Van't Hoff, in a lecture before the Society of German Scientists and Physicists in 1898, stated that the world's production of sodium in 1897 was 260,000 kg., all of which was obtained by electrolysis.

C. E. Acker, of East Orange, N. J., patented a process of making sodium and potassium by forming electrolytically an alloy of some heavy metal, such as lead, with sodium and potassium, and then distilling away the latter. The apparatus employed is a chamber in which the salts of the metals required are electrolyzed in connection with a fused metallic cathode with which they are capable of forming an alloy. Means are provided for circulating the molten metal, so that the richer surface alloy is conducted into another chamber in which the separated metal is volatilized out of contact with the electrolyte in an inert atmosphere, which is circulated through the chamber, and assists the removal of the volatilized metal.*

G. Wolfram patented a process for preparing the metals of the alkalies wherein carbides are introduced into the melted hydrates, resulting in the formation of alkali hydrate and volatile light metals. The vapors of potassium or sodium can be carried off and the metals secured. If the vapor is condensed, oxides free of water result, from which the peroxides may be formed by giving

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* English Patent No. 6,636, March 18, 1898.

them access to air or oxygen. Cyanides are formed in the presence of a compound of nitrogen.[*]

Estimation of Sodium and Sodium Salts.

Volumetric Determination of Sodium.—H. J. H. Fenton proposes to determine sodium volumetrically by mixing a concentrated neutral solution of the substance to be examined with an excess of potassium dihydroxytartrate and keeping the mixture at 0° C. for half an hour. The precipitated sodium salt is washed with a little ice-water, dissolved in an excess of dilute sulphuric acid, and titrated with potassium permanganate. The results obtained with sodium sulphate, chloride, nitrate, etc., are accurate within 0·3%. The presence of magnesium does not interfere with the accuracy of the method, but ammonium salts present in excess lead to low results. The method depends upon the fact that dihydroxytartaric acid in presence of dilute sulphuric acid is readily oxidized by potassium permanganate at the ordinary temperature.[†]

Sodium Bicarbonate, Detection of Monocarbonate in.—M. Kubli[‡] proposes as a test for sodium monocarbonate in the bicarbonate a 0·4% solution of quinine hydrochloride. If equal volumes of this and a 6% solution of the bicarbonate made in the cold are mixed, no immediate and permanent turbidity should be produced. In the presence of 2% of normal carbonate, a permanent turbidity is obtained at once.

Analysis of Raw Sulphide of Sodium.—According to F. Jean, the crystals of raw sulphide of sodium are crushed, and 10 g. dissolved in water and diluted to 1 liter. To 10 c.c. of this solution starch is added; it is then titrated with decinormal iodine. To another 10 c.c. of the original solution a solution of sulphate of ammonium (at 6·7 g. per liter) is added, in quantity equal to the number of c.c. of the solution of iodine used in the first case; then add about 30 c.c. of water and distil over the ammonia set free by the monosulphide of sodium. Then boil to drive off the sulphuretted hydrogen, and titrate the excess of acid: 1 c.c. of decinormal acid saturated with ammonia corresponds to 0·0039 g. of monosulphide of sodium. The liquid remaining in the retort is titrated with a decinormal solution of iodine: 1 c.c. of the iodine corresponds to 0·0079 g. of hyposulphite of soda.[§]

Estimation of Sodium Bicarbonate.—Prof. Lunge describes a rapid and sufficiently accurate method, which was used at the soda works at Trenton, Mich. It is based on the following reaction: $NaHCO_3+NaOH=Na_2CO_3+H_2O$. When all of the bicarbonate has been converted into the normal carbonate the addition of a single drop of caustic soda solution causes the mixture to give a brown coloration with silver nitrate solution on a test-plate. The total alkali is determined by titration with normal acid, and the two determinations give all the data necessary for the calculation. The caustic soda solution is made up from commercially pure NaOH to 20° B., precipitating with $BaCl_2$, saturating with barium hydrate and diluting to normal strength.[‖]

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[*] *Zeitschrift für Angewandte Chemie*, Jan. 17, 1899.

[†] *Proceedings of the Chemical Society*, 1898 (188), 21.

[‡] *Archiv. der Pharm.*, 236, 321; *Pharm. Jour.*, 61 (1480), 481.

[§] *Journ. de Pharm. et de Chim.*, VII., No. 4.

[‖] *Zeitschrift für Angewandte Chemie*, 1897, pp. 169–171.

OCCURRENCE AND RECOVERY OF SODIUM CARBONATE IN THE GREAT BASIN.

BY S. A. KNAPP.

DEPOSITS of various salts are numerous and widely scattered in the Great Basin. The high elevation and dearth of moisture in the atmosphere, coupled with the intense and long-continued heat of the summer, causes a very heavy evaporation (ranging from 7 to 11 ft. per annum) and gradually the waters of the great prehistoric inland sea have disappeared, leaving traces of their former existence in the widely separated lakes of the present time, and in the dry lake beds scattered all over the area formerly covered by water.

Origin and Character of the Deposits.—The present lakes owe their existence to the streams originating from the snows of the Sierras, which replace a portion of the evaporation, but the lakes are all gradually receding, and constantly increasing in density and strength of their saline contents, which are dissolved from the soil and decomposing rocks, and brought in by the inflowing streams. Some of these lakes have already reached a high degree of density, while others are still only slightly brackish.

The lake beds not reached by streams fed by regions of heavy snow are empty, and their bottoms are covered by a deposit of salts, thick or thin, depending upon the amount of salts present in the lake waters when the final precipitation and evaporation took place and the area over which the water was spread. The character of these salts depends upon the mineral constituents and characteristics of the country from which the lake received the drainage, modified, altered and changed, apparently according to the length of time exposed, the relative amount of moisture present and the amount of organic matter. In general these salts are a mixture of sodium chloride, sodium carbonate, sodium bicarbonate, sodium sulphate, and potassium chloride and occasionally biborate of sodium (borax), and different borates of lime. These occur in varying proportions (generally all are present to a greater or less degree), and it is upon their relative quantity that the separation of pure salts of economic value depends, as well as upon the conditions surrounding the deposit. The salts of economic value, under present conditions, are sodium carbonate and bicarbonate, and locally borax and calcium borate.

Geographical Distribution and Composition.—The deposits seem to be highest in the sodium carbonates along the eastern skirts of the Sierra Nevada, where the drainage of that range was received, and as we progress eastward the carbonates decrease and are replaced by sulphate until we reach the eastern limit in Wyoming, where the salts are almost entirely sulphates. This is best illustrated by reference to the analyses of the salts from the various deposits.

(*a*) *Owens Lake*, situated in Inyo County, Cal., near the eastern base of the Sierra Nevada, shows a density of solution of 11° B. and composition of solid salts as follows: Na_2CO_3, 34·95%; $NaHCO_3$, 7·40; Na_2SO_4, 14·38; $NaCl$, 38·16; KCl, 4·07; $Na_2B_4O_7+10\ H_2O$., 0·63; $(CaMg)CO_3(Al_2Fe_2)O_3$, 0·13; SiO_2, 0·28—total, 100% (Chatard).

(*b*) *Mono Lake*, Mono County, Cal., at the eastern base of the Sierra Nevada, shows a density of about 6° B. and composition of solid salts (in grams per liter), as follows: Na_2Co_3, 18·67%; $NaHCo_3$, 3·90; Na_2SO_4, 9·87; KCl, 1·83; NaCl, 18·50; $Na_2B_4O_7$, Al_2O_3, MgO, CaO, SiO_2, etc., 0·70—total, 53.47% (Chatard).

(*c*) *The Soda Lakes at Ragtown*, Churchill County, Nev., within the basin of the Carson River drainage, show a density of about 13° B. and composition of solids as follows: Na_2CO_3, 12·32%; $NaHCO_3$, 12·44; Na_2SO_4, 14.96; NaCl, 54·88; $Na_2B_4O_7$, 0·34; KCl, 4·06; $MgCO_3$, 0·75; SiO_2, 0·25—total, 100%.

(*d*) *Albert Lake*, in Oregon, on the eastern skirt of the Sierras, shows a density of about 4° B. and composition of solid salts as follows: Na_2CO_3, 27·09%; $NaHCO_3$, 12·44; Na_2SO_4, 2·68; NaCl, 54·58; KCl, 2.62; SiO_2, etc., 0.59—total, 100% (Russell).

(*e*) *The Double Springs Marsh* (a dry deposit), in Esmeralda County, Nev., about 4 miles east of the north end of Walker's Lake, within the basin of the Walker River drainage, shows composition in an average sample of the deposit of salts as follows: Na_2CO_3, 20%; $NaHCO_3$, 25; Na_2SO_4, 15; NaCl, 10; insoluble, 15; H_2O, 15—total, 100%.

(*f*) *In Ruby Valley*, Eureka County, Nev., a small dry deposit, shows an analysis of clean selected salts as follows: Na_2CO_3, 58·69%; $NaHCO_3$, 8·09; NaCl, 2·11; Na_2SO_4, 28·32; K_2SO_4, 2·79—total, 100% (Woodward).

(*g*) *Near Laramie, Wyo.*, a soda deposit, affords salts which show: Na_2SO_4, 44·55%; H_2O, 54·98; insoluble, 0·47—total, 100%.

(*h*) *Near Wilmington, Wyo.*, a soda lake yields salt which has the following composition: Na_2CO_3, 3·10%; Na_2SO_4, 71·37; insoluble, 22·82; NaCl, 1·83—total, 99·12%.

In the lakes the solution is generally uniform, slightly varying at points where fresh water flows in; but in the dry deposits the relative percentage of salts varies greatly in different portions.

SODA LAKES.—*Technology.*—The treatment for the recovery of the valuable salts varies with the material and its chemical composition. In the case of lakes or deposits in solution it may be divided into two classes: (1) the hot weather or summer process; (2) the cold weather or winter process.

The hot weather process depends upon fractional crystallization, the presence of both sodium bicarbonate ($NaHCO_3$) and carbonate (Na_2CO_3), and the fact that in dense solutions one molecule of the bicarbonate will unite with one molecule of the carbonate forming sesquicarbonate, known also as urao (Na_2CO_3,$NaHCO_3$+2 H_2O, but generally carrying a slight excess of $NaHCO_3$), which precipitates and has a definite crystallization of its own. To obtain this salt in a pure state other chemical facts are taken advantage of, viz., that sodium chloride will not precipitate unless the density rises above 32° B., its solubility being in 100 parts water, 35·52 parts at 32° F. and 37·25 parts at 140° F.; while sodium sulphate will remain in solution up to 34° B. providing the temperature of the solution is above 75° F., its solubility being in 100 parts water, 5·02 parts at 32° F., 37·35 at 80° F., 50·00 at 95° F., and 45·00 at 140° F. Hence by maintaining the temperature above 75° F. and keeping the density just below 32° B. the formation and precipitation of pure sesquicarbonate is allowed to progress and the contaminating $NaCl$ and Na_2SO_4 are kept in solution.

The method employed is to pump the natural solution from the lake and run it through a distributing canal to the evaporating and crystallizing vats, having impervious clay bottom and sides, and allow the solution to evaporate by solar heat. Thus the solution is caused to accumulate gradually at a density of about 20° B. until warm weather approaches, when it is allowed to run up to 25 or 26° B. and kept at that density by adding clear water from time to time. When the solar heat is sufficient to keep the temperature of the solution at 75° F. or more the density is allowed to run still higher but never to exceed 32° B., and crystallization commences. A sheet of the pure sesquicarbonate soon forms, covering the bottom of the vat. When the solution is exhausted of the urao it is run off to waste, and the vat is refilled with strong solution from the evaporating vats and the process repeated, the cake of sesquicarbonate gradually increasing in thickness with each successive precipitation.

This process continues until cold weather approaches (when there is danger that the temperature will fall so far that sulphate will be precipitated and contaminate the crop), and then the solutions are drawn from the crystallizing vats, leaving the cake of sesquicarbonate on the bottoms, varying from a few inches to as much as 12 in. in thickness. This cake is then broken so that men can handle the pieces and conveyed to a washer, where the soda is cleansed by a stream of water from a hose, under pressure from the pumps, of any adhering clay or dirt and piled up to dry in the air. The composition of the cake, if the work is carefully done, is generally $NaHCO_3$, 36·46%; Na_2CO_3, 45·86; $NaCl$, 0·32; Na_2SO_4, 1·25; H_2O, 16·16; insoluble matter, 0·02—total, 100·07% (Chatard).

After drying in the air the cakes are broken to nut size or smaller by a crusher, and the material is then calcined and roasted in a suitable furnace to drive off the excess of moisture and carbon dioxide and also any organic matter present, its composition being then (from material such as I have mentioned): Na_2CO_3, 97·58%; $NaCl$, Na_2SO_4 and insoluble matter, 2·43—total, 100%. This is an extremely pure bluish-white soda ash, which after grinding is ready for shipment.

Plant Required.—As will be observed, the plant required, aside from the pumping, crushing, roasting, and grinding machinery, is the canal to convey solution to the vats and the clay vats themselves. The canal is usually laid out to conform to the contour of the surface, with a very slight grade, and where permissible—if in clay ground—the water is allowed to spread to increase the evaporating surface. The vats are arranged generally in three tiers, parallel to each other, with a slight difference in elevation, so as to allow the solutions to flow from one to the other, the tier next to the canal, and taking water from it, being the highest, the next slightly lower, and the third the lowest of all. The bottom of the lowest vat must, however, be high enough to permit exhausted solution to be drawn off and run to waste.

Where a long line of vats is employed it is usual to raise the grade slightly from the pumping and central plant toward the extreme end, to permit the drainage of the strong solutions from the vats at the greatest distance toward the crystallizing vats, which are generally those nearest the works, and where the soda can be taken up, washed, and piled to dry with the least handling and expense.

The vats are generally dug 100 ft. wide and 400 to 600 ft. long, as circumstances and the ground permit, and about 3 ft. deep. To prevent washing of the banks by waves created on the surface by the wind and thus contaminating the soda, a lining of 1-in. boards 36 in. high is put around the edge of the vats, the boards being fastened to stakes driven into the clay and extending to the level of the ground surrounding the vat, and clay is tamped in behind the boards to keep the lining in place. The strong soda solution is so destructive that these boards do not last over two or three seasons, and it is essential to have a clay that is impervious to and will stand the decomposing action of the strong solutions. Most of the soda deposits are accompanied by such a clay in some portion, it being of a dark blue-color, very tenacious, and full of soda crystals, generally supposed to be of volcanic origin, and by many termed "volcanic mud."

The pumping plant may be adapted to the circumstances, surroundings, character of fuel, etc. At a typical and well-arranged works the pumping, crushing, furnacing and grinding is done at one central plant, thus economizing steam and labor. The pump is an 8-in. centrifugal, driven by a Westinghouse high-speed engine, its capacity being 4,000 gal. per minute. The crusher consists of a toothed cylinder revolving rapidly over a grate, the teeth of the cylinder passing between the bars of the grate. This appliance is the outgrowth of necessity, as the material is difficult to handle, owing to its tough character.

Calcining Furnace.—The furnace is also an evolution, brought about by necessity, and was adopted after repeated failures with other standard furnaces. It is a combination of the Howell-White revolving roasting cylinder, and a step-hearth reverberatory furnace, into which the former discharges. The first hearth of the latter is 8 ft. long by 8 ft. wide, and the second hearth 7 ft. long by 8 ft. wide, with 1 in. drop from first to second hearth. The furnace is fired from one end, the flame passing over the hearth of the reverberatory and through the revolving cylinder, a dust chamber at the end of the cylinder catching any fine particles carried over. This furnace has a capacity—when the soda is crushed fine—of about 30 tons per 24 hours. It consumes 4 cords of nut pine wood, and requires the labor of three men per shift of 12 hours to handle it and draw the furnaced soda. The principal difficulty with the furnacing is the high heat required, and at the same time to handle the amount necessary. The requirement of this high heat is due to a rather curious condition in this process.

In all these dense alkaline waters there is what is known as the alkali shrimp, a small crustacean which appears to develop in the solutions, where they attain considerable strength; they are very active and swarm through the solution, when the density is from 15° to 20° B., but when it runs above 20° they die, and their bodies rise to the surface, where they gather in patches. When the exhausted solution is drawn off these patches settle and become attached to and ultimately inclosed in the cake of soda. In the furnace, if it were not for the presence of these organic remains, a moderate heat of 600° to 700° F. would be all that would be necessary to properly calcine the soda, but owing to their presence when the soda receives the heat necessary to calcine it (which it does in the revolving cylinder), these remains become carbonized and give the product a grayish cast; hence, to clear the color and complete the process the soda receives a finishing heat on the reverberatory hearth, where the temperature is maintained just below the fusing point of carbonate (1497° F). The soda is held there, turned and raked until it just softens, when it is drawn. This manipulation renders the soda dense, which is a very desirable condition for the glassmaker, who desires to get as much material into his pots as it is possible to reduce at one heat.

When drawn from the furnace the soda falls into the hopper of a cooling conveyor, which is a sheet-iron trough with a water jacket on the bottom, through which the water from the pump passes on its way to the canal. This conveyor is 80 ft. long, the soda being moved along the trough by scrapers attached to an endless chain carried by sprocket wheels. The soda enters the conveyor red hot, and cooled to below 100° is discharged at the end into a storage bin. From this storage bin it is drawn to the grinding mill, where it is ground and packed for shipment.

Climatic Conditions Affecting the Process.—In the hot weather process the yield of soda—which only approximates 25% of the total amount available—depends not only upon the percentage of bicarbonate present in the original solution, but also upon the length and intensity of the warm season, and the character of the weather while crystallization is in progress. Consequently the weather indications and changes are closely watched. When a heavy wind comes up and strikes the surface of a vat filled with solution dense and crystallizing, it will throw down a slushy mixture of salts, including chloride and sulphate, on top of the cake of good soda, which would, if allowed to remain, lower the grade of the resultant cake. To obviate this, clear lake water is admitted to reduce the density, prevent precipitation, and also to re-dissolve the precipitated slushy salts. A heavy rain will reduce the density and prevent precipitation until evaporation again brings up the density. A spell of cold weather will precipitate some contaminating sulphate unless the density is promptly reduced. There are other factors constantly occurring which must be watched and guarded against to insure a high-grade crop at the end of the season; hence the soda maker's occupation requires constant watch of the solutions, weather, temperature, evaporation, etc., and the knowledge of what to do and the ability to do it promptly and at the right time.

The cold weather process is adapted to solutions containing only Na_2CO_3, $NaHCO_3$ and NaCl, the presence of Na_2SO_4 being detrimental, since this salt is precipitated into the resultant product, from which it cannot be separated. The process is substantially the same as the hot weather process, so far as accumulating strong solution is concerned, except that the solution is kept down to about 24° B. (so that no hot weather soda will be precipitated), and held until freezing weather comes, when the solution chills. All the carbonate, Na_2CO_3, unites with its water of crystallization, forming decahydrate (sal soda), Na_2O, 21·68%; CO_2, 15.39%; H_2O, 62·93%; which precipitates. The bicarbonate present falls as such, and the chloride (NaCl) remains in the solution and is drawn off and run to waste. The cake of soda is carefully taken up, cleaned of adhering dirt, and piled on protected platforms, which allow a free circulation of air, but keep off the direct rays of the sun, which when it is warm would cause the soda to melt in its own water of crystallization and run to waste. In the dry air of the Great Basin the soda gradually loses its water of crystallization, and in the spring, as warm weather approaches, the piles are broken down and the soda spread to dry. It gradually effloresces as it dries, until the whole mass of carbonate is reduced to monohydrate: Na_2O, 50·00%; CO_2, 35·48%; H_2O, 14·52%. The bicarbonate scattered through the mass remains still unchanged. The soda is then heated by a suitable furnace, generally an open plate furnace, to drive off the moisture and excess of CO_2 of the bicarbonate, leaving anhydrous carbonate or ash, a low heat accomplishing this result. A high heat, such as is required with hot weather soda, is not necessary, since the organic remains previously mentioned are not present in this soda. But few places have been found where this process could be adopted, one being at the Little Lake at Ragtown, Nev., where the tenor of sulphate is low, and the amount made is small. This soda is not dense, but on the contrary is rather bulky, its particular value and use being in the manufacture of washing powders, for crutching into common soap, in wool scouring, and sal soda manufacture.

Workable Lakes.—The solution deposits capable of being worked under present conditions are: Owens Lake, estimated to contain 22,000,000 tons of carbonate; and the Ragtown Lakes, with an area of 268 acres, which are 147 ft. deep at deepest point. Other deposits, which in time may be available to a certain extent, are: Mono Lake, in Mono County, Cal., estimated to contain 96,000,000 tons of sodium carbonate; Albert Lake, in Oregon, and Black Lake, a small lake near Benton, Mono County, Cal.

DRY DEPOSITS.—The dry deposits of soda are more widely scattered, and but few of them have yet received attention, unless they contained the more valuable borates. The composition of the salts differs very much in the different deposits, and within the deposits themselves varies in different portions.

The Double Springs Marsh, Nevada.—The only dry deposit worked for carbonate thus far is the Double Springs Marsh, in Esmeralda County, Nev. This is situated about 4 miles east of the head of Walker's Lake, and is 4 miles from railway transportation by the Carson & Colorado Railroad, which also reaches Owens Lake. As this is a typical deposit, a description of it and of the methods employed and projected will be sufficient for all.

This deposit comprises about 800 acres, 500 of which are covered with a deposit of salts varying from 2 to 14 in. in thickness, the average being about 6 in. The average composition of this surface incrustation is: Na_2CO_3, 20%; $NaHCO_3$, 25; Na_2SO_4, 15; NaCl, 10; sand and insoluble matter, 15; H_2O, 15—total, 100%. Beneath this is a body of damp soda clay, filled with soda crystals and strong soda solution, which is constantly rising to the surface by capillary attraction, and the water evaporating leaves the salts, which add to the deposit previously existing. This process is slow so long as the top incrustation is undisturbed, but when that is removed and the sun and air reach the clay it is rapid, the growth on stripped land being about 1 in. in depth per annum, its composition being about the same as the original top incrustation; hence the deposit, when worked, is constantly renewing itself.

In some portions of the deposit, beneath the top incrustation, it is found that the underlying clay has fissures or cracks running through it filled with pure crystals of sodium bicarbonate, forming a feathery mass, and known to the workmen as "feathers." Dried samples of this show 99% sodium carbonate. In other portions of the deposit, upon digging down into the clay a solid mass of crystallized sodium sulphate, of unknown thickness, is encountered, and in other places large masses of sal soda or decahydrate are found. Ditches dug in the clay fill with solution, and by solar evaporation it is brought to a high density, with good salts. The water level rises in the fall and lowers in the spring, and during the summer is at its lowest, the entire deposit being then perfectly dry, down to the clay before mentioned.

First Method of Soda Recovery.—When work was first inaugurated at this deposit the method employed was to dissolve the top incrustation in hot water, in a large open boiling pan, 36 ft. long and 7 ft. wide (running the density up to 26° B.), then conducting the solution to large circular settling tanks, 12 ft. in diameter by 4 ft. in depth, where all sand, dirt and foreign substances were settled out, leaving a clear strong solution, which was then conducted to large vats dug in the volcanic clay previously mentioned, where precipitation and crystallization were carried on in the same manner as was described in the hot weather process, the product being the sesquicarbonate, or urao. A considerable amount of very high grade soda was made by this process; but it was found that the destruction of the bicarbonate in boiling was very considerable, reducing a large proportion of it to monocarbonate, and as the amount of sesquicarbonate precipitated depended upon the amount of bicarbonate present in the solution, the yield was reduced below the profitable point.

Improved Method.—In order to secure an adequate yield a series of careful experiments and tests was made, and as a result a different process was adopted, the foundation of which is precipitation by converting all carbonates to bicarbonate, and the low solubility of bicarbonate in cold water.

The solubility of sodium bicarbonate in 100 parts of water is 6·90 parts at 32° F., and 16·40 at 140° F., while that of sodium carbonate is 6·97 at 32° F. and 46·20 at 140° F. Hence when a dense solution containing carbonates is treated with carbonic acid (CO_2) with formation of bicarbonates, a large proportion immediately crystallizes out, and when the temperature is reduced still further amounts are forced out of the solution.

By this means 80 to 85% of the total available soda can be recovered in the form of bicarbonate crystals, which being washed, dried, and furnaced, give a pure anhydrous sodium carbonate. As an example of the purity of the product obtained by this process, I append an analysis of bicarbonate produced in considerable quantity—25 tons at one time—at Double Springs Marsh, and as a comparison, following in parenthesis, is given an analysis of chemically pure bicarbonate: Na_2O, 37·00% (36·94%); CO_2, 52·30% (52·36%); H_2O, 10·70% (10·70%) —total, 100% (100%). With perfect furnacing this would give 100% sodium carbonate or 58·49% ash.

Solution and Carbonation.—The process consists of gathering crude soda and conveying it to the works on a light T rail track which is shifted along the surface to keep up with the stripping. The crude soda is then dissolved in hot water in an agitating tank with revolving arms. When the density has reached 27 or 28° B. the entire contents of the agitator or dissolver are drawn off into a settling tank, covered and inclosed so as to maintain the temperature, where all sand and dirt are settled out, leaving clear solution, which is then drawn off and conveyed by hose to the carbonating cylinder, a large cylinder of boiler iron 18 ft. long and 6 ft. in diameter, mounted on revolving rollers (much the same as a Brückner roasting cylinder), with a 12-in. opening in the center, for charging and discharging, fitted with a hinged door making a gas-tight joint with a gasket and screw clamp. When the cylinder has been filled with the strong clear solution, the door (which is then uppermost) is closed and sealed, and connection is made with a valve at the end of the cylinder by a hose leading from the carbonic acid receiver, where the gas is stored under pressure of 30 lb. per sq. in. When the pressure gauge attached to the head of the cylinder shows that the pressure in the cylinder has raised to that of the receiver the valve is closed, hose disconnected, and cylinder started to revolve. In about an hour the gas admitted has been almost entirely taken up in the formation of bicarbonate, and a sample of the solution is taken through a small pipe and valve, and quickly tested to determine whether more gas is required to complete the operation; if so the connections are made again and more gas admitted until the operation is completed. When the carbonates are practically all changed the cylinder is turned so that the door opens downward over a box from which a sluice with considerable grade conducts the solution and precipitated bicarbonate to the cooling and precipitating vat. The cylinder is cleaned out with a hose and pipe and steam jet, and is then turned over with the door uppermost, ready to receive a fresh charge of solution.

The cooling and precipitating vats are made of galvanized iron, 18 in. deep, set with a slight grade toward the discharge opening, and supported by a framework of timber. They are proportioned to the size of the cylinder, so that each vat holds one charge. They are arranged side by side and are worked in succession. As the solution cools in the precipitating vat the bicarbonate in suspension is rapidly thrown down. When the temperature falls under 70° F. it is not economical to hold it much longer, and the solution then containing all the chloride and sulphate, and some carbonates, is drawn to waste; but if it be hot weather it may be run into a clay vat for precipitation of any urao it may yield. This leaves the precipitated bicarbonate in the vat. Clear cold water is then run in so as to just cover the bicarbonate, the whole mass is stirred up with a hoe, and the discharge opening at the lowest point of vat—having been covered previously with a piece of sacking to act as a filter—is opened and the water drained off. This water dissolves any sulphate or chloride which is attached to the crystals of bicarbonate and also dilutes the mother liquor left behind. It is run to the sump and used again in the dissolver.

Calcination in Retort Furnace.—The bicarbonate is now drawn from the vat to a centrifugal machine, which quickly throws out the last remaining solution, leaving a mass of pure bicarbonate crystals. After this drying the bicarbonate is placed in a retort furnace, in charges of 3,000 to 4,000 lb. This furnace is an iron cylinder set on a slight grade in brick work, so that the flame plays all around it, and inside of it is a set of scrapers attached to a shaft which revolves slowly, thus keeping the mass stirred up and new surfaces constantly exposed to the heat. All openings and joints are gas tight, and as the heat is low (only 500 to 600°) it is not difficult to keep the cylinder in good condition. The gas evolved is conducted through a pipe to a cooling scrubber and thence to a compressor, which compresses and stores it under pressure in the receiver, whence it is drawn for use in carbonating a fresh lot of solution.

When the calcination is complete a door at the lower end of the furnace is opened, the shaft carrying the scrapers is revolved a few times, and soda ready for grinding is delivered to the conveyor, which takes it to the grinding mill, where it is ground and packed for shipment, the product being an extremely pure anhydrous carbonate without any organic matter or other impurities. This process is continuous, and can be worked winter and summer.

Cost of Production.—The cost of the treatment of the deposits, both wet and dry, is much the same, and where the conditions for labor and fuel are fair, may be stated generally at about $6 per ton of 2,000 lb. for ash at works ready for shipment.

Other dry deposits are Hot Springs Marsh, Teel's Marsh, Columbus Marsh, and Fish Lake Marsh, all in Esmeralda County, Nev.; Saline Valley, Inyo County, Cal.; and to the south and east of the last.

STONE.

The chief varieties of stone produced in the United States are limestone (including marble), granite, sandstone and slate. A review of the slate industry, together with the statistics of production in 1898, is given under a separate caption, while many details concerning the production of limestone of various kinds are to be found under that caption in an earlier part of this volume, although complete statistics of production were not collected. It is exceedingly difficult under any circumstances to collect statistics of the various branches of the stone quarrying industry, which shall be thoroughly representative, owing to the conditions under which this industry is prosecuted, a considerable production being made by individuals who keep no record of their work. The total production of stone in the United States in 1898, not including slate, or limestone used for smelting flux, or sandstone used for the manufacture of grindstones, which are reported elsewhere, was about $32,500,000, divided as follows, with the corresponding estimate for 1897 in brackets: Granite, $9,500,000 ($8,800,000); marble, $4,000,000 ($3,750,000); sandstone, $4,200,000 ($4,000,000); bluestone, $800,000 ($800,000); limestone, $14,000,000 ($12,650,000); total, $32,500,000 ($30,000,000).

Granite.—Granite is quarried in 27 States of the Union, but very nearly 50% of the total output is furnished by Massachusetts, Maine and Vermont, which rank in importance in the order named, and each make outputs valued at upward of $1,000,000 per annum. New Hampshire, Rhode Island and Connecticut also have large granite industries, and the combined output of the six New England States amounts to about two-thirds of the total for the United States.

The granite industry of Massachusetts is referred to in the special paper by Mr. Whittle which is appended hereto. The more part of the production is obtained from Quincy and Cape Ann. In Vermont the most important granite quarries are situated in the vicinity of Barre, these quarries usually making about two-thirds of the total output of the State. In Maine there are large quarries in the vicinity of Blue Hill, Millbridge and North Jay. The chief quarries of New Hampshire are situated in the vicinity of Concord, West Concord and Milford, while those of Rhode Island are located at Westerly and Niantic. A large part of the production of New Hampshire and Rhode Island is quarried for monumental purposes.

The chief uses for granite may be classified as (1) for building and structural purposes, partly cut and partly in the rough, (2) for road making or macadamizing and for paving in the form of rectangular blocks, and (3) for monumental and cemetery purposes. Approximately 60% of the total output is used in construction of all kinds, including not only houses, mills and stores, but also dams, retaining walls, bridge work, etc., while about 25% is used in the construction and maintenance of highways.

Sandstone.—Sandstone, including quartzite, is produced in 28 States of the Union, but the number in which the industry is of importance is comparatively small, there being only four States of which the annual product exceeds $250,000 in value. These are Ohio, New York, Pennsylvania and Connecticut, which rank in importance in the order named. Their combined product comprises about 70% of the total of the United States. The domestic production of sandstone is used chiefly for building and structural purposes, this not being a stone which lends itself well to use as road material. The variety of sandstone most commonly used in building is known as brownstone, the light-colored sandstones and quartzite, which are generally white, being of less importance. A good deal of a very hard, compact sandstone or quartzite is quarried in Ohio for the manufacture of grindstones, which subject is referred to under the caption "Silica," elsewhere in this volume.

Brownstone occurs in several places in Ohio, but the only one known in the general market is the Killbuck stone, which is quarried near Killbuck Station, on a branch of the Akron, Cleveland & Ohio Railway. The color of this stone varies from dark gray to dark brown, and on account of the variation in color it is not classed with the finer building stone. It occurs in a stratum from 8 to 30 ft. thick.

Brownstone is quarried in New York from the Triassic formation and the Medina and Potsdam beds of the Silurian and Cambrian, the Medina being the most important. The Potsdam quarries are situated in the vicinity of Potsdam, Port Henry and a few other places. The Potsdam sandstone is a fine-grained, strong, durable stone of pink and red color, so hard that it might almost be called a quartzite, and ranks among the most valuable building stones. However, the most important brownstone quarries of New York are opened in the Medina formation in the western part of the State, the principal quarries being situated along the New York Central Railway, west of Rochester. This stone is fully as brown as much of the Triassic sandstone, but it is hardly so uniform in color. It is used extensively for building purposes at Buffalo.

The brownstone quarries at Portland, Conn., are the oldest, largest and best known in the United States. At that place the stone occurs in regular beds, varying in thickness from a few inches to 12 ft., the entire thickness of the series being unknown, although it has been found to be more than 500 ft. in one place. The stone is fine grained and of a remarkably uniform dark-brown color. It has been used extensively in New York and other cities along the Atlantic coast. The production is made by three companies, namely the Brainerd, Shaler & Hall Co., the Middlesex Quarry Co. and the Connecticut Steam Brownstone Co. Brownstone quarries are also operated in Connecticut at Cromwell.

THE BUILDING AND ROAD STONES OF MASSACHUSETTS.

BY CHARLES L. WHITTLE.

THE building stones of Massachusetts are classified in the order of their importance into granitic rocks, sandstones, marbles and limestones, conglomerate or pudding stone, and slate.

Granite.—This State is pre-eminently the home of granitic rocks. Along the eastern border, facing the sea in some instances, the granites occur from Rockport, on Cape Ann, to Fall River and New Bedford. Other patches and belts occur near Fitchburg, Worcester and Taunton, and at Monson. In a strictly scientific sense, the granites are poorly represented in Massachusetts, the rocks commonly so classified being varied in their composition and structure. There are few rocks that are being exploited at the present time within the borders of Massachusetts, outside of the granites quarried at Fitchburg, which are comparable with the type represented by the massive granites of New Hampshire, with their component orthoclase, quartz, muscovite and biotite micas. Most of the granites are in reality granitites or gneisses. The first of these classes differs from granite proper in the absence of white mica, which gives a pronounced character to such well-known stones as those from Concord, N. H., Hallowell and North Jay, Maine, and from many other localities. The greater part of the granitites, like the two important and widely known stones from Quincy and Rockport, are composed essentially of orthoclase, quartz, and hornblende, while the Milford granite has a great amount of black mica in place of hornblende.

Under the head of granite there are also quarried gneissic rocks in several localities. Gneisses are crystalline rocks which possess a more or less well-marked banding, due to the arrangement usually of the micas in defined planes. In the beautiful Milford granite this banding is observable, although not sufficiently to detract from the value of the stone, or cause an appreciable decrease in crushing or other strength tests. A more marked example is found at Monson, where a characteristic gneiss is quarried on a large scale. This rock was once a true pudding stone which has lost nearly all its original characters and become transformed into gneiss.

The enormous development in the two largest granite regions, Quincy and Rockport, has taken place as a result of the excellent quality of these stones, the early opening of the quarries, and the fortunate location, particularly of some of the quarries on Cape Ann, close to the sea. The opening of the Railway quarry in Quincy, for the purpose of obtaining stone for the monument at Charlestown, placed on the market a granite of a blue-gray color, which for many years was essentially the only variety of granite quarried in the State. It has thus become the standard of beauty for structural and monumental purposes throughout Massachusetts and the United States as well. This position was held without serious competition until the exploitation of the granites near Gloucester and Rockport, which were discovered in an effort to find other granites like those of Quincy.

The granite industry in Quincy has steadily grown since 1810 with little decrease in the relative demand for this type of stone. The business consists in supplying the cheaper forms of stone for substructures, retaining walls, paving blocks, etc., as well as the higher class for constructional and ornamental purposes. Within the last few years immense quantities of the lower grades have been used in the work of raising the tracks of the Boston & Providence Railroad, near Boston. The Quincy granite is not without differences of color, although its structure and grain are very constant throughout large areas. In the town of Braintree a red phase occurs which has much merit as far as color is concerned, and has entered the field as a competitor of Scotch granite as well as "Red Beach" from Maine.

The granite of Cape Ann is also very generally used in the East, but along somewhat different lines. The possibility of placing orders requiring blocks of exceptional dimensions has given this stone an enviable reputation. Other advantages are found in the nearness of some quarries to the water's edge, where the stone is loaded directly upon transports for shipment. Rockport granite is also fortunate in the perfection of its planes of rift, lift and cut-off, thus permitting the economical extraction of paving blocks and a proper working of the quarries. This industry is carried on very extensively and the production of paving stone exceeds that from any other part of the State. Within a year one of the large quarries has uncovered a massive bed* of granite about 30 ft. in thickness, free from knots and blemishes. At the present time this is being successfully taken out by means of plugs and shims alone.

To the ordinary uses to which granite is put there must be added the somewhat recent demand for the sap portions for decorative ashlar work and rustic masonry. Those quarries which possess sap rock of the desirable colors are earning a new and adventitious revenue. In fact, the increasing demand for seam-face granite and other rocks has led to the opening of several granite quarries in the towns of Weymouth and Hingham, which produce only stock of this kind. The granite here is traversed by a parallel set of joints (see engraving) from a few inches up to 2 or 3 ft. apart, thus permitting its economical extraction. The joint faces are usually stained various shades of dark or light green, due to the occurrence of chlorite or epidote, or they may be colored yellow to various shades of brownish red. These latter stains are produced by one or more oxides of iron.

By a fortunate relation between the general parallel set of joints and a cross set, it is possible to extract coigns having intersecting joint planes making angles of 90° and 120° in such abundance as to supply large orders. Stock of this kind, while not unusual in most quarries, seldom occurs in large amount or with such essential constancy of angle as to be considered of especial value in itself. The demand for seam-face granite is steadily increasing; its introduction is becoming more general and its artistic merits are winning better appreciation.

Not far southeast of Worcester, the Milford granite quarries have been recently opened and are furnishing an important percentage of the State's production. This stone has won its introduction mainly on its color merits, and is a severe

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*The word "bed" is used in the quarryman's sense only.

Example of Seam-Face Granite.

competitor of the older, better-known granites. Milford granite possesses a slightly banded, or gneissic structure, but any criticism due to this fact is commonly remedied by working the stone in such a manner that its gneissic structure stands vertically and parallel to a given building face—a position exactly the reverse of the proper attitude for common bedded sandstone. This stone is being widely used and may be seen in the new Boston public library and in many other buildings throughout the metropolitan district and New England. Many other granites are quarried and used somewhat more locally, as those about Fall River, Taunton, New Bedford, Fitchburg, Peabody, etc. The Fitchburg granite is being used at the present time very extensively in the new dam at Southboro, Mass., built by the Metropolitan Water Board.

In the town of Monson, as previously mentioned, there is a growing granite industry. This granite, or, more properly, gneiss (for it is decidedly banded), has peculiarities that adapt it for flagging, curbing, and paving purposes, or for any use where thin tabular stock is required. It is also adapted to most applications for which granite is suitable, outside of uses requiring polished surfaces. Its field is largely in the central portion of the State. There are many other locally used gneisses which well serve the needs of a cheaper class of building material.

Sandstone.—The sandstones of the State are found mainly in the central western part, near Springfield and Longmeadow, and extending northward as far as Northampton. Building stones of this nature are much more restricted in their distribution than the granitic rocks. Geologically, they are the northward continuation of the celebrated brownstone quarried at Portland and Middletown, in Connecticut. A zone of sandstones and shales occupies the valley of the Connecticut and traverses the entire State in a north and south direction, but phases which have an economic value are not persistent throughout this distance. In the town of Seakonk, Carboniferous sandstone is quarried which is locally used in the construction of foundations and for similar purposes.

Sandstone of several colors is quarried in the Longmeadow region, and there is a certain massiveness of the several types which is insuring its introduction in preference to the brownstones of other places. One of the most serious defects of ordinary sandstone, if we make an exception of the quartzites—such as the so-called Kettle River sandstone of Minnesota—is found in its planes of bedding. The more massive a given sandstone, or the less marked its bedding planes, other things being equal, the better the stone will withstand the rigors of temperate latitudes. The Longmeadow sandstone is not subject to this criticism of sandstones in general. Some of the earliest quarries of note in the State were opened in that region, but the special value of the stone seems to have become better appreciated in recent years.

Marble.—A belt of marble and limestone extends parallel to the sandstone, along the Housatonic Valley, in Berkshire County. The marbles range in composition from marbles proper to dolomites, or varieties carrying a high percentage of magnesium carbonate. There are also some few occurrences of marble in the eastern part of the State, but they have little commercial value.

The marbles and limestones of Massachusetts have a close resemblance to those in the Rutland Valley in Vermont, but they have deservedly failed to attain any such great development as they have to the north. They occupy a belt along the Housatonic Valley and are equivalent in age to those of Vermont. The white marble of Lee is a well-known stone, both at home and outside of New England. It is strictly a true dolomite, and as such has claims to durability not possessed by non-magnesian varieties. Near North Adams a thinly bedded marble is worked to a limited extent for the local demand.

Conglomerates.—Under the head of conglomerates are included those varieties of stream-transported and water-deposited rocks composed of various rounded rock fragments which occur as pebbles. Conglomerate represents one extreme, or coarse phase, of rocks of this type, while sandstone and slates represent the other, or fine phase. Situated immediately south and west of Boston, conglomerates occupy a considerable area and have been quarried very extensively for many years.

About 1840 the coarse conglomerates situated in Roxbury began to receive renewed attention as suitable for the cruder uses to which stone is put. Where the rock was traversed by a parallel set of joints it was found that the natural faces thus produced, especially when stained reds and yellows by iron oxides, were artistically adapted for building construction, and the better grades commenced to be used for this purpose, as well as for substructures of various kinds. Many of the finest churches in Boston and vicinity have been built of it and the stone has been shown to possess not only much beauty, but great endurance. It is still used for buildings of this class. The brilliantly stained joint blocks are sought for rustic masonry of all descriptions, and much of the beauty in work of this class in Boston's system of parkways is due to effects thus produced. At the present time the city of Boston operates many stone crushers which use this rock in the absence of a better road metal within the city limits. Few stones have superior artistic merit or are more enduring than this one.

Slate.—Roofing slate, as ordinarily defined and understood in the productive regions of Pennsylvania and Vermont, is not quarried in Massachusetts. The old workings in Lancaster have long since been abandoned as a source for roofing slate, with the exception of an occasional shipment about 1892. After lying unworked for many years the Lancaster slate quarries were reopened in 1885 for the manufacture of red glazed bricks. For various reasons this product is no longer manufactured, but the slate is still quarried and ground, and is said to be used for a body in asphalt paving, in the production of "electric cement," in the manufacture of oilcloth, and in certain lines of rubber goods and paint. At the present time slate is also quarried in the Somerville district, where it is known as bluestone. It does not, however, correspond either in character or in use to the roofing slates of Hydeville and Fairhaven, Vt., or to the bluestones of the Hudson River or Pennsylvania. It has a purely local use for the cheaper grades of work, such as cellar walls and underpinning. It has been used recently, but with ill success, in the construction of macadam roads, but as telford foundation it will no doubt prove fairly satisfactory.

Other Building Stones.—There are many other stones in the State which have a local or temporary value for one purpose or another. The flinty quartzite of the Pittsfield Valley is used to some extent in and about North Adams. A few miles south of that city the same rock is quarried and ground into powder of various sizes used in the manufacture of sandpaper, in the clay industry, for the manufacture of glass, etc. In the town of Lynnfield a dark serpentine has been recently exploited. Near Westfield there has been worked recently a deposit of verd antique marble which recent developments shows to be valuable. The Westfield deposit yields stone of various types. Prominent among these, and of great beauty, is the so-called "Serpentine marble," which is of a very dark green color mottled with patches of a lighter green serpentine and sparse random areas of a white mineral, probably in part calcite. This stone is in part a serpentinized, basic, igneous rock, no doubt originally a peridotite; the light green patches of serpentine are alterations from olivine, and the mass of the rock, forming a background for the light colored areas, is serpentinized enstatite. Large crystals of altered enstatite are revealed on the polished surface by a beautiful golden sheen often characteristic of this mineral, an effect adding not a little to the merits of the stone.

The next most desirable type is termed "Spangled marble," which consists of a ground mass of gray-green calcite thickly sprinkled with strongly contrasted, linear crystals and blotches of serpentinized actinolite, a stone closely corresponding to verd antique in composition; in fact, a finer-grained phase also found here is thus designated. While serpentines are quarried elsewhere in this country and in Europe, none resemble the dark variety from Westfield. A ready market should be found for this material, particularly for all forms of interior work. The stone is now being used for flooring purposes in the post offices at Washington, D. C., and Milwaukee, Wis.

Several churches and many private residences have been built wholly or in part of small bowlders, used entire, gathered from vacant lots. The increased consumption of these hitherto objectionable stones, brought about by their use in building construction and as road metal, has served to lessen the supply very materially. In some places they have come to have considerable value, and when of good quality are easily sold for 50 to 75c. per ton in localities where a better class of road material is not conveniently attainable. In Duke's County, on the island of Martha's Vineyard, the large bowlders have become of especial value.

Besides the varied types of stones produced in this State and largely consumed locally, or exported, there are many others which will come into the market when trade conditions are ripe. Among these the brecciated felsites and porphyries of Eastern Massachusetts deserve to be prominently mentioned. In ancient times these stones, expensive to work, but possessing rare and unusual beauty, were eagerly sought after for mosaic and other interior work. The time is not far distant when the wealthy classes in this country will seek for material of this kind, regardless of expense, for the decoration of their homes, and no doubt such stones will also be used in many public buildings where artistic merit is not entirely subservient to utility.

Road Metal.—The enactment of a State road law and a liberal annual appropriation for this purpose is proving of great benefit to the commonwealth in many ways. The numerous dikes and lava flows occurring in the State have taken on a new interest and value. Such stones, with the possible exception of the Somerville diabase, which has long been known under the common name of black granite, have been hitherto but little exploited. With the advent of the road law, fine-grained, basic rocks, like diabase and diorite, began to be sought after and soon assumed a recognized value. There are now some half-dozen companies in the field scattered over the State which make a regular business of furnishing broken stone. To these should be added many plants operated by cities and towns and by the Highway Commission itself. It is worthy of note, as illustrating the condition of public opinion regarding the need of good roads in Massachusetts, that Boston alone, in 1898, consumed 170,529 tons of macadam and 6,100 tons of telford.

The commonwealth possesses many high-grade road metals, and with the exception of Duke's, Barnstable, and Berkshire Counties, they are well distributed for economic use. The largest area of trap (diabase) within its borders stretches across the State, inclosed in the belt of sandstones and shales occupying the Connecticut Valley. This belt has been exploited at several points. In the eastern part of the State great numbers of dikes and bosses suitable for this purpose are being extensively quarried. In regions not having any outcropping rocks adapted for road building, recourse is had to various sources of supply. Where a seashore town possesses a coarse, pebbly beach a judicious selection of these is turned to account; in others the coarse gravels or morainic deposits afford good material. Reference has thus far been made only to the road stones adapted to the needs of macadam; the stones suitable for building telford are even more widely spread and available.

Those concerns operating stone crushers, which use high-grade trap, are increasing their output by the demand for broken stone for railroad ballast and cement foundations of all kinds; for a body in Portland cement floors, sidewalks, and granolithic pavements; and screenings are displacing gravel somewhat for roofing purposes in connection with tar and asphaltic mixtures.

The economic importance of using the highest grade of road metal has led to the establishment of a State bureau, which determines the value of all stones submitted to it by a system of tests, the accuracy of which is usually borne out by practical trial upon the roadways. The properties commonly ascertained are toughness, as determined by the abrasive test; cementation and recementation values. The materials that approach the qualities of an accepted standard are dark-blue, fine-grained, basic rocks composed of feldspar and one or both of the common minerals, hornblende or pyroxene. Such rocks are usually free from the brittle mineral quartz, are very tough and readily cement together, under the action of the steam roller, to a mass having the maximum wearing surface and the greatest number of desirable qualities. The consumption of road metal has assumed large proportions and no doubt will increase until the principal highways of the State are macadamized.

SULPHUR AND PYRITES.

Sulphur was mined in the United States in 1898 in Utah, Nevada and Louisiana, the total production being 2,726 long tons, valued at $59,754, against 1,690 long tons, valued at $34,814, in 1897. The production of sulphur in Louisiana in 1898 amounted to 1,319 long tons. Nothing was done at the Louisiana mines after May. These mines were worked during the first six months of 1897 by the Frasch process (of melting by superheated water in pipes), as described in The Mineral Industry, Vol. V., p. 514. The production during this semester was 690 long tons; the production in 1898 brings the total since the first application of this process, in 1895, up to 4,509 long tons. In 1898 the conclusion was reached that the exploitation of the mines by the Frasch process was unprofitable. It is now conceded that this process is a complete commercial failure, and has greatly injured these deposits of sulphur, if it has not actually destroyed their value entirely, since their exploitation by shafts and regular underground mining is now fraught with danger, owing to the uncertain extent of the openings from which the sulphur has been dissolved by the Frasch experiments. In view of the heavy flow of water which may be expected through certain strata, this uncertainty is an element of great peril.

Some exploration and development work was done in 1898 in the sulphur deposits of Texas, especially on those near Guadalupe. The recent prospecting at this place has shown that a very considerable area is underlain by the mineral, but lack of experience as to the probable yield prevents any trustworthy expression of opinion as to the amount that may be derived from these deposits. It is expected that they will become productive in 1899. A new deposit was located on the western rim of the Salton desert, at the fork of the old Yuma and San Diego stage roads, in San Diego County, Cal. Some mineral was dug and piled up, but the owner stated that no shipments would be made until 1899. A deposit of sulphur was also developed near Whittier, Cal., and the discovery of a deposit in the Mojave desert, about 50 miles from Flowing Wells, on the Southern Pacific Railway, was reported. A new discovery of sulphur was reported on Unalaska, an island on the route from Seattle to Alaska, about 1,750 miles from the former place. These deposits are to be developed during 1899.

Prices.—The Sicilian sulphur trust maintained prices during 1898 upon a basis profitable to itself. Best unmixed seconds sold in New York from January to the end of March at $21·44@$22 per ton. The Spanish-American war caused a rise in values, brimstone being declared contraband, and in May a small quantity sold at $37·50. At this time some shipments were made to the United States via Montreal. In June prices were still high, one seller quoting $40, but with the destruction of Cervera's fleet there was a rapid fall, and in July supplies were plenty at $21@$21·50 for best unmixed seconds and $19 and a fraction for thirds. During the remainder of the year seconds ranged from $20 to $23 and thirds from $18 to $20.

PRICE OF BRIMSTONE PER LONG TON IN NEW YORK.

Month.	1897.		1898.		Month.	1897.		1898.		Month.	1897.		1898.	
	Seconds.	Thirds.	Seconds.	Thirds.		Seconds.	Thirds.	Seconds.	Thirds.		Seconds.	Thirds.	Seconds.	Thirds.
January..	$21·00	$20·10	$21·44	$20·56	May.......	$19·75	$19·12	(a)	......	September	$21·00	$199·2	$20·56	$1[illegible]
February	20·55	19·65	21·50	20·38	June......	20·00	19·50	(a)	......	October...	21·75	207·5	22·40	19·70
March ...	20·10	19·20	22·00	20·25	July.......	20·50	19·75	21·40	$19·60	November.	20·75	197·5	21·[illegible]	19·81
April.....	19·87	19·12	26·20	24·50	August....	20·50	19·62	20·69	19·12	December.	21·50	205·0	[illegible]	19·[illegible]

(a) The quotations for May and June are not given here as they would vitiate the averages for the year. Average for 1896: Seconds, $19·00; thirds, $18·30. Average for 1897: Seconds, $20·60; thirds, $19·04. Average for 1898: Seconds, $21·92; thirds, $20·14.

SULPHUR PRODUCTION, IMPORTS, AND CONSUMPTION IN THE UNITED STATES.

Year.	Production.			Imports.								Consumption.	
	Sulphur.			Crude.		Flowers of Sulphur.		Refined.		Totals.			
	Long Tons.	Value.	Value per Ton	Long Tons.	Value.	Long Tons.	Value.	Long Tons.	Value	Long Tons.	Value.	Long Tons.	Value.
1892	1,630	$54,750	$33·00	100,938	$2,189,481	158·00	$5,439	26·00	$4,106	101,122	$2,199,026	102,752	$2,253,773
1893	1,200	26,880	22·50	107,601	1,903,191	240·96	5,746	41·73	1,017	107,885	1,909,954	109,020	1,935,474
1894	441	7,056	16·00	124,467	1,734,643	165·00	4,145	41·00	1,207	124,673	1,739,641	125,114	1,747,405
1895	1,650	26,400	16·00	125,950	1,593,148	581·00	12,888	229·00	4,378	126,760	1,610,414	128,410	1,636,814
1896	3,800	72,200	19·00	145,318	2,085,076	665·00	13,266	447·00	8,226	146,430	2,106,568	150,230	2,178,768
1897	1,690	34,814	20·60	138,846	2,442,240	319·00	7,950	148·00	3,387	139,313	2,453,577	141,003	2,488,391
1898	2,726	59,754	21·92	159,790	3,081,974	507·00	14,548	163·00	4,391	160,460	3,100,913	163,186	3,160,667

WORLD'S PRODUCTION OF SULPHUR. (a) (IN METRIC TONS.)

Year.	Austria	France. (c)	Hungary	Germany	Greece.	Italy. (b)	Japan.	Russia.	Spain.	Sweden	United States.
1893........	44	3,733	70	2,161	2,400	417,671	23,930	403·0	4,686	75	1,219
1894........	76	851	93	2,168	1,946	405,781	18,787	589·0	3,417	36	448
1895........	(d)	4,213	102	2,061	1,480	370,766	15,557	190·0	2,231	(d)	1,676
1896........	(d)	9,720	138	2,263	1,540	426,353	12,540	437·0	1,800	77	3,861
1897........	(d)	10,700	112	2,317	358	496,658	(e)	(e)	(b)3,500	(d)	1,717

(a) From the official reports of the respective governments. The sulphur recovered as a by-product by the Chance-Claus process in the United Kingdom, amounting to about 31,000 long tons in 1898 is not included. (b) Crude. (c) Raw mineral; limestone impregnated with sulphur. (d) *Nil.* (e) Statistics not yet published.

Hawaii.—Efflorescences and in some places considerable beds of sulphur, deposited from sulphurous vapors and waters, occur in the crater floors and sides of Kilauea and Mauna Loa, but are not utilized to any extent.

Italy.—The sulphur shipments from Sicily showed another large increase in 1898, but the actual production was not so large since stocks were drawn upon heavily. The Anglo-Sicilian Sulphur Co., Ltd., continues to control this industry and seems to have the business well in hand. This company reported its receipts of sulphur in Sicily from August 1, 1897, to July 31, 1898, at 275,110 long tons. During the same period it sold 284,960 tons of crude and refined sulphur at an average gross profit of 9s. 10d. per ton. In England it received 31,500 tons of Chance-Claus sulphur, and delivered 31,534 tons, at an average gross profit of 5s. 4d. per ton. The contract with the Catania refiners proved fairly satisfactory and helped the company to keep control of the market in Sicily. The gross profit for the year was £143,941, to which was added £5,880 for interest on deposits, etc. After deducting various amounts for current expenses in London and Sicily, depreciation in valuation, unforeseen contingencies, etc., there remained £90,867 10s. 6d. for distribution in dividends and application to the reserve fund. According to this company the production of sulphur in Sicily during the year ended July 31, 1897, was 421,704 tons; during the year ended July 31, 1898, 439,879 tons; while the exports increased from 413,081 tons to 450,287 tons. The increased demand for sulphur is attributed largely to the greater requirements of the wood pulp industry.

SHIPMENTS OF SULPHUR FROM SICILY TO THE UNITED STATES. (IN LONG TONS.)

Port.	1892.		1893.		1894.		1895.		1896.		1897.		1898.	
	Seconds.	Thirds.	Seconds.	Thirds.	Seconds.	Thirds.	Seconds.	Thirds.	Seconds.	Thirds.	Seconds.	Thirds.	Seconds.	Thirds.
New York	34,390	14,700	29,146	14,250	33,150	13,725	35,888	19,975	50,557	17,796	57,174	13,300	49,614	22,475
Charleston	4,010	500	11,665	1,860	3,273	12,023	700	8,450	2,330	5,370	1,500	3,630	500	1,600
Philadelphia	3,600	6,800	1,900	6,260	350	5,050	1,200	7,150	500	5,500	199	5,210	1,200	5,400
Baltimore	900	11,455	2,050	7,900	600	14,700	1,100	8,620	3,650	10,500	3,798	10,033	2,350	12,015
Boston	1,825	1,500	500	......	1,017	3,300	2,350	2,600	4,600	700	7,220	1,000	4,500	1,550
Savannah	600	570	3,450	1,880	5,695	4,100	3,784	800	8,370	1,025	4,700	......	1,980	......
Wilmington, N. C.	......	......	......	1,140	......	1,890	......	650	1,260	1,400	......	1,550	500	1,200
New Orleans	......	......	1,900	......	2,400	......	1,700	......	2,100	......	3,340	......	500	2,000
Other ports (*a*)	4,000	......	......	......	800	3,700	1,880	2,380	7,975	1,290	4,883	600	18,159	200
Totals	49.325	35,525	50,611	33,290	47,285	58,488	48,602	50,625	81,342	43,581	82,814	35,323	79,303	46,440

(*a*) Norfolk, Mobile, San Francisco, Portland, Me., and Portland, Ore.

TOTAL EXPORTS OF SULPHUR FROM SICILY SINCE 1890. (*a*) (IN LONG TONS.)

Country.	1890.	1891.	1892.	1893.	1894.	1895.	1896.	1897.	1898.
United States	106,656	97,520	84,450	83,901	105,773	99,227	124,923	118,137	138,435
France	71,790	56,168	73,176	89,736	56,932	69,696	76,739	84,895	88,657
Italy	40,231	42,212	38,711	54,486	49,895	49,349	54,009	73,052	62,652
United Kingdom	26,213	23,408	24,853	27,453	22,165	24,043	21,913	24,520	26,983
Greece and Turkey	18,103	11,414	*b*14,845	*b*13,840	*b*16,870	*b*16,195	*b*18,556	*b*13,866	24,808
Portugal	16,695	11,439	13,490	14,545	8,670	14,562	12,001	7,054	8,257
Russia	17,158	11,930	14,178	19,730	17,977	17,962	18,752	17,532	12,285
Germany	15,703	10,629	14,326	16,259	16,437	15,472	15,680	19,721	27,048
Austria	8,746	10,575	9,096	10,169	11,494	12,170	13,799	15,993	15,796
Turkey	4,231	3,000	(*b*)	(*b*)	(*b*)	(*b*)	(*b*)	(*b*)	(*b*)
Spain	5,679	3,845	7,382	3,499	3,445	5,753	5,910	4,039	3,233
Belgium	7,279	5,089	5,133	4,358	5,644	6,410	7,527	9,253	8,402
Holland	....	....	2,183	2,957	2,365	3,335	3,834	3,599	5,646
Sweden and Norway	3,314	2,252	4,561	6,579	7,887	5,730	14,540	11,226	12,331
South America	....	....	....	....	....	....	....	....	....
Australia	....	....	1,200	....	....	....	....	....	....
Denmark	400	300	(*c*)	(*c*)	(*c*)	(*c*)	(*c*)	(*c*)	(*c*)
Other countries	2,565	3,542	3,152	1,680	3,376	7,732	8,563	7,651	12,791
Totals	344,763	293,323	309,536	349,192	328,930	347,636	396,745	410,538	447,324
Stock in Sicily at end of year	106,770	117,037	175,299	210,665	198,513	203,756	222,999	240,367	248,023

(*a*) From report of A. S. Malcomson, New York. (*b*) Exports to Greece and Turkey combined after 1892. (*c*) Included in exports to Sweden.

Japan.—According to a British consular report the exportation of sulphur from Hakodate, island of Hokkaido, in 1897 amounted to 6,723 long tons, valued at £24,386, against 7,992 (£22,649) in 1896. There is a small domestic demand for the manufacture of sulphuric acid, which is estimated at 2,000 to 3,000 tons per annum. This demand is increasing steadily. The exports in 1897 were largely from stocks, which accumulated to the extent of 6,500 tons during 1896 and 1897. These stocks had not yet been exhausted, and it was estimated that the shipments in 1898 would be between 8,000 and 9,000 tons, after which the exportation is expected to drop to 3,000 tons per annum.

With one insignificant exception all the sulphur mines of Hokkaido and the northern mainland of Japan are volcanic deposits. The principal mines are Osoresan, in Rikuoku, Nambu, north mainland; Atosanodori, in Kushiro, Hokkaido; Sharesan, in Kitami, Hokkaido. The only mine of any importance now being worked is that called Ponto, situated in Kunashiri, one of the Kurile islands, on the northeast coast of Hokkaido, from which about 2,000 or 3,000 tons are produced annually.

Mexico.—A small amount of sulphur was produced in Lower California in 1898 by the Mexican Sulphur Co., the product being shipped to California via Yuma, Ariz. This company did considerable exploration work on its property, whereby between 30,000 and 40,000 tons of sulphur is said to have been exposed. The exploitation of these mines is made difficult by the lack of transportation facilities. The company has surveyed a railway route to Yuma, and also a line from the mines to San Diego Bay, Cal., the distance of the former being 67 and of the latter 170 miles. As yet no decision has been reached as to what will be done. All of the mineral that has been shipped thus far has been carried in wagons to a point on the Hardy's Colorado River, 11 miles from the mines. At that point it has been loaded on barges, which have been towed to the mouth of the Hardy's, and from there up the Colorado River to Yuma, whence it has been shipped to California.

Russia.—A company has been formed in Moscow to work the sulphur beds of Czarkowski. The sulphur is to be separated from the gangue by the Patkonoff process, in which the ore is heated by steam in a closed vessel and the melted sulphur drawn off through a sieve.

Venezuela.—About 15 km. S.S.E. of Carupano there are said to be large deposits of sulphur at an elevation of 300 meters above sea-level, which have remained unexplored until recently on account of the high cost of transportation to the seaboard, packing by mules being the only means. There are five large deposits, not covered by overgrowth, presenting a surface area of about 300,000 sq. meters, besides a number of smaller deposits which are concealed from observation. So far as tested these beds extend to considerable depth. Five samples of the mineral tested in London gave an average of 62·5% S, and it is estimated that the cost of the mineral on board ship will not exceed $4 per ton, and that it can be laid down in New York at $6 per ton. The Venezuelan Trading and Investment Co., of New York, which controls these deposits, does not expect to export sulphur before the middle of 1899, or until proper shipping facilities can be provided. The present plan is to build a wire tramway or a surface narrow-gauge railroad from the mines to the port of Carupano, which is capable of accommodating vessels of the largest class.

It is probable that a breakwater and wharf combined will be built at the port, so that vessels can lie alongside when loading from "pockets," capable of holding about 2,000 tons each. The Venezuelan government has decided that no wire tramway shall be built to the river, 10 km. distant, as originally contemplated, and insists that the sulphur must be shipped from Carupano.

PYRITES.

Production.—The production of pyrites in the United States in 1898 showed a large increase over 1897, especially on the part of the Virginia mines. The Davis Sulphur Ore Co., of Massachusetts, a large producer, also increased its output, but not so much proportionately as did the various producers in Virginia. New pyrites mines were exploited in Tennessee, New York and California, a considerable production being reported from each of these States. In New York the old Stella mine, near Herman, was reopened, and was productive during the latter half of the year. It affords an ore averaging about 35% S as mined, which can be burned down to less than 1%. This mine is expected to be a large producer in 1899. The total production of pyrites in the United States in 1898 was 191,160 long tons, valued at $589,329, of which 132,400 long tons, valued at $352,093, was mined in Virginia and North Carolina; the production in 1897 was 133,368 long tons, valued at $404,699, of which 89,328 long tons, valued at $246,155, was mined in Virginia.

Increased Demand.—The increase in the production of pyrites has come about largely through the prevailing high prices for brimstone, and a further increase is looked for, not only on the part of the sulphuric acid makers, to supply the requirements of the fertilizer, oil and general chemical trade, but also from the sulphite process paper makers, who heretofore have been limited practically to the use of brimstone. They have been large consumers of the latter mineral. Probably on an average 160 lb. of sulphur are required per 1,000 lb. of sulphite pulp. The high price for brimstone led some of the important paper manufacturers to investigate last year the possibility of using liquid sulphurous acid, to be made from pyrites.

On the present sale basis per unit of sulphur, American pyrites is sold at an average of about 7½c. per unit at the mines, and the freight rates to points of consumption average 3½ 4c. per unit. At these prices foreign ores, which are sold on the Atlantic coast at 11@13c. per unit of sulphur, cannot materially increase their consumption. With increased demand, there is every probability of stable prices, with some prospects of an increase in prices at inland points. Summing up the trade situation, 1898 was a very satisfactory year, while 1899 promises to be the best one ever known since this industry was established.

Prices.—American lump ore (basis 42%) sold in 1898 at $3·25 per long ton f. o. b. at mines in Virginia, and at $5@$5·25 f. o. b. at mines in Massachusetts; fines sold at $2·75@$3 and at $4·25, respectively. These ores contain an average of 42 to 44% sulphur. Pilley's Island pyrites, which contains over 50% sulphur, sold at $6·50 and $4·50 for lump and fines, respectively, delivered in New York. The domestic acid makers obtained their usual large supplies from the copper-bearing pyrites imported from Canada and Spain, the price of the latter ranging from 10 to 13c. per unit, long tons, according to the percentage in sulphur, delivered ex-ship at New York or other coast ports.

PYRITES STATISTICS OF THE UNITED STATES. (*a*)
(In tons of 2,240 lb.)

Year.	Production.		Imports.			Consumption.	
1894	107,462	$466,466	164,000	45%	590,905	253,485	$1,057,371
1895	107,371	342,587	190,436	46%	673,812	297,787	1,016,669
1896	109,282	292,626	199,678	47%	1,140,571	308,960	1,433,197
1897	133,368	404,699	259,546	47%	847,419	392,914	1,252,118
1898	191,160	589,329	171,870	47%	544,165	363,030	1,133,494

a These statistics do not include the auriferous pyrites used for the manufacture of sulphuric acid in Colorado.

Virginia.—(By W. H. Adams): "The most important developments during 1898 were in the South. On the Virginia mineral belt the year was an unusually active one. The Sulphur Mines Co. and the Arminius Chemical Co. exceeded all records heretofore made for shipment of pyrites, and a new mine (that of the Pyrites Mining and Chemical Co.) was added to the list of producers, which property lies between the lands of the two companies mentioned. The shipments made by these three mines in 1899 will exceed those of 1898, from present indications, by at least 40,000 tons. There were no new developments on the mineral belt, although there are rumors that several properties will be prospected in 1899, and many options have been taken of late on lands to the northeast of this section.

"A notable change from the established methods of mining pyrites in this district will shortly be inaugurated at the Arminius mines, where a cableway nearly 500 ft. in length is about completed and ready for the excavation and removal of over 1,000,000 tons of overburden, iron ore and pyrites from the main deposit of the company. The hoisting and removal of this enormous tonnage is to be done by electrically driven motors of new design, the successful installation of which promises to revolutionize the handling of heavy materials of this character under the peculiar existing conditions."

WORLD'S PRODUCTION OF PYRITES. (*a*) (IN METRIC TONS.)

Year.	Belgium.	Bosnia.	Canada.	France.	Germany.	Hungary.	Italy.	Newfoundland.
1893	6,301	1	53,109	231,025	121,329	68,180	29,460	35,495
1894	3,050	....	36,766	283,439	134,787	76,870	22,638	41,438
1895	3,510	....	31,024	253,416(b)	127,036	69,195	38,586	34,879
1896	2,560	2,000	30,586	282,064	129,168	52,697	45,728	27,712
1897	1,828	3,670	35,299	303,448	133,302	44,454	58,320	33,316

Year.	Norway.	Russia.	Spain.	Sweden.	United Kingdom.	United States.	Totals
1893	53,754	16,271	220,000	480	16,090	95,000	949,504
1894	70,850	19,901	60,000	656	15,771	107,462	873,535
1895	61,994	11,042	60,267	221	9,193	107,371	807,732
1896	60,507	13,200	100,000	1,009	10,178	117,782	875,191
1897	(*c*) 70,552	(*d*)	217,605	517	10,753	128,468	1,041,432

(*a*) From the official reports of the respective governments except the figures for Spain in 1897, which are taken from C. Le Neve Foster's report. (*b*) Includes also auriferous pyrites. (*c*) Exports. (*d*) Statistics not yet published.

Newfoundland.—The Pilley's Island mine produced 32,479 long tons of pyrites (averaging 51·5% S) in 1898, against 30,900 in 1897, the entire product each year being shipped to the United States. Of the shipments in 1898, 25,958 tons were lump ore, averaging 51·86% sulphur, and 6,521 tons fines, averaging 50·70%. One cargo of 1,000 tons of lump ore assayed 52·34% sulphur, or within less than 1% of the tenor of pure pyrites. During 1898 Messrs. B. G. Reid & Sons, of Montreal, the Newfoundland railway magnates, opened a promising deposit of pyrites at Bay of Islands, where about 100 tons were mined. The lode is said to average 40 to 50 ft. in width, the ore being of similar quality to that of Pilley's Island. The Newfoundland Pyrites Co., with a capital of $1,500,000, was organized to exploit these mines, working in connection with another new company, the Newfoundland Bleached Pulp Co.

Hungary.—The pyrites mines at Schmöllnitz have three main lenses of ore, the largest being 1,200 ft. in length, 120 ft. wide, and 420 ft. deep. The ore contains 48% sulphur and about 46% iron. After burning, the residues, which contain 1 to 3% Cu, are leached for their copper contents, and the final product, assaying 60 to 65% Fe, is sold to iron smelters.

Spain.—According to Señor Adriano Contreras, in the annual statistical number of the *Revista Minera*, the exports of iron pyrites from Spain in 1898 amounted to 260,076 metric tons, an increase of 42,471 over 1897, almost the whole amount each year being derived from Huelva. The Rio Tinto Copper Co. in 1898 produced 70,814 tons of pyrites carrying no copper, the first production of this kind reported by the company. The Aguas Tenidas Co., formerly a large producer, made no report for 1898.

SULPHURIC AND SULPHUROUS ACIDS.

The manufacture of sulphuric acid is the most important branch of the chemical industry in the United States. If the entire amount of brimstone and pyrites used in the United States were converted into sulphuric acid, it would have been equivalent to a make of 1,144,968 short tons of acid of 66° B in 1897 against 1,019,501 in 1896; this calculation has not been made for 1898. As a matter of fact a large amount of the domestic consumption of brimstone and pyrites, especially the former, is converted into sulphurous acid for use directly in the manufacture of sulphite pulp for the production of paper. No accurate statistics as to the amount used for this purpose or as to the actual make of sulphuric acid are available, but there is no doubt that the more part of the domestic production of sulphuric acid at the present time is derived from pyrites, partly mined in the United States and partly imported from abroad. The chief uses for sulphuric acid are in the manufacture of acid phosphate of lime and the refining of petroleum. It takes one pound of sulphuric acid of 66° B to produce a gallon of kerosene and it requires 850 to 1000 lb. of acid of 50° B to decompose 1000 lb. of phosphate rock. It is estimated that about 90% of the total make of sulphuric acid in the United States is consumed for these purposes. The total amount of sulphur consumed in the form of pyrites in the United States in 1898 is estimated at 164,889 long tons against 180,668 in 1897, while the consumption of sulphur in brimstone was 150,446 long tons in 1898 against 137,725 in 1897. The statistics of sulphur consumption in the United States have been summarized as follows:

	1896.	1897.	1898.
Domestic production of sulphur	3,800	1.690	2,726
Imports of brimstone	145,318	138,846	150,790
Total consumption	149,118	140,536	153,516
Sulphur contents at 98%	146,136	137,725	150,446
Domestic production of pyrites	109,282	133,368	191,160
Imports of pyrites	199,678	259,546	171,870
Sulphur in domestic pyrites at 44%	48,084	58,683	84,110
Sulphur in foreign pyrites at 47%	93,849	121.986	80,779
Total sulphur consumed in pyrites	141,933	180,668	164,889
Total sulphur consumed in brimstone	146,136	137,725	150,446
Grand total	288,069	318,393	315,335

In addition to the above there is a considerable amount of sulphuric acid recovered as a by-product in roasting zinc blende, copper pyrites and auriferous iron pyrites, especially the first. The production of sulphuric acid as a by-product amounted to 47,558 short tons in 1898 against 42,352 in 1897 and 27,512 in 1896, these figures being reduced to a common basis of 66° B. As a matter of fact, by far the largest part of this production was marketed in the form of chamber acid, wherefore the actual tonnage handled was very much larger. The average price of sulphuric acid, concentrated and chamber, at New York in 1897 and 1898, is shown in the following table:

Month.	Conc. Acid, 66° B. Per 100 lb.		Chamber Acid, 50°. Per Ton.		Month.	Conc. Acid, 66° B. Per 100 lb.		Chamber Acid, 50°. Per Ton.	
	1897.	1898.	1897.	1898.		1897.	1898.	1897.	1898.
January	$0·85·00	$1·00	$6·00	$7·25	July	$0·85·00	$1·10	$6·00	$11·55
February	0·85·00	1·00	6·00	6·50	August	0·85·00	1·10	6·00	11·00
March	0·85·00	1·00	6·00	6·50	September	0·73·75	1·10	6·00	11·50
April	0·85·00	1·02	6·00	7·50	October	0·70·00	1·10	6·00	11·50
May	0·85·00	1·10	6·00	11·50	November	0·70·00	1·10	6·00	11·50
June	0·85·00	1·10	6·00	11·50	December	0·97·50	1·10	7·50	11·50

The average price of 66° B. acid in 1897 was 82·6c., and $1·07 in 1898; 50° chamber acid averaged $6·13 per ton in 1897, and $9·98 in 1898.

Sulphuric acid is made in California by six works which reported a production in 1898 of 38,282 short tons, of which 28,200 was made from brimstone imported chiefly from Japan, 7,483 from pyrites, while 1,600 was recovered as a by-product, all these figures being reduced to a basis of 66° B.

Liquid sulphurous acid is made in the United States by one concern, namely the Somerset Chemical Co., of Bound Brook, N. J., which employs the process of Haenisch & Schroeder. Its production in 1898 was not large, the industry being still in its infancy, but an increased output is expected to be made in 1899. The process consists essentially in passing the sulphurous gases through long flues in order to cool them, after which they enter an absorbing tower at the bottom, and rise against a spray of water in which the sulphurous anhydride is absorbed, the gas discharged from the top of the tower being almost free from sulphur. The aqueous solution of sulphurous acid is drawn off to a special apparatus in which it is heated to boiling, thereby giving off pure sulphurous acid, mixed with steam, which is cooled in surface condensers, separated from the condensed water and liquified.

THE MANUFACTURE OF CHAMBER SULPHURIC ACID.*

BY F. J. FALDING

THE problem which confronts the intending manufacturer of sulphuric acid is a complicated one. It is complicated for the manufacturer who proposes merely to make sulphuric acid as a raw material in an industry already established, the location of which is fixed either *de facto* or from considerations paramount to the cost of sulphuric acid, such as fertilizer manufacture or a metallurgical process. It is doubly complicated for the manufacturer who expects to dispose of his product in the general market.

In the manufacture of sulphuric acid the disproportion between the values of raw material, labor, freights, and manufactured products are marked and unusual. The cost of raw material will average 80%, and over, of the total cost of 50° B. acid, while labor and maintenance of plant will rarely average over 17%. The proportion differs somewhat in the stronger acids, but not materially. The freights on raw materials from the various sources of supply to the various centers of manufacture will vary from 10% to more than 100% of their cost at point of production; and the rate of freight on manufactured acid also bears a high ratio to its cost of production.

The dominant feature in the manufacture of sulphuric acid is the high proportion of cost of raw material to labor, which indicates the necessity for the utmost care in its selection, transportation and handling at the works, and of economy in its utilization during the manufacturing process. On the other hand, the relatively small proportion of the cost of labor indicates that it should be of a high quality and competent to secure economy in the handling and working up of the raw materials. A saving of 10% on the cost of raw material would average over 8% on the cost of the acid produced, while 10% on the labor cost would never exceed 2% on the cost of the acid. One of the most common mistakes made by acid makers is the want of appreciation of this condition. It is no uncommon thing to see a 2% saving on labor cost secured at the expense of an increased cost on manufactured acid of 10 to 25% or more, to say nothing of depreciation of value of plant, which is at the mercy of the men in charge of it, perhaps to a larger extent than any other form of manufacturing plant.

An endeavor to elucidate all such matters which bear upon the practical and economical production of sulphuric acid is the object of this paper. There is no intention to discuss the scientific features or theories of the process, which have been already ably treated in technical literature. Nor is such knowledge necessary to enable a manufacturer to obtain a thorough grasp of his business, nor to secure desirable financial results. This knowledge should be possessed by the trained chemist and acid maker, and can be purchased by the manufacturer as a necessary part of his manufacturing equipment.

THE RAW MATERIALS, THEIR RELATIVE VALUE AND PRELIMINARY TREATMENT.

I. *Sulphurous Acid Gas* (SO_2): (*a*) obtained as a waste product at many metallurgical works; (*b*) from sulphide ores, the chief or only value of which is

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* From 50° B=105° Tw.=1·526 sp. gr. to 60° B=141° Tw.=1·706 sp. gr.

the sulphur; (*c*) from sulphide ores containing valuable metals in which the sulphur is of secondary importance; (*d*) from brimstone; (*e*) from spent oxides from gas works. II. *Nitrous Acid Gas*[*] (N_2O_3): (*a*) from nitrate of soda; (*b*) from nitric acid; (*c*) from nitrous vitriol, including nitrous acid gas recovered from various chemical processes and the spent acid from manufacturers of high explosives. III. *Steam;* (*a*) from coal, coke or wood; (*b*) gas, natural or artificial; (*c*) petroleum; (*d*) water; (*e*) exhaust steam.

Dilute sulphuric acid (SO_2+O+H_2O, or H_2SO_4) is formed by the oxidation of sulphurous acid gas (SO_2) to sulphuric anhydride (SO_3) by means of an oxidizing agent, *i.e.*, nitrous acid (N_2O_3), in the presence of water. The sulphurous acid gas is produced in dilute form by the combustion of one or more of the above-named raw materials, and is passed into a series of acid-resisting apparatus, where in combination with nitrous acid gas, produced by the decomposition of one of the above-named raw materials, and water, usually supplied as steam, it is finally, after various chemical reactions, condensed into more or less dilute sulphuric acid, the oxidizing agent, nitrous acid gas, being more or less completely recovered.

In actual practice, and with reasonably good management, and a modern plant, the following approximate proportions of raw material will be required to produce 2,000 lb. of 50° B. sulphuric acid: Sulphurous acid as actual sulphur, 420 lb.; nitrous acid as nitrate of soda, 13 lb.; steam as water, 800 lb.; and as good steam coal, 80 lb.; in addition to which a further amount of steam will be required as power. It must be noted that the amount of sulphur is not the gross amount of sulphur contained in the raw material, but only that which actually enters the chambers as sulphurous acid gas; also that conditions as regards steam will vary, and other fuels will be used locally in preference to coal. The raw materials available in the United States and Canada at the present time are summarized in the paragraphs immediately following.

I. SULPHUROUS ACID GAS.—There is hardly a locality in the northern part of of the continent which is likely at any time to become a manufacturing center where a supply of sulphur-bearing material is not known to exist, in some form or other, within a reasonable distance, but since few of these are developed, either as to the deposits themselves or as to their transportation facilities, it is hardly worth while considering any except such as are immediately available.

(*a*) *Waste Sulphurous Acid Gas.*—The gas from the roasting of sulphide ores at many metallurgical works can be made, by a modification of the roasting furnaces, available for the manufacture of sulphuric acid, but it is now generally wasted, except at the works of the Matthiessen & Hegeler Zinc Co., the Illinois Zinc Co., the Arizona Copper Co., and the Western Chemical Co.

(*b*) *Sulphide Ores the Chief or Only Value of which is the Sulphur.*—The non-cupriferous Spanish, Portuguese and Norwegian ores, the Newfoundland ore, the Davis ore from Charlemont, Mass., the ore from the various mines in Louisa and Prince William counties in Virginia, from the Charlotte mine in South

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[*] It is true that part of the niter is supplied to the system as nitric acid and NO, but as N_2O_3 undoubtedly forms the active nitrogen oxide in the process, it has been assumed as the raw material. The part played by nitrogen oxides in the formation of sulphuric acid is very complicated and has been discussed by Lunge in a masterly way in the second edition of *Sulphuric Acid and Alkali*, to which reference should be made.

Carolina, and the Stella mine in St. Lawrence County, New York, are of this type.*

(*c*) *Sulphide Ores Containing Valuable Metals in which the Sulphur is of Secondary Importance.*—The cupriferous Spanish, Portuguese, and Norwegian ores, containing also gold and silver; the Eustis Canadian ore, containing copper, gold, and silver; concentrates from many gold mines; cupriferous gold and silver-bearing ores in Colorado, Montana, and the Pacific Coast; the zinc blendes of Missouri, Kansas, and Wisconsin; the mixed copper, zinc, and lead sulphides of Maine and New Hampshire; and the copper-nickel pyrrhotite of Sudbury, Ont.; are all of this type. As regards the copper-bearing sulphides, they generally contain sufficient sulphur to burn without additional fuel. The blendes, pyrrhotites, and some concentrates must be burnt in muffle furnaces or with auxiliary fuel.†

The only constituent of the ores of classes *a* and *b* which concerns the sulphuric acid manufacturer is the sulphur, and he is interested only in that part of the sulphur contents which by reasonable care he can expect to obtain as sulphurous acid gas in his chambers; or, in other words, the available sulphur. The following classification of ores by available sulphur contents will be found convenient, but this is not intended to be precise, since, according to the care exercised in the selection at the mine of some of the ores, they will yield 1 or 2% more or less available sulphur than the mean here given. (1) Ores containing 48% available sulphur: The Spanish, Portuguese, some of the Norwegian, and the Newfoundland ores; (2) ores containing 38% available sulphur: The Massachusetts, Virginia, New York, Eustis, and most of the Western cupriferous ores, and some concentrates; (3) ores containing less than 35% available sulphur: All the zinc and zinc-lead ores and most concentrates; these ores require auxiliary fuel or enrichment with high-grade pyrites.

When ores contain, among others of minor importance, the following minerals, these will, at the ordinary temperatures of burners used by sulphuric acid manufacturers, retain in chemical combination the following proportions of sulphur, which will therefore not be available for making acid.

For each per ct. of	Zn.	ZnO.	Cu.	CuO.	Pb.	PbO.	CaO.	MgO.	Fe_2O_3.
There will be found	0·50% S.	0·39% S.	0·50% S.	0·40% S.	15% S.	0·14% S.	0·57% S.	0·80% S.	0·60% S.

This on the assumption that all the sulphides are converted to sulphates, which is by no means the case.

An ore, therefore, which assays 43% sulphur, but also contains, say, 5% zinc,

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* The cinder (about 70 or 80% by weight of many of these ores) consists principally of iron oxide, and since it is entirely free from phosphorus, it possesses according to its purity a value as a Bessemer iron ore for admixture in small proportions with the regular blast furnace charges. This value is represented by the market price at the nearest blast furnace and consequently in many localities is entirely absorbed by cost of handling and freight. However, large quantities of the "fines" cinder are now being used by fertilizer manufacturers as a "filler" for their manufactured goods.

† As regards the cinder of this class of sulphides, it may be stated generally that the total produced by any except the largest sulphuric acid manufacturers is too small to justify further metallurgical treatment at the acid plant. It is generally reserved by the seller of the ore and sold or treated by him for his own account. Otherwise, the value of the cinder to the acid manufacturer generally is represented by the value of the same delivered to the nearest metallurgical plant having facilities for treating such cinder.

2% copper, and 1% lime, would really contain 43—4·07=38·93% available sulphur, and as even ores which practically contain nothing but iron and sulphur are rarely burnt down on an average much under 1%, an additional 1% would have to be deducted for ordinary loss in burning, making the available sulphur in this ore: 43—5·07=37·93%.

Moreover, since the only sulphur which an acid manufacturer recovers is the "available sulphur," and since pyrites is usually sold at a price per unit of sulphur by assay, it is necessary to remember that the unit price is greater than it seems. Take, for instance, the above ore at a unit price of 10c.; 44% sulphur at 10c.=$4·40 per ton of 2,240 lb.; but as the manufacturer can only recover 37·93% S, he is really paying more than 11·5c. per unit for all the sulphur he can possibly expect to recover. Or, if he has the choice between an ore containing only 42% S, but no impurities, and an ore of 44% S, with 4% unavailable on account of impurities, deducting at least 1% more for loss in burning, then the ores compare not as 42:44, but as 41:39.

It is also necessary to consider the proportional value of the 48% and 38% grades of sulphides, and, for the sake of convenience, these ores will also be compared with brimstone containing say 98% S. One ton (2,240 lb.) of brimstone containing 98% S is equal to 2·042 tons of sulphide ore containing 48% available sulphur, or to 2·579 tons of sulphide ore containing 38% available sulphur. (If the sulphide ores contain 49 and 40% available sulphur, then the proportion is 1:2:2·45 respectively.) Therefore, 1 ton (2,240 lb.) of brimstone (98% S) has available 2,195 lb. of sulphur; 2·042 tons (4,574 lb.) of sulphide ore (48% S) have 2,195 lb. of sulphur, and 2·579 tons (5,779 lb.) of sulphide ore (38% S) have 2,195 lb. of sulphur. Or one ton (2,240 lb.) of available sulphur will be produced by 1·02 tons of 98% brimstone, costing at seaboard $20 per ton=$20·40; 2·084 tons of 48% sulphide ore, costing at seaboard, $5·50 per ton=$11·46; or 2·632 tons of 38% sulphide ore, costing at mines $3 per ton=$7·90. (These prices are approximate only and *pro forma.*)

From these figures it is evident that the cheapest source of supply depends directly on the location of the works, or, in other words, on the freights. For instance, a freight rate of $1·25 per ton on a 38% ore from the mines to the seaboard would make such ore cost $1·25×2·632+$7·90=$11·19, or almost as much as the high grade 48% ore would cost at works located on the seaboard.

The proportional weight of one ton of actual available sulphur having been established, therefore, as a ratio of 1·02 for 98% brimstone, 2·084 for 48% ore, and 2·632 for 38% ore, the same ratio holds good for each of these raw materials as regards labor at the works, including loading, storing, breaking, burning and in the case of cinder which has no value, of removing the same.

As Lunge points out (Vol. I., p. 51) also "under conditions otherwise equal the unburnt sulphur in the cinders is the same by weight. If, for instance, 5% of sulphur is left in the cinders, this amounts with a 35% ore to 5÷35=$\frac{1}{7}$; with a 45% ore to only 5÷45=$\frac{1}{9}$; the proportion to be kept in view is accordingly not 35:45=7:9, but 30:40=3:4."

A final consideration of the chemical condition of the ore refers to such constituents or impurities as pass over into the acid itself, with deleterious effect. These are principally arsenic and selenium. These impurities are of no consequence to fertilizer manufacturers, and in some other manufactures, but for acid intended for "pickling" iron previous to tinning or galvanizing, for the drug trade, or in fact for the general run of the chemical trade, they are absolutely inadmissible. Fortunately, as will be shown elsewhere, these impurities can be practically eliminated without excessive cost, and pyrites acid can be made as pure in every way as brimstone acid. The prejudice which exists against pyrites acid is largely the fault of the manufacturers themselves, many of whom have neglected to supply themselves with the necessary appliances.

So far as the chemical condition of the sulphide ore is concerned, therefore, the intending purchaser in any given locality, in order to decide on the cheapest source of supply, must consider, at the prices quoted to him, (1) the amount of sulphur in any ore which will be actually available for his use; (2) what the freight on that actually available sulphur will be, and what the cost of handling the material necessary to obtain such available sulphur at his works, and (3) whether the ore contains arsenic, selenium or other impurities which he is not prepared to eliminate, but which may unfit the acid produced for his purposes.

Suppose we have two ores, one with 44% S; 42 Fe; 4·5 Zn; 0·5 Pb; 1·2 Cu; 0·01 As, and 7·3 insoluble residue; and the other 51·5% S; 46·0 Fe; 0·15 Pb; 0·4 Cu; 0·1 CaO; 0·15 MgO; 0·12 As, and 1·45 insoluble; from analysis I. we must deduct the sulphur which will be bound by the zinc, lead and copper, or 2·92%, and add 1% for loss in burning, or 3·92% S, making the available sulphur 40·08%; from analysis II. we have to deduct the sulphur bound by the lead, copper, lime and magnesia, or 0·4%, and add 1% S=1·4, making the available sulphur 50·1%; consequently the two ores will bear the ratio of 40 to 50. For each ton of actual available sulphur purchased by sample I., 2·5 tons of the ore must be bought, and of sample II., 2 tons. Freights will cost, therefore, 25% more on ore I. than on ore II., and works' expenses in the same proportion. Ore II., however, has the disadvantage to some purchasers of a rather large content of arsenic. For each locality, and for each manufacturer, the equation has to be solved between prime cost and cost of actual available sulphur.

With the exception of the Newfoundland and washed or leached[*] sulphides, nearly all the foreign ores shipped to this country are suitable for use in lump burners. The domestic ores on the other hand are mostly granular, and although a considerable proportion of all these ores can be obtained suitable for lump burners, the tendency is to use fines burners to a greater extent. Certainly, in order to purchase to good advantage, no matter what ore is used, it is well to have a certain proportion of fines, as well as lump burners. In this way "run of mine" can be bought. Since this is inconvenient in a small works, the tendency is to provide only fines burners, especially as this class of burner has been much improved recently, and types of fines burners are in use which will handle such part of the product of most domestic mines as will

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[*] Washed or leached sulphides are such as have been exposed to the action of the weather, in immense piles, for a number of years for the purpose of copper extraction. Mechanically, they are in such condition as to crumble very readily, and should be screened through 1 in. mesh before being used in lump burners.

pass through a screen of 1 in. mesh, giving as complete a roast as most lump burners. Without exception the zinc blendes, zinc-lead and zinc-lead-copper sulphides and pyrrhotites, as well as most concentrates, must be roasted as fines, either in muffle furnaces, or with the enrichment of pyrites high in sulphur or other fuel.

(*d*) *Brimstone.*—Sicilian brimstone is imported as firsts, seconds and thirds, according to quality, and sold at a fluctuating value by the ton of 2,240 lb. delivered at Atlantic seaports. The Pacific coast is now being supplied by brimstone from Nevada, delivered in San Francisco at about $19 per ton. The crude material is of very high grade, and is sold guaranteed 95% S.

(*e*) *Spent Oxides of Gas Works.*—Most works producing illuminating gas from coal purify their product from sulphurous acid gas (resulting from the decomposition of pyrites contained more or less in all coals) by passing it through a mixture of ferric hydroxide and sawdust. Ferrous sulphide and sulphur are formed. On exposure to air the ferrous sulphide reverts to ferric oxide with the formation of more free sulphur. After being used and regenated many times the mixture becomes too thoroughly impregnated with sulphur to be of further use as an absorbent and it is replaced with fresh ferric hydroxide. The sulphur of this waste product runs as high as 60%, the average being between 45 and 50%. The presence of lime in the waste or spent oxide, however, renders a portion of this sulphur unavailable for acid making, the sulphur being retained as calcium sulphate.

It is not, however, advantageous to use sulphur in the spent oxide directly for making sulphuric acid on account of the ammonia salts, and ferro and sulphocyanides contained therein. These not only are of value themselves, but unless removed would seriously interfere with the sulphuric acid process by destruction of niter. The ferro and sulphocyanides, together with ammonium chloride, may be extracted by hot water, and by evaporation of the solution a separation of the residue by alcohol into insoluble calcium ferrocyanide and soluble sulphocyanide and ammonium chloride can be effected.

M. Richter washes or steams the spent oxide every time before it is regenated, preferably by treating it with air in a box fitted with divisions. In this way the generation is combined with the recovery of ammonia since the process of oxidizing the ferrous sulphide to oxide and free sulphur evolves so much heat that the ammonia is driven off and can be washed out of the escaping gas by means of acid. Hempel & Sternberg treat the spent oxide, after having first removed the ammonium salts, by washing with 10 to 12% aqua ammonia. This dissolves the Prussian blue as ammonium ferrocyanide without acting on the sulphur. From the solution the Prussian blue can be precipitated by acid, or else it is distilled with lime to recover the ammonia, and the calcium ferrocyanide is converted by potassium carbonate into potassium ferrocyanide.*

The spent oxides having been freed from ammonium salts and ferro and sulphocyanides are desulphurized by combustion in any suitable furnace, but

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* Lunge, *Coal Tar and Ammonia*, 2d Ed., p. 605.

preferably in such burners as are used for desulphurizing ore "fines." Where spent oxides are produced, hydrogen sulphide from the manufacture of ammonium sulphate is also frequently a by-product. This can be burnt in connection with the spent oxide and used with advantage in the manufacture of H_2SO_4. The sulphur can also be extracted by solution in hot benzol, from which it is recovered by cooling and distillation.*

II. NITROUS ACID GAS (Nitrous anhydride, N_2O_3) is obtained chiefly from nitrate of soda.

(*a*) *Nitrate of soda* ($NaNO_3$) is imported from South America. The average composition of nitrate of soda for chemical works is 96% sodium nitrate (including nitrite, iodate, etc.); 0·5% chloride (calculated as NaCl); 0·75% sulphate (calculated as $NaSO_4$); 2·75% moisture—total, 100.† The usual contract in the United States calls for "not less than 96% nitrate of soda and not more than 1·25% salt" (sodium chloride). It is of great importance, however, to the sulphuric acid manufacturer to buy on the actual nitrate of soda contents. Nitrate of soda invariably contains a certain percentage of nitrate of potash. This is not objectionable when the nitrate is for agricultural purposes,‡ but in acid making a loss of 2% has been observed from this cause.

The nitrate of soda mixed with sulphuric acid is decomposed, aided by the heat of the burner gas, and the nitric acid gas converted in the presence of SO_2 into nitrous acid gas, enters into the Glover tower and subsequently the chambers together with the sulphurous acid. Theoretically, about 70 lb. of 61° B. acid should be added to 100 lb. of 96% nitrate, but in practice about equal parts by weight should be used.

(*b*) *Nitric acid* (HNO_3) is made by distilling nitrate of soda with an excess of sulphuric acid. The nitric acid is condensed in special apparatus. In this manufacture there is always a certain proportion of the nitric acid produced either too weak or too impure to be desirable or marketable in all localities. Further, there is a considerable loss of non-condensed oxides of nitrogen, both in the manufacture of nitric acid and of mixed acid, which can best be saved by finally passing the exit gas through a scrubber or tower and absorbing the contained nitrous and hyponitric acids in sulphuric acid of about 60 or 62° B., thus forming nitrous vitriol, containing 2 to 3% N_2O_3. This weak or impure acid and the nitrous vitriol are very desirable as means for supplying the chambers with nitrous acid gas by means of the Glover tower.

(*c*) *Nitrous acid gas* recovered from various chemical processes and the spent acid from high explosives and other manufactures, *e.g.*, the manufacture of cupric sulphate where nitric and sulphuric acids are used, may be of local value. Providing the nitroglycerine has been properly separated from the waste acid, and the latter can be promptly run to the Glover tower, such spent acid is a valuable source of nitrous acid for the chambers, and this is the most rational means of recovery of the nitric acid and partial concentration of the sulphuric.

In selecting the raw material to be used for the production of nitrous acid gas, the choice is practically limited to nitrate of soda, except locally, or in the

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* Broadbery, *Gas World*, 1895, XXIII., p. 643. † Lunge, Vol. I., p. 78. ‡ Lunge, Vol. I., p. 78; III., p. 756.

following cases: (1) Where the sulphuric acid manufacturer is also a manufacturer of nitric acid, or uses some process where he has waste nitric acid, or fumes of nitrogen oxides. (2) Where he can purchase waste or spent acids from high explosive or other manufacturers. (3) When his operations are on a sufficient scale to justify his manufacturing nitric acid for the chambers alone.

The relative advantages and disadvantages of using nitrate of soda by the ordinary process of potting and liquid nitric acid, or nitrous vitriol, by the agency of the Glover tower, are thoroughly discussed by Lunge, Vol. I., pp. 399–402. The advantages of using nitric acid or nitrous vitriol on the Glover tower are so great, the regularity secured in the process is of such value, that no minor considerations should prevent its use where it is in any way possible.

III. STEAM.—Coal, coke, wood, natural gas, or petroleum are used for raising steam, depending to a considerable extent on the locality. In the concentration of acid, natural or fuel gas or oil are much to be preferred to coal, coke or wood, on account of the even temperature which can be maintained by their employment.

A good supply of pure cold water is of great importance to the acid maker; pure on account of the boilers; cold for cooling the acid and consequently securing an efficient recovery of niter. Apart from steam supplied to the chambers, which need not be over 20 lb. pressure and may be less, power is required principally for pumping acid to the towers; also for fire protection, heating, etc. In case pure water is costly, or hard to get, then a separate supply of less pure or even sea water may be used for cooling the acid, or better still the water first used for cooling may be afterward used for the boilers; in other words, the coolers may be so arranged as to act as feed water heaters. Where a reasonably steady supply of exhaust steam is available, it may by a simple arrangement be used for the chambers, either entirely, or as an auxiliary to live steam.

PREPARATION OF RAW MATERIALS.—When the sulphurous acid is produced as a metallurgical by-product it will often be necessary to subordinate its conditions to the exigencies of the metallurgical operations. In case no alteration is possible in these operations it is merely necessary to determine the average conditions and value of the gas in question to decide on the character of the condensing apparatus, or whether the gas is suitable, or can be made available, for the manufacture of sulphuric acid.

The principal consideration is the constancy or regularity of the conditions of the gas, whatever these may be. Under certain restrictions it is far more easy to make acid from a uniformly poor gas than from one which at irregular periods is alternately rich and then poor. Another consideration is the temperature of the gas. In metallurgical operations this is apt to be rather low on account of the necessity generally existing of eliminating dust on account of its metallic value or its high content of arsenic. A complete elimination of dust is hardly possible at a temperature much above that of the normal atmosphere. This, however, does not seriously interfere with the use of the gas in acid making, with proper condensing apparatus, as for examples the practice* at Freiberg, Saxony. A third consideration is the percentage of SO_2 contained in the gas.

Percentage of SO_2.—The desirable and normal percentage in SO_2 produced from metallic sulphides is from 7·5 to 8%. The minimum at which acid can be made profitably, under most circumstances, would be about 4% SO_2, or slightly under, but it must be remembered that the poorer the gas in SO_2 the greater will be the first cost of the condensing apparatus.

The general idea that the carbonic acid gas contained in so many of these metallurgical gases is harmful to the process does not seem to be substantiated by facts. It is certainly an adulterant gas, as is the case with nitrogen. Its higher specific gravity rendering possible the formation of separate layers should be overcome by such improvements in the condensing apparatus as plate columns and better draft facilities, such as fans.

In very few instances, however, is it necessary that metallurgical gas shall be so low, or nearly so low, as 4% SO_2. Properly constructed furnaces will yield gas containing almost if not quite as high a percentage of SO_2 as furnaces intended for burning pyrites expressly for manufacturing sulphuric acid, and in very few instances need the gas fall below 6% SO_2. Where such is not the case, however, it will usually be found more economical to substitute properly constructed furnaces, rather than to incur the higher cost of condensing apparatus necessitated by a weaker gas, or to go on wasting the gas. It is also in many cases possible to enrich the gas either by an admixture of ore richer in sulphur or by burning such an ore or even brimstone, in separate auxiliary furnaces.

Lump and Fine Ore.—When on the other hand the sulphurous acid gas is produced from an ore expressly for the manufacture of sulphuric acid, whether the residual cinder is of value or not, then these ores divide themselves into the following classes: (1) Ores which are prepared for burning on grates in lump form to best advantage; (2) ores which are fines resulting from the preparation of the lumps and which can be burnt to best advantage on fire clay shelves in shallow layers without auxiliary heat; (3) ores which can only be burnt as fines and which require auxiliary heat either by the enrichmentment of higher grade sulphides, admixture of carbonaceous fuel, or by means of fire-clay shelves with bottom heat (zinc blende, pyrrhotite, etc.)

In the preparation of these ores for burning to advantage on the several types of furnaces, the following points should be considered: As ores are purchased under the name of "run of mine," or "broken to kiln size," or "smalls," or "fines," it is implied either that the ore sold has received at the mine no preparation for burning other than a more or less complete elimination of foreign matter (gangue) or else that it is sold ready for burning in the kilns suited to the various sizings represented by "kiln size," "smalls," or "fines," and generally at an advanced cost for such preliminary treatment, at any rate in the case of lump ore. In the one case the manufacturer must be supplied at his works with machinery for crushing, sizing and sampling the ore; in the other case this work has been done at the mines. The advantages

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* Lunge, 2d Ed., pp. 301, 473.

to the manufacturer of preparing his own ore, at any rate to the manufacturer on a large scale who is supplied with the various kilns necessary, are first that the work is more likely to be properly done and suited to his own kilns, and next that he will buy run of mine cheapest. The only objection to having the ore sized at the mine is that some of the more friable ores disintegrate considerably during transit and handling at the works. Otherwise the preparation of the ore at the mine would be the most rational way, since in handling large quantities the work can be done more cheaply and much dust and dirt saved at the acid plant.

Necessity for Proper Sizing.—Too much stress cannot be placed on the necessity for proper sizing of the ore, but this subject, except in so far as it suits the convenience of the miner, is almost entirely neglected. The miner dumps the product of his mine into a breaker the size of the product of which seems to be indifferent and to vary at each mine. He screens this through say a 0·5 or 0·25 in. mesh. What passes this screen is shipped as "smalls" or fines; what does not pass, as "broken to kiln size." The result is that "broken to kiln size" means according to the breaker and screen used at any mine, a mixture of sizes varying from dust to pieces 1 in. to 4 in. in size. If a more careful sizing is attempted the miner will endeavor to make his complete output of kiln size as large as possible, "because that is the best size for burning ore," and incidentally, perhaps, because a less proportion of fines is produced by breaking to this size. Without due consideration the manufacturer will often buy ore of "broken to kiln size" indiscriminately. Consider the result: Probably not half a dozen benches of lump kilns, in this country at any rate, have been erected specially for burning any particular ore. The change from brimstone has been made recently, often hurriedly and generally from stereotyped drawings and not often under competent advice. The grate bars of these burners when once built are spaced in such a way that their relative position cannot be changed without almost tearing the kiln to pieces. The maximum spaces between these bars varied in 12 different cases recently measured from 0·5 in. to 2·25 in. The manufacturer who purchases ore broken to an average size of 3 in. and whose grate bars are spaced 0·5 in. apart should hardly expect satisfactory results. But the ore is generally condemned, whereas the size to which it is broken is simply not suited to the spacing of his grate bars.

PYRITES BURNING.—There is no mystery in the burning of sulphide ores. The problem is one of combustion pure and simple. The oxidation of sulphur in self-burning sulphide ores and of carbon in anthracite coal are in many respects analogous. No care in stoking could produce satisfactory combustion of an indiscriminate mixture of all sizes of anthracite, from rice or culm to furnace, on any kind of haphazard grate, and yet the combustion of each kind by itself can be satisfactorily accomplished on a suitable grate. With ore properly sized, and the proper grate used for the proper size, and the conditions as to depth of bed and draft suitable, each for its own size, the difficulties of proper burning and insufficient yields will disappear. Burner men frequently tell with pride what percentage of fines and smalls they can burn with their charges.

It is simply impossible to do this except at the cost of loss of efficiency in the burner or of sulphur in the cinder, except in so far as by skillful work a burner man can dispose of a limited amount by carefully shovelling it, sorted entirely by itself, around the walls of the furnace, where the draft is most free. This, it is hardly necessary to say, is rarely if ever done.

In a bench of burners having grate bars an average distance apart it is often possible to obtain excellent results by sizing the ore and setting aside certain burners for the different sizes burning; in some nothing but the smaller pieces, say, from 0·75 in. to 1·25 in., in others pieces from 1·25 in. to 2 in., and in others pieces from 2 in. to 3 in. Each size of course requires its own depth of bed and draft and care in shaking down, but the extra care will be amply repaid in increased yield. Of course when the grate bars are all spaced alike it is better to insist on the ore being properly sized when purchased, contracting for ore not under and not over certain sizes suitable for the furnaces in question. Serious as the necessity is of having the spacing of the grate bars reasonably suited to the size of the ore to be burnt, that is not by any means the only reason for proper sizing, the trouble termed "scarring" or "clinkering" being almost entirely attributable to imperfect sizing.

Clinkering.—In the combustion of pyrites a double oxidation is taking place, namely, oxidation of the sulphur to SO_2 and of the FeS_2 to FeO. In the absence of sufficient oxygen the sulphur will partly sublime, and the FeS_2 will be converted into FeS or a fusible matte, scar or clinker. Now one can readily understand what is taking place in a kiln filled with reasonably even-sized pieces of ore. The oxygen-bearing air passes with fair regularity almost completely around each piece, and the combustion goes on regularly; but when there is a considerable proportion of smaller pieces, and maybe fines and dust, there will surely be some points where the air will not be able to pass, the little channels being choked. The heat at and near the surface being great, the pieces so choked off from air will melt and become slightly fused; gradually the surrounding pieces will become involved and the scar will grow larger. Then the burner man will intervene, and with a heavy steel bar break up this scar, which ought never to have been. As the burner man's principal faculty, unfortunately, is generally the wrong application of his muscle, he will play havoc with his burner by breaking up to powder the unusually friable ores used in the United States and lay the foundation for all kinds of scars. It is no uncommon thing to see whole benches of burners scarred on account of want of sizing and excessive use of the bar. With reasonable care in sizing scarring would be unknown, and all the difficulty and vexation, and the oftentimes excessive financial loss so often incurred in burning pyrites in this country would disappear.

Preparation of the Ore for Burning.—Whether this be done at the mines or not it will consist of the following operations: (1)Breaking; (2) sizing; (3) sorting or "dressing;" (4) sampling; and with some ores (5) concentrating.

(1) The breaking is best done with a rock breaker, the only advantage which can possibly be advanced in favor of hand breaking being the smaller quantity of fines produced. And here we come to the very gist of the matter, and that

is the fact that fines have not been considered of the same value as lump. The miner therefore naturally has endeavored to make as few fines or smalls as possible, and therefore to break up his ore as little as possible, naturally making his lump ore contain as many of the biggest pieces he can get in as kiln ore, and at the same time make it contain as much of the small ore and fines as he can get the buyer to accept. He has generally been bound by contract to a maximum content of fines, or ore which would pass a ¼ in. to ½ in. mesh screen. In some cases he has contracted to take back fines, which has burdened him with an additional and entirely unnecessary freight; in others the manufacturer, not having the furnaces adapted for burning fines, has made a double loss: first, by forcing his lump burners to take as much fines as possible, and thereby securing execrable yields, and second, by piling them up in his yard, where he has finally sold them at a loss of from $1·50 to $2 or more per ton.

Labor Required in Pyrites Burning.—The principal, it might be said the only, reason for the inferior value of fines is the greater labor which has been involved in their use, and a rather larger capital outlay for furnaces capable of burning them, and unsatisfactory results when mechanical furnaces were adopted for their use. These conditions should be, and undoubtedly will be, almost if not entirely reversed. One of the principal reasons for their existence seems to have been the exclusiveness of the sulphuric acid manufacturer. Metallurgists have been using mechanical furnaces for many years which with slight modifications would burn fine ore perfectly for acid making. Had one tithe of the engineering skill been applied to acid manufacture which has been applied to the metallurgical treatment of sulphide ores the difficulty would have been solved long ago. Unlike the conditions existing in Europe, where chemical and metallurgical operations often work hand in hand, such cases in this country are isolated, largely no doubt because metallurgical works dealing with these ores are largely in the West and are of comparatively recent growth, whereas the sulphuric acid manufacture and market are on the Atlantic Coast.

The mechanical burning of pyrites fines economically may be considered as an accomplished fact. The various modifications of the McDougall type of furnace, and others hereafter described, are undoubtedly admirably adapted for this purpose. As regards the labor cost of burning coarse or fine sulphides at present the matter stands thus: Unskilled labor is employed to bring the ore to the furnace and take away the cinder. Where the quantity involved is large enough to justify the outlay, mechanical means can be substituted, and as whether the ore is lump or fines the labor involved is the same for equal tonnage, this factor may be eliminated as common to either condition of the ore. Skilled labor is used in the management of furnaces. In lump furnaces of modern construction the charge consists of about 1,000 lb. in 24 hours, and two men cannot well take care of more than 20 furnaces, or say 20,000 lb. of ore, in 24 hours. In hand shelf furnaces of modern construction the charge consists of 2,000 lb. to each burner in 24 hours, and two men can take care of six burners, or say 12,000 lb. of ore, in 24 hours. In the various mechanical furnaces, hereafter described, from 6,000 to 200,000 lb. can be [illegible] in 24 hours by two men. One man's labor therefore represents in the case of lump burners a maximum of 10,000 lb. of ore; hand shelf burners, 6,000 lb.; and mechanical burners, 100,000 lb. and over; or the old proportions of labor capacity, viz.: Lump: Fines :: 5 : 3 now stands Lump : Fines :: 5 : 50, and the mechanical fines burners, for the same capacity, cost very little more than the lump burners.

Relative Value of Lump and Fine Ore.—The one drawback to the mechanical fines burners is the amount of dust produced. So far as the manufacture of chamber acid is concerned and where the process is carried no further, this is often immaterial; when concentration by the Glover tower or further concentration in platinum is involved this is a serious difficulty, but it is purely mechanical, and as such may be overcome. It will be observed, however, that up to a consumption of 18,000 to 20,000 lb. of lump ore representing two men per 24 hours, or of 12,000 lb. of fines ore, mechanical burners show no advantage in economy of labor, and works of such capacity are not uncommon in this country. The large investment of capital in lump burners, the difficulty of purifying the acid, the existence of small plants of from 20 to 25 tons daily capacity chamber acid, in which there would be no reduction of labor cost, and other reasons make it quite unlikely that the use of lump ore will be discontinued. It is exceedingly likely, however, that the value of lump and fines ore will be equalized, and with every approach to such equality of value the interests of the miner and the manufacturer, so far as the preparation of the ore is concerned, will be more nearly identical.

(2) *Sizing.*—In considering the sizing of the ore, therefore, the probable increase in value of the fines should not be lost sight of. While it seems probable that a reasonable sizing of fine ore would give better results, not only as to yield (dead roast) but as to capacity of any furnace, than an admixture of every ore from dust to small lumps, yet both the hand shelf-furnaces and the mechanical furnaces will desulphurize a certain proportion of small lumps up to what will pass a 1 in. circular hole. Some ores will break at once so that being screened through a 1-in. circular hole the proportion of such lumps from ½ to 1 in. will not be too great, say only 3 or 4%. If so the preliminary sizing or differentiation between lump and fines may start with a 1-in. screen. Generally, however, it will be found necessary to commence with a ½-in. or at most a ¾-in. screen. The starting size for any given ore to produce the most economical results is a most important matter for the miner, and one upon which he should obtain expert advice. If he can start with a 1-in. screen and produce a fine ore suitable for the manufacturer well and good; if not, he should start with say ½ in. or ¾ in. and size his ore by not more than 1 in. increase of diameter, say, ½, 1, 2, 3 in., or ¾, 1¼, 2, 3 in., or 1, 2, 3 in., but in no case should the ore be larger than 3¼ in.*

Where the condition of the acid is of importance and the cinder of no value, much trouble will be saved if the fines are washed so as to free them from the impalpable dust or slimes which enter the condensing apparatus with the gas and cause much trouble. In many ores which are low grade the percentage of

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* The screens will wear to make the sizes slightly larger before being worn out.

sulphur can be readily raised four to ten units or more by a rough concentration which will amply repay its cost.

(3) *Sorting* is generally done on revolving or traveling tables, by boys who pick out the gangue or foreign matter.

(4) *Sampling* is done best by automatic samplers as the ore passes to the bins from the sorting table.

(5) *Concentrating* as such applies only to ores having the greater part of their value in the cinder. Almost every ore requires different and expert treatment.

PREPARATION OF THE NITROUS ACID.—Nitrate of soda is received at the works in bags, but should not be so stored. The bags should be emptied and the nitrate weighed and stored in a brick building with a cement floor, in which it can be kept as free from moisture as possible. Before using for "potting" all lumps should be thoroughly broken up. The bags should then be washed in warm water. Two vats should be used with steam coils in the bottom. After the bags have been washed in the first vat they should be wrung out and washed again in the second vat, then dried and burnt or otherwise sent away from the premises, since they are dangerously inflammable. The water in which they are washed should then be evaporated in lead pans, the organic matter having been first precipitated. By reasonable care the crystals of sodium chloride which form first can be collected separately so that the resulting nitrate of soda will be exceedingly pure. Or the evaporation may be done in the usual iron pans and the resulting nitrate of soda used for potting.

The preparation of nitric acid need hardly be considered here. Nitrous vitriol will be considered under "the chamber process." Nitrous acid gas recovered from other processes when not absorbed in sulphuric acid as nitrous vitriol may be led directly into the absorbing apparatus. So far as spent acids from manufacturers of high explosives are concerned, these acids contain from 10 to 12% of nitric acid and 70 to 80% sulphuric acid, and should be run over the Glover tower as soon as possible. If these acids are purchased from outside, great care must be taken that they are perfectly freed from explosives. They should be used immediately upon receipt and if of necessity stored must be watched, since minute quantities of nitroglycerine will form in them for some time. If for any reason these acids contain mud, they may then be denitrated by steam and the nitrous gas passed into the absorbing apparatus, or they may be purified by filtration or settling (W. R. Quinan).

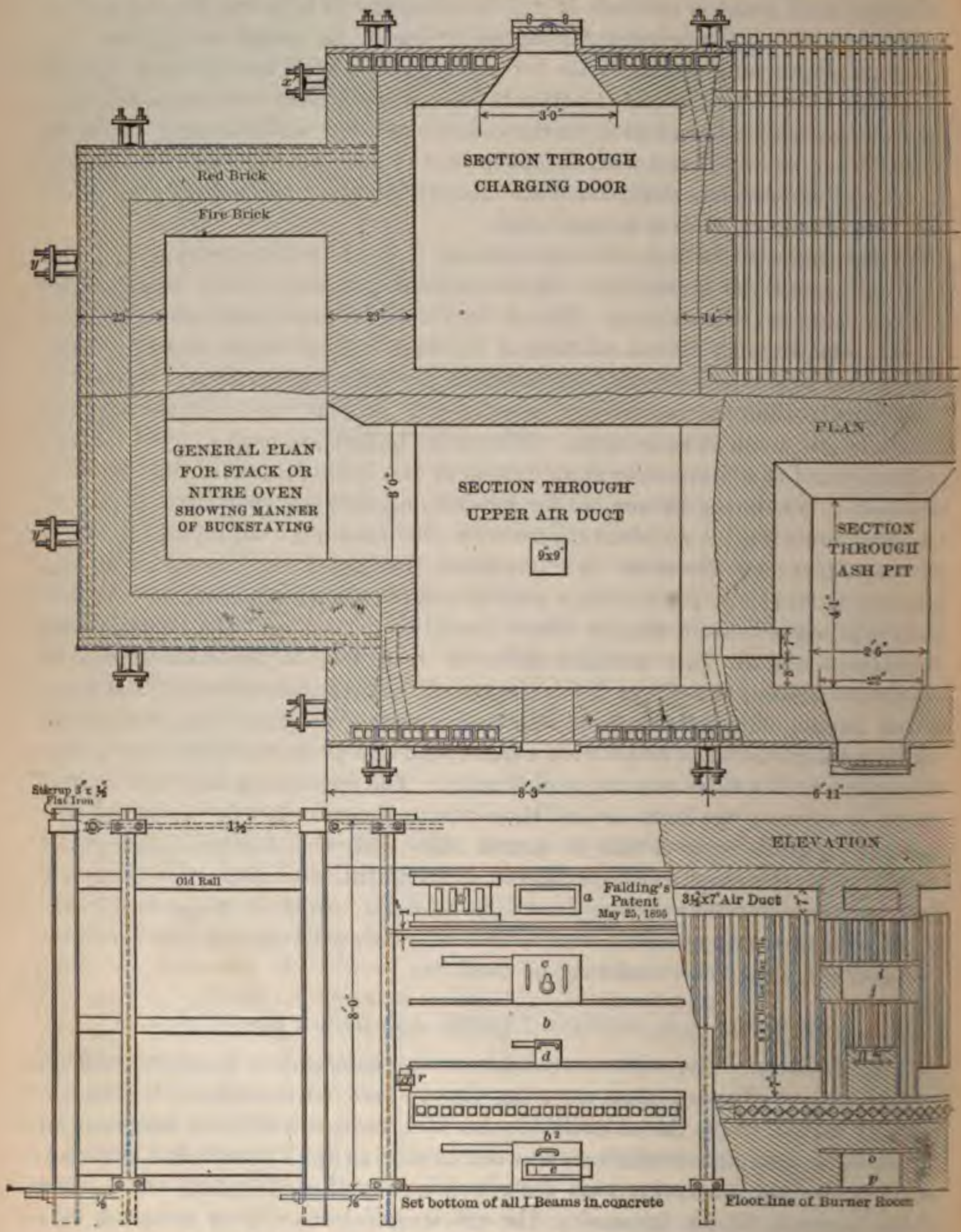
PREPARATION OF STEAM.—For the manufacture of chamber acid the only steam required, as steam, is for the addition of the necessary water to the acid fumes in the absorbing apparatus, and as previously pointed out, this may consist wholly or in part of exhaust steam. All other steam used is for power and may be applied either in the form of electricity, compressed air, or gas. The principal and in the case of some plants the only power required is for pumping acid, and as this is invariably done by means of compressed air, this air may be brought from a central or distant plant. In case, therefore, as commonly happens, the sulphuric acid works are only a small part of a larger plant, the steam for the process (live or exhaust) can generally be brought over in well-covered pipes, and the compressed air generated near the boilers in the central power plant of the works, the pumps for fire protection and water supply being also located centrally. On the other hand, if the sulphuric acid plant is isolated, it will be necessary to have two boilers each of which is capable of supplying the necessary steam for water supply, fire protection, compressing sufficient air for pumping the acid, for the acid-making process and for heating. It is better to have these boilers of ample size, since the acid-making process is a continuous one and one which should never be discontinued or interfered with except in case of absolute necessity; the apparatus once in operation will deteriorate more from two or three days' shut down than from many months of normal work.

For the process of sulphuric acid making it is of first importance that the supply of good dry steam shall be of constant pressure, which need not be great providing it is constant. About 20 lb. is advantageous unless in some special case, such as when a mixture of live and exhaust steam is being used, when a less pressure is admissible, the steam pipes and connections of course being larger in diameter and in any case properly proportioned to the size of the plant and the work to to be done. Where the boilers are used simply for the production of this steam they should be kept at a higher pressure than actually required. Where the boilers are for general purposes they will naturally be at a higher pressure. A constant pressure on the main pipe supplying steam to the chambers can therefore be maintained by providing the main at the entrance to the acid plant with a good pressure reducer and gauges. At this point also a good steam trap or dryer should be placed and the main beyond the reducer should have a slight fall away to a trap at the extreme end of sufficient capacity to drain the main and all branch connections. The main and all branch connections should be well covered and each branch supplied with a ¾-in. globe valve fitted with a brass disk and pointer so that the degree of opening of the valve may be readily seen. The degrees on this disk should be cut or sawn in the brass so that they cannot become effaced. In using this disk with a globe valve it will of course show only the degree of opening of one turn of the stem. This, however, is sufficient to register the comparatively small variations required from time to time. Suppose the normal opening of a valve is two complete turns, then the disk will register changes from 2—1 to 2+1, or a brass cock may be used.

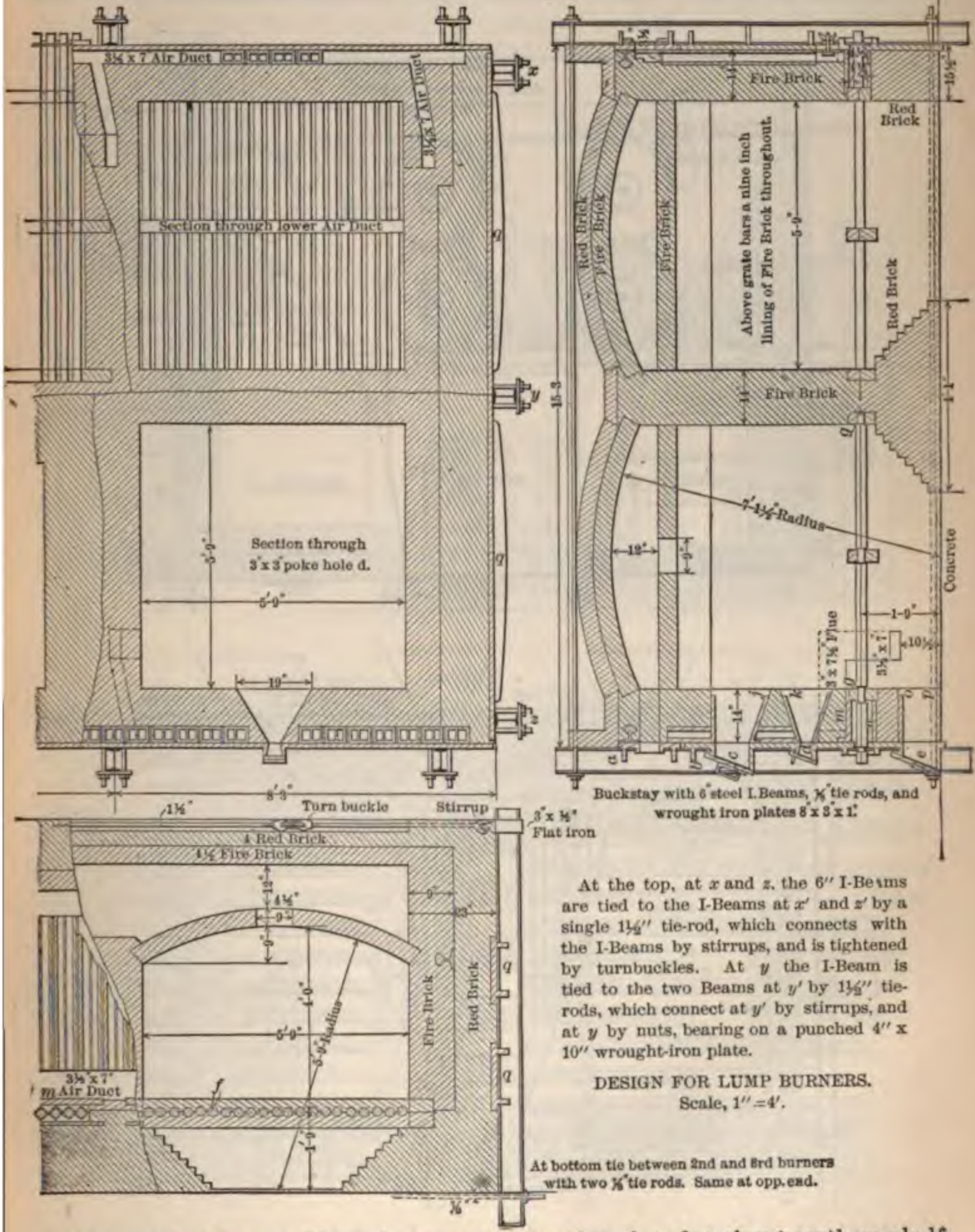
FURNACES FOR BURNING PYRITES AND THEIR OPERATION.

Lump Burners.—The accompanying drawings illustrate a bench of modern lump burners. The principal novelties about these burners are: (1) The use of an air jacket which serves either to prevent radiation of heat, keeping the burner room cool and rendering the work of serving the burners lass arduous, at the same time conserving heat for concentration of acid in pans on the top of the burners, or for increasing the efficiency of the Glover tower in this respect, or as a source of supply of hot air to the burners themselves, which is of considerable advantage in burning some ores. (2) The method of holding in position the grate bars so that they may be drawn and replaced, the false air admitted may be reduced to a minimum and the position of the bars may be seen at any time.

In the construction of any type of furnace care must be taken to make all flues or openings into flues perfectly air-tight, since all air admitted to the burners or flues, except where it will aid combustion and will be under complete control, is of great detriment to the process and renders its proper



regulation difficult. In lump burners covered by cast-iron plates carrying lead pans for concentration of acid especial care should be taken to secure tight joints.



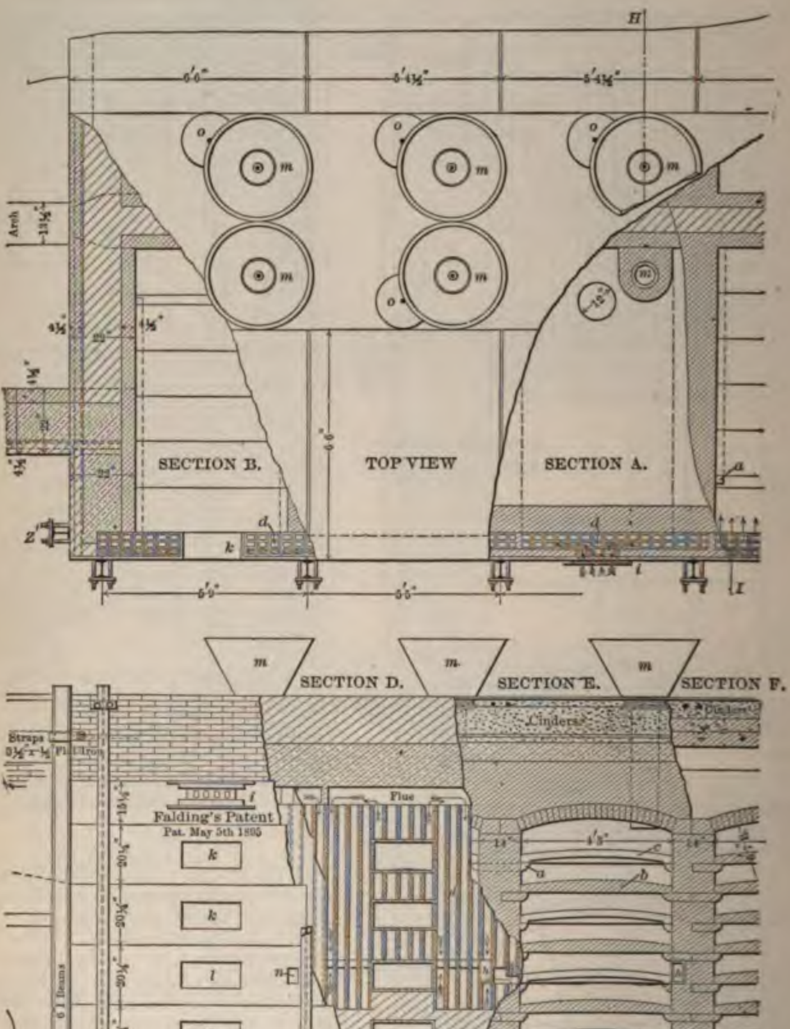
Buckstay with 6" steel I.Beams, ⅞" tie rods, and wrought iron plates 8" x 3" x 1".

At the top, at *x* and *z*, the 6" I-Beams are tied to the I-Beams at *x'* and *z'* by a single 1½" tie-rod, which connects with the I-Beams by stirrups, and is tightened by turnbuckles. At *y* the I-Beam is tied to the two Beams at *y'* by 1½" tie-rods, which connect at *y'* by stirrups, and at *y* by nuts, bearing on a punched 4" x 10" wrought-iron plate.

DESIGN FOR LUMP BURNERS.
Scale, 1"=4'.

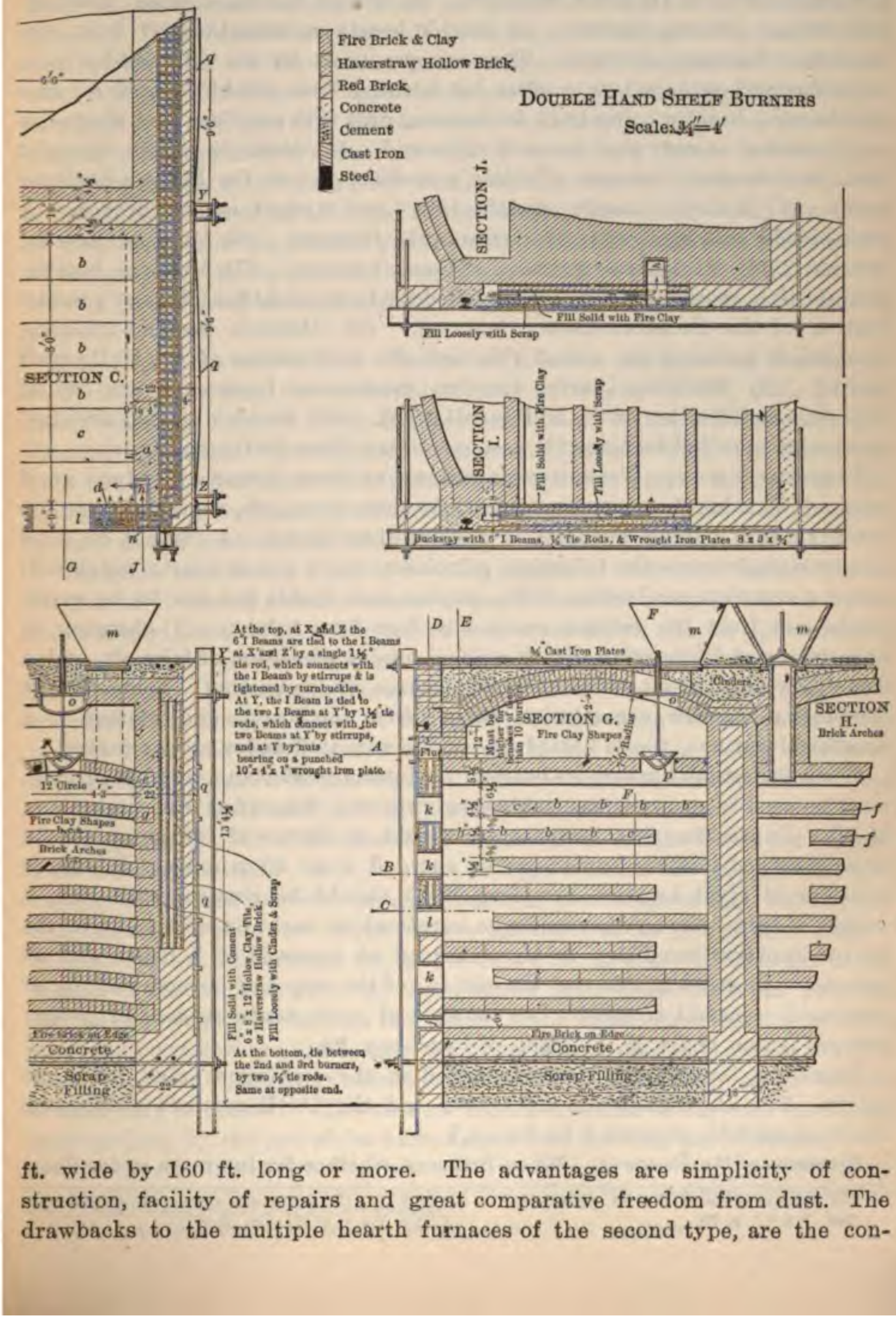
At bottom tie between 2nd and 3rd burners with two ⅞" tie rods. Same at opp. end.

Burners for Fines.—These burners all consist of a fire-clay hearth or shelf with an arch or another hearth or shelf placed a short distance above so as to localize the heat on the ore bed by reflection as fully as possible. They are of two types. (1) Where the furnace consists of one hearth of moderate width and considerable length, the ore being fed to the furnace at one end and being moved forward gradually, being discharged at the other end when burnt out; and (2) where a number of short hearths of moderate width are



superimposed one above another, the ore being fed to the top hearth and moved forward and down one hearth to another below until the ore is discharged, burnt out, from the lowest hearth. These furnaces may be either hand worked or mechanically worked, and may have hearths arranged in a straight line or circular.

The drawback to furnaces of the first type is the extent of space required in their construction, some furnaces being built with hearths 10 ft. wide by 160 ft. long or more. The advantages are simplicity of construction, facility of repairs and great comparative freedom from dust. The drawbacks to the multiple hearth furnaces of the second type, are the con-



DOUBLE HAND SHELF BURNERS
Scale: ¾″=4′

verse, namely, more complicated construction, difficulty of repairing and excessive dust, the advantage being compactness and the small amount of space occupied and conservation of heat.

Examples of these types are as follows: (1) Single hearth, straight line, mechanical furnace (Holthoff-Wethey*). (2) Single hearth, circular, mechanical furnace (Pearce turret†). (3) Double hearth modifications of both the preceding furnaces are built. They are also built for use with self-burning sulphides and arranged as muffles for burning base sulphides such as zinc blende, etc., besides being built for burning ores with auxiliary fuel where the gas is wasted or not used for sulphuric acid. (4) Multiple hearth, straight line, hand-worked furnaces (Falding's modification of the Malètra-Shaffner type). (5) Multiple hearth, straight line, hand worked muffle furnaces for zinc blende and base sulphides (Rhenania furnace). (6) Multiple hearth, straight line, mechanical furnaces (Spence burners). (7) Multiple hearth, straight line, mechanical furnaces, muffled for base sulphides (Hegeler's modification of the Eichhorn-Liebig furnace). (8) Multiple hearth, circular, mechanical furnaces, air cooled (Herreshoff's modification of the McDougall type).‡ (9) Multiple hearth, circular, mechanical furnaces, water cooled (Frasch's modification of the McDougall type). (10) Multiple hearth, circular, mechanical, muffled furnaces for base sulphides (Haas-McDougall type).

Brimstone Burners.—Various constructions of these furnaces produce good results. It is hardly necessary to describe them at length, since their use for sulphuric acid making purposes is rapidly diminishing. A furnace of good construction involves the following principles: (1) A pan or hearth which will secure a complete combustion of the sulphur and enable the ash to be easily discharged from the furnace practically free from sulphur; (2) charging of brimstone and discharging of ash without undue admission of false air, and a complete control of air admitted to the furnace for combustion; (3) an efficient combustion chamber, separate and distinct from the hearths and common to a number of burners, where heated air may be admitted under proper control.

When brimstone burners are built on the intermittent plan a number of them must be used to secure a reasonably steady current of gas of an even percentage of SO_2. In working these burners the weight of charges of brimstone should be so arranged that one burner can be charged every 20 minutes. In case of a bench of eight burners, therefore, No. 1 should be charged with such a weight of brimstone as can readily be burnt out in such time as to leave the burner in proper condition to be recharged at intervals of 2 hours and 40 minutes. Charged in this way the exit gas of the bench of burners will be of reasonably constant strength. An example of good construction of brimstone burners is that of Dr. K. F. Stahl, of Pittsburg, Pa.

Brimstone burners are also constructed on the continuous plan, but more rarely. The Harrison-Blair, the Glover and the De Hemptinne continuous furnaces are fully described by Lunge.§

Operation of the Furnaces.—When furnaces, whether for lump ore or for fines,

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* THE MINERAL INDUSTRY, Vol. VI., pp. 449-450.

† *Ibid.* Vol. III., pp. 205-211, and Vol. V., pp. 267-268.

‡ *Ibid.*, Vol. VI., p. 237.

§ *Ibid.*, Vol. I., 2d Ed., pp 198-205

are operated for the sole purpose of producing sulphurous acid gas for sulphuric acid making, their only function is to produce at the least cost for labor and repairs a steady stream of gas, containing a constant percentage of sulphurous acid gas, with as complete an oxidation of the sulphur in the ore as may be possible, or, in other words, with as little waste of sulphur in the cinder as possible. This is their one and only function, and the nearer the construction and operation of the furnace in question approximates such result the more perfect it is. The object being a steady stream of gas at a constant strength of sulphurous acid gas, it is first necessary to determine what this percentage should be for the sulphuric acid process. There are two factors in this matter: (1) The furnaces must be supplied with sufficient air to properly oxidize the sulphur, an excess or insufficiency of air being about equally fatal, depending on the character of the furnace and the chemical and mechanical composition of the ore; and (2) the condensing apparatus must be supplied with a gas containing an excess of atmospheric air, or rather oxygen, to secure a satisfactory process.*

Experience has shown that the gas as it leaves the furnaces, to enter the condensing apparatus, should contain from 7·5% SO_2 to 8% SO_2, estimated by Reich's test, in order to obtain the best results both in the furnaces and condensing apparatus. In some fines burners with certain ores, this percentage may be slightly increased without detriment to either part of the process while in burning brimstone, in furnaces suitably constructed to avoid sublimation, the gas should contain from 10 to 11% SO_2.†

The great importance, whether burning pyrites or brimstone, in any type of burner, of obtaining a steady stream of gas of constant composition, cannot be too strongly accentuated. The nearer such a condition can be approximated the more nearly will the whole process be uniform, and in fact almost mechanical. Nearly all the trouble experienced in making sulphuric acid springs from fluctuations in the percentage of SO_2 in the burner gas. Other parts of the process are almost entirely mechanical, such as the regular supply of nitrous acid gas and steam. For this reason it becomes imperative to be able to test the gas for SO_2 frequently and easily at the exit of the burners.

The gas at the exit of the absorbing apparatus is usually tested for oxygen. This test may be useful, as in case the exit gas contains the proper excess of oxygen the furnace gas will contain the proper percentage of SO_2; but the contrary also holds good, and if the furnace gas contains the proper percentage of SO_2 the exit gas will contain the proper excess of oxygen. In the case of the testing of the exit gas, however, the mischief when discovered has gone too far, in fact completely through the system, while on the other hand a variation from the normal of the furnace gas can often be remedied even before the variation has been felt to any extent in the absorbing apparatus, and so serious trouble may be averted. The burners should therefore be adjusted and managed entirely by the test of the burner gas where it leaves the furnaces.

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* Atmospheric air consists of 21 parts of oxygen and 79 parts of nitrogen; the nitrogen passes through the whole sulphuric acid process as an entirely inert gas.

† For an elaborate discussion of this matter, see Lunge, 2d Ed., Vol. I., pp. 303–315.

Regulation of Air Supply.—To maintain this regularity of the burner gas it is necessary to secure with any given ore of average sulphur content: (1) A regular supply of ore to the furnaces. In the case of mechanical furnaces this is done by means of a constant mechanical feed; in the case of hand-worked furnaces, by means of using a series of such furnaces, which are charged and discharged at regular intervals, in such a way that the average of the gas produced by the series remains practically constant. (2) A regular supply of air admitted to the furnaces. This implies not only that the openings through which the air is admitted to the furnaces shall be adjusted, but also that the speed of the current of air shall be uniform; and further, in the case of intermittently charged hand-worked furnaces, that the supply shall vary approximately as the sulphur contents of the ore in each individual furnace. It is manifest that an ore containing its maximum of sulphur, say 50%, requires more air than when it has been burnt down to 2 or 3% S, at which stage an excess of air simply serves to cool the ore below its combustion point and to prevent the complete oxidation of the sulphur, thus causing a waste of sulphur in the cinder and an excess of false air to the burner gas. On the other hand, in mechanical furnaces for the same reason the point of application of the air is important. It should be supplied at the point in the furnace of most active combustion, and not allowed to sweep over and cool the almost burned out cinder on the lower hearths or at the discharge end of the furnace.

It is obvious, therefore, that apart from the method or location of application of the air the matter of prime importance is the speed of the current of the air, or, in other words, the draft. It is also reasonable to say that, since it is of supreme importance to the process that the absorbing apparatus should be supplied with a constant stream of gas of a uniform composition, where it will pass through innumerable chemical changes under entirely different conditions, the regulation of the speed of the air current admitted to the furnaces should be accomplished by means entirely independent of and not affecting the absorbing apparatus.

Draft System.—The arrangements for draft in sulphuric acid plants are largely empirical. A certain attention is given to the size of the flues and connections in some cases, with a view to avoiding undue obstruction to the gas current. The Glover tower is often packed loosely with the same object, although at the cost of efficiency, but in very few instances is there any well-grounded certainty as to what the final conditions of draft will be in the erection of any sulphuric acid plant. In very few instances indeed are the conditions of draft such as to obtain the maximum efficiency both of the furnaces and absorbing apparatus, and in no case where this important matter is left to natural causes is the draft, and consequently the composition of the furnace gas and the condition of the whole acid process, unaffected by adverse or changeable meteorological conditions.

In the ordinary sulphuric acid plant the draft on the furnaces is produced (1) by the vacuum caused in the absorbing apparatus by the condensation of the gases into sulphuric acid, and the contraction caused by cooling; (2) by the siphon-like action of the cool gas behind the Glover tower, in some cases of construction, and the tendency of the hot gas to ascend, and (3) by the chimney or stack at the exit of the whole system, or by steam inspirator or other appliance at the same point.

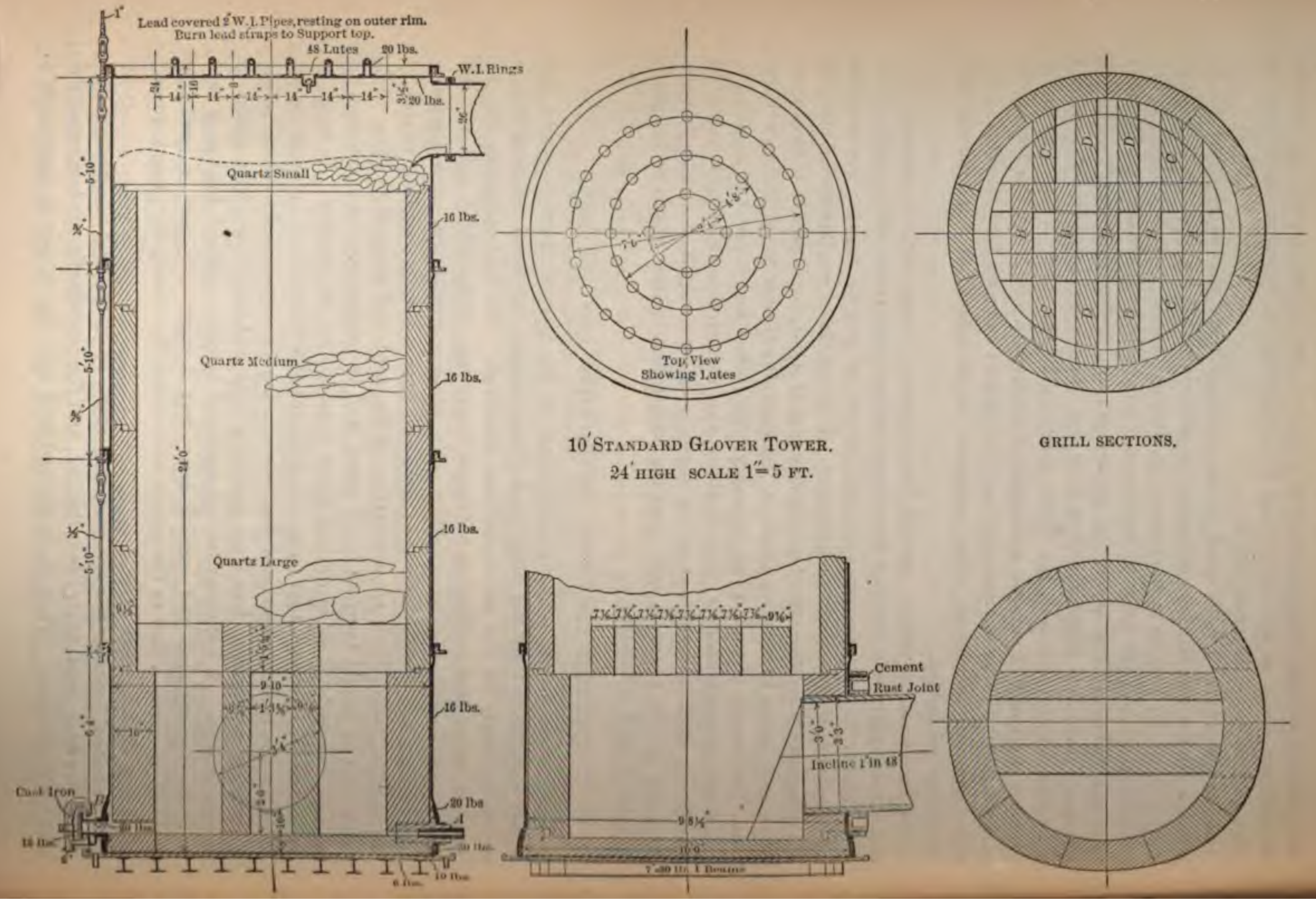
Of course if the proper draft of the furnaces were the only object to be attained there would be no objection to securing it by a stack with proper damper control, just as in the case of the furnace under a steam boiler, but the conditions are not the same by any means, for between the stack and the furnaces is interposed the absorbing apparatus, and in this apparatus a chemical process is going on which requires time and conditions entirely different and of its own kind. Further, the draft produced by the stack is not only unreliable on account of varying atmospheric conditions, but is rendered more so, in the majority of cases, inasmuch as it is used in common for boiler furnaces, still furnaces, and many other purposes which affect its temperature and regularity of action.

The furnaces, therefore, should be capable of being drawn mechanically entirely separate and distinct from the absorbing apparatus, and in such a way as to deliver to that apparatus a steady stream of gas of regular composition independent of atmospheric conditions. This condition can be effected cheaply and completely by the use of a properly constructed fan, such as I have used for several years in my own practice, between the Glover tower and the chambers. By means of such a fan, properly constructed and applied, it is possible to adjust instantaneously the draft on the furnaces to meet any required conditions, such as variations in the quantity of ore charged or a change in the character of ore charged, and when the adjustment is once made to maintain it steadily and constantly.

CHAMBER AND TOWER CONSTRUCTION AND OPERATION.

THE GLOVER TOWER.—The construction of the Glover tower requires and repays care more than any other part of a sulphuric acid plant. The system is shown typically by the accompanying drawings. The best materials for the interior lining and packing of a Glover tower are undoubtedly lava for the lining and quartz for the packing. No other material will give anything approaching the results both as to acid and heat-resisting qualities, purity of the resulting acid, high concentration of the acid and longevity. The additional cost is repaid fourfold in all cases, and in cases where concentration of the acid beyond 62° B. is desired these materials are imperative. It must be remembered, however, that there are different qualities both of lava and quartz and only the best close-grained flawless lava should be used and massive quartz of vitreous appearance free from cleavage lines and admixture with schist or foreign matter. A little pyrites, or oxidized pyrites, will not hurt as it will soon be decomposed and carried away, but it must not exist in sufficient quantity to cause a flaw in the quartz or to cause the quartz to "break down" when it is decomposed.

If chemical brick or tile are used they must be of the best selected quality. No matter how good, their life is limited, and they constantly contaminated the acid with alumina. When it is remembered that the tower is practically a



10' STANDARD GLOVER TOWER.
24' HIGH SCALE 1" = 5 FT.

GRILL SECTIONS.

large boiler, in different parts of which acid up to 62° B., or even as high as 93·5% H_2SO_4, is brought into contact with gases having a temperature of from 600° to 1200° F., and in the presence of nitrogen oxides and steam, and where the acid in presence of these highly oxidizing gases is actually flowing over packing material, it can readily be seen that too much care cannot be exercised in the choice of materials.

The size and shape of the tower must largely depend upon the demands intended to be made on it and on the means taken to create a draft through it, whether high concentration is desired or high oxidation. In any case the tower must be so proportioned as to absorb in concentration nearly or all of the heat produced by oxidation in the burners, and to prevent this heat from passing into the chambers. The temperature at the exit must be sufficient to prevent the condensation of any considerable proportion of the steam formed in the lower part of the tower. Towers of greater height than 18 ft. to 25 ft. (according to the volume of gas they are designed to handle) are unnecessary, and while higher towers may on account of the cooling of gas produce more acid owing to condensation, it can only be at the expense of the concentrating efficiency of the tower. The shorter tower will supply the chambers with gas in proper condition for the formation of acid there and it is the legitimate function of the chambers to condense this acid just as concentration and not condensation is a legimate function of the Glover tower.

The Functions and Management of the Glover Tower.—The functions of the Glover tower are: (1) The denitration of the nitrous vitriol. The nitrous acid gas absorbed in sulphuric acid of 60° B. to 62° B. (141° Tw. to 150° Tw.), and recovered thus by means of the Gay-Lussac tower from the chamber process, is here liberated and again put into circulation or made available for the chamber process by means of dilution and the action of the hot sulphurous acid gas in the Glover tower. (2) To supply the chamber process with nitrous acid gas lost during that process. When such loss is supplied by "potting" nitrate of soda in the hot gas just before the entrance of the gas into the Glover tower, then the Glover tower converts the nitric acid gas into nitrous acid gas and secures its thorough admixture with the sulphurous acid gas. When, however, such loss of niter is made good by the addition of nitric acid, nitrous vitriol, or waste nitrous acid of any kind to the nitrous vitriol from the Gay-Lussac supplied to the Glover, such acids are acted upon in the same way as the regular supply of nitrous vitriol, and this is by far the most efficient, direct and most easily controlled method of supplying the chamber process with nitrous acid gas. (3) Concentration. The nitrous vitriol to be properly denitrated must be diluted. The weak acid or water thus added is evaporated by the heat of the hot burner gas and converted into steam which, passing into the chambers, supplies a part of the steam necessary for that process, and the acid thus concentrated to 60° B., or 62° B., is again ready when cooled for reabsorbing nitrous acid gas in the Gay-Lussac tower. (4) The cooling of the furnace gas or the absorption of the heat of combustion in the furnaces. This heat if allowed to pass into the chamber system would, when added to the heat of oxidation of the SO_2 to SO_3, be detrimental not only to the process but also to the leaden chambers. (5) The purification of the furnace gas from impurities mechanically carried over from the furnaces. A considerable deposition of arsenic and selenium takes place in the Glover tower and except in extreme cases nearly all the dust of oxide of iron and unburned pyrites is also retained, being washed out of the tower with the acid either in solution as sulphates or in suspension. (6) The oxidation of SO_2 to SO_3 in the highly nitrous atmosphere of the upper part of the tower, thus preparing the gas for rapid formation into sulphuric acid in the chambers.

Value of the Glover Tower.—The Glover tower is usually regarded as the natural corollary of the Gay-Lussac tower and nothing more. The nitrous acid gas absorbed by the Gay-Lussac tower and recovered from the chamber process is again made available by the Glover tower for use in the chamber process and the denitrated acid is concentrated for use in the Gay-Lussac for again absorbing nitrous acid gas. Incidentally the furnace gas is cooled and purified. The value of the Glover tower as an apparatus for completely utilizing the waste heat of combustion of the ore, however, has not received sufficient attention. It may be safely stated that the heat produced by the oxidation of most ores used for making sulphuric acid is sufficient if properly conserved to concentrate the acid produced to commercial oil of vitriol (93·5% H_2SO_4). The difficulties in the way of such high concentration are purely mechanical and are as follows: (1) The conservation of the heat in the burners and flues leading to the Glover; (2) the proper proportioning of the Glover and its suitable construction involving the use of sufficiently refractory materials; (3) the purification of the concentrated acid; and (4) the reconciliation of the functions of high concentration and proper denitration in the Glover tower.

As will be shown in a subsequent paper on the concentration of sulphuric acid, none of these objections is by any means insuperable and the Glover tower may be made to concentrate the acid to the extent of the almost complete utilization of the waste heat of the furnaces, at an extremely low cost. One of the great advantages the Glover tower possesses as a concentrating apparatus is the total absence of losses as "weak distillates," all such distillates being carried over and recovered in the chambers.

For purposes of the manufacturer of acid of 50° to 60° B. only and for towers of ordinary construction the limit of concentration in the Glover tower may be taken as 62° B. (150° Tw.). The tower will readily concentrate to this strength more acid than is required for the Gay-Lussac towers. This surplus can either be used as such or diluted with acid which is weaker than required for any special purpose, or if there is no such supply of weak acid it may be diluted with water to the required strength. In case such acid is too impure for the purpose required it may be purified by sedimentation or filtration. It can also often be cleared up sufficiently by the addition of small quantities of potassium chlorate dissolved in water.

The temperature of the gas entering the Glover tower where no unusual precautions have been taken to conserve the heat is from 600° to 800° F. The temperature of the gas leaving the tower is from 190° to 200° F. As the heat produced by the oxidation of the SO_2 to SO_3 is about 200° to 210° F. in the zone of greatest activity it is evident that the heat of combustion in the furnaces has been nearly if not quite absorbed by the Glover tower, and the heat in the exit pipe of the Glover and the first part of the chambers is entirely due to the oxidizing processes going on in the production of sulphuric acid.

Construction.—Formerly when Glover towers were built from 40 to 50 ft. high much condensation of aqueous vapors must have taken place in the upper part of the tower itself. For this reason these lofty towers probably made more sulphuric acid than the modern towers of from 20 to 25 ft., the upper part of the tower performing the functions of a chamber to some extent. In the low modern towers the steam is passed directly to the chambers and the acid is mostly condensed there, which is proper. In the construction of a Glover tower its dimensions must bear a rational proportion to the volume of gas which passes through it, so that there may be no obstruction to the draft, and also to the heat units produced by the combustion of the raw material used in the furnaces. However, 25 ft. appears to be the limit of height necessary and in most cases from 18 to 20 ft. is ample height. This corresponds to a height of actual packing of from 10 to 17 ft.

Operation.—In starting up the Glover tower, simultaneously with the admission of the burner gas, nitrous vitriol with an admixture of chamber acid or water is supplied to the luted holes in the cover of the tower. At first the heat of the burner gas will be absorbed by the packing and lining of the tower and it will not be able to accomplish much in the way of concentration. There may also be no supply of nitrous vitriol available. The nitrous acid gas is therefore supplied to the system entirely by means of "potting," or if available by running nitric acid down the tower, and the strongest chamber acid at hand is used. In any case the nitrous vitriol is only supplied with just sufficient weak acid or water to secure its denitration in presence of the SO_2 in the tower, and only enough must be run down to keep the tower moist and to approximate the desired strength, as nearly as possible, as it flows from the tower. Gradually as the tower grows hotter it will reach its full concentrating capacity and the flow of weak acid will be as gradually increased until there is a good stock of acid of from 60 to 62° B. for the Gay-Lussac and a good stock on hand of strong nitrous vitriol.

Degree of Denitration.—When the tower is in full working order it should denitrate to a trace of N_2O_3 sufficient strong nitrous vitriol in 24 hours to be equivalent to 15 or 20% of nitrate of soda calculated on the actual available sulphur burned in 24 hours. The nitrous vitriol should be from 60° to 62° B. and should contain from 2·5 to 3% N_2O_3. The niter contained in this quantity of nitrous vitriol may be called the niter in circulation, that is, the stock of niter constantly being liberated by the Glover tower for use in the chamber process and as constantly recovered by the Gay-Lussac tower. Since from 2 to 3% of this 20% stock of nitrate of soda in circulation is constantly and apparently unavoidably lost in the process, or in other words since the Gay-Lussac towers fail to recover about 2 to 3% of the niter liberated by the Glover, it becomes necessary therefore to supply the system with this lost niter—and this is done as constantly and regularly as possible, preferably by means of additional nitrous vitriol or nitric acid supplied to the Glover tower; otherwise by means of "potting." Since nitrous vitriol containing say 2·80% N_2O_3 contains the equivalent of 6·22% $NaNO_3$ pure, or about 6·5% commercial nitrate of soda, it will be necessary for every 100 lb. of actual available sulphur burnt in 24 hours, to run over the Glover tower 308 lb. of nitrous vitriol, containing 2·8% N_2O_3, or a quantity equal to about four-fifths of the daily make of acid calculated as 60° B.

THE CHAMBERS.—From what will appear to be the functions of the chambers, it is evident that in considering the construction of any given set of chambers, the question of the total number of cubic feet required is not the only one, but also the question as to the division of the total chamber space in such a way as to increase its efficiency, *i.e.*, to proportion the chamber space into leading, second and third chambers. By proper attention to such arrangement and by securing a regular and suitable constitution of the chamber gas 11 to 12 cu. ft. of chamber space to the pound of sulphur burned as pyrites will be sufficient in plain chambers without Lunge columns. With these columns added, a considerable further reduction can be secured. Just what this further reduction is does not seem to be quite settled yet, but certainly the chamber space can be reduced 20 to 30% or to 8 or 9 cu. ft. per pound of sulphur.

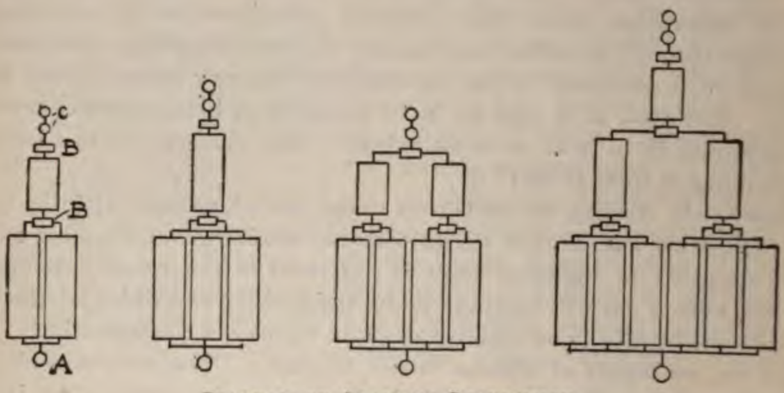


DIAGRAM OF CHAMBER SUBDIVISION.

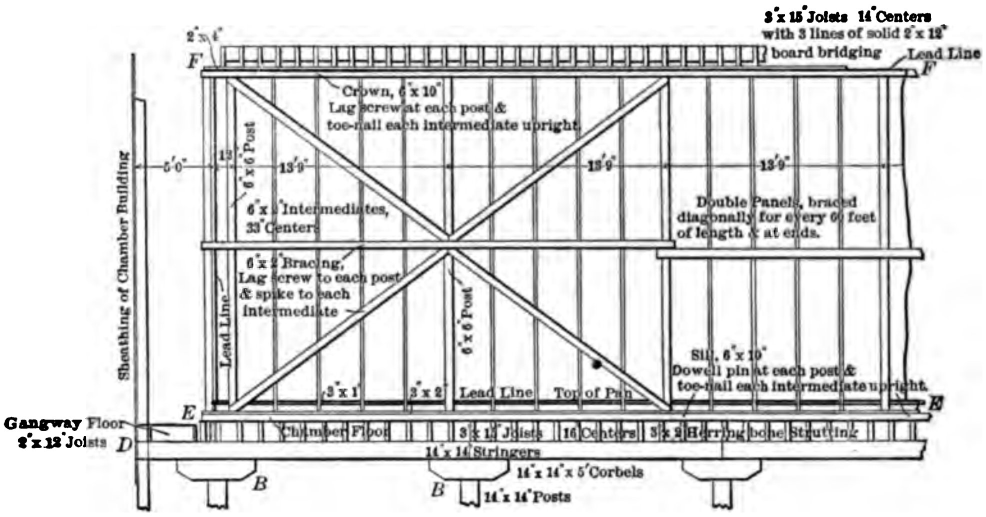
A, Glover Tower. B, Intermediate Column. C, Gay-Lussac Tower.

Example: Supposing the total cubic feet required are 100,000 and the sectional area is 520 sq. ft. (26×20 ft.) Then 100,000÷520=193 ft. (nearly), and the system would be divided into three chambers about 110 ft., 55 ft., and 30 ft. respectively. If 400,000 cu. ft. are required and the sectional area is 720 sq. ft., (30×24), 400,000÷720=550 ft. (nearly), or 320 ft. for the leading chamber, 160 ft. for the second and about 75 ft. for the third, which would be subdivided into say, four leading chambers (80×30×24 ft.), two second chambers (80×30×24 ft.), and one final chamber (75×30×24 ft.).

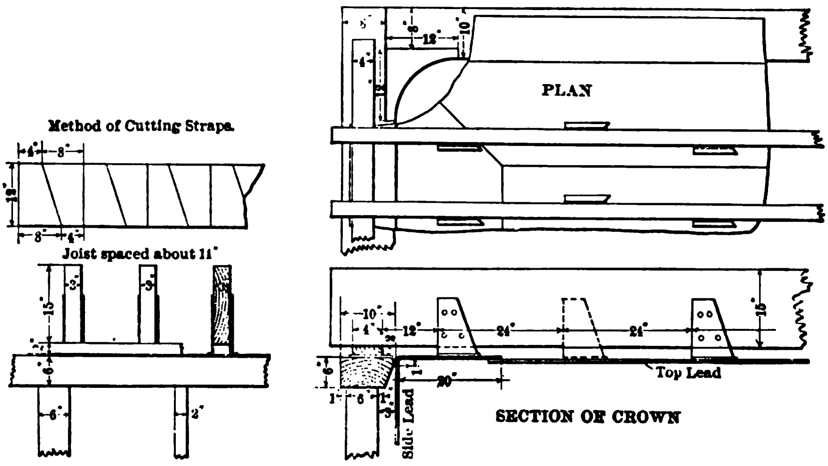
Division of Chambers.—The proportion between the chambers is approximately as 4:2:1 for the leading, second and third. To make the division practically, divide the total cubic feet required by the sectional area, which will give the total length of chambers. Then proportion this length as 4:2:1.

and if the cubic capacity is so large that the first chamber will be over 100 ft. long, then divide such length into two chambers or more as shown in the accompanying diagram.

Use of Lunge Columns.—If Lunge columns are placed between the leading and second and between the second and third and before the Gay-Lussac towers, the second and third chambers can be largely reduced in size if not actually done away with, or in other words, the three towers one after the other will take the place of the last three-sevenths, or up to 42% of the total chamber space, providing that all the details of the plant, especially the Lunge columns and Glover and Gay-Lussac towers, are properly proportioned and



ELEVATION OF CHAMBERS.
Scale: ¼ in. = 8 ft.

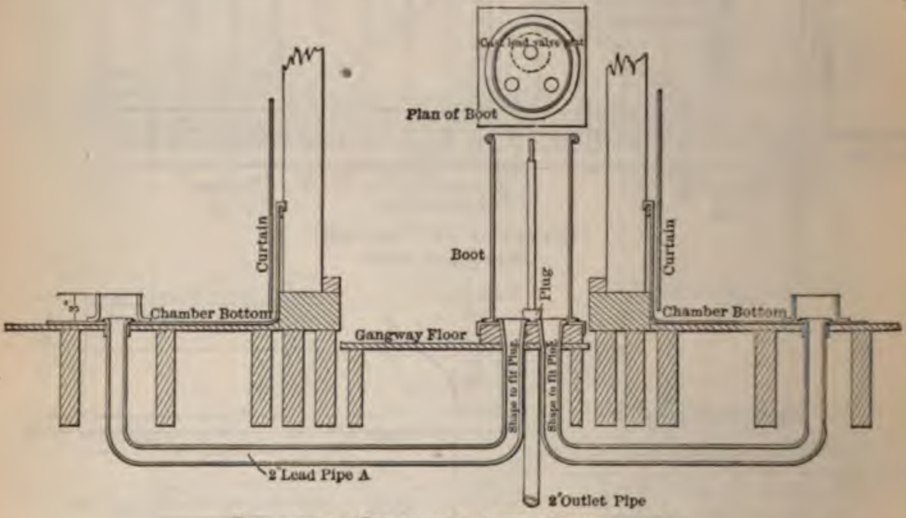


DETAILS OF CHAMBERS.
Scale: 1 in. = 3 ft.

the chamber gas is kept uniform as pointed out above; in other words, providing the system is carefully managed, in the way indicated throughout this article, more especially as regards the management of the Glover and Gay-Lussac towers and the nitrous vitriol.

Sectional area depends more on the size of the plant and considerations of economical construction than on its influence on the process; that is to say, within reasonable limits. Out of the scores of arbitrary and fantastic sections, (1) 30×24 ft.=720 sq. ft., and (2) 24×20 ft.=480 sq. ft., are recommended for ordinary use. Under certain circumstances sections (3) 40×26 ft.=1,040 sq. ft., or (4) 20×18 ft.=360 sq. ft. might be used.

In any given set of chambers it is hardly advisable to vary the sectional area, and certainly it is not necessary, and can seldom be done with saving of ground space, lumber or building economy. The question is merely structural and involves: The height, form, and cost of the building, especially the roof trusses; the length and cost of timber and its economical use in different localities; the cost of lead, the cheapest sectional area at equal cost of construction in other respects being that which will give the least lead area with the greatest cubic capacity; and the total cubic capacity desired in any one set of chambers, in small sets, for instance, a large sectional area being perhaps incompatible with the suitable division of the chambers.

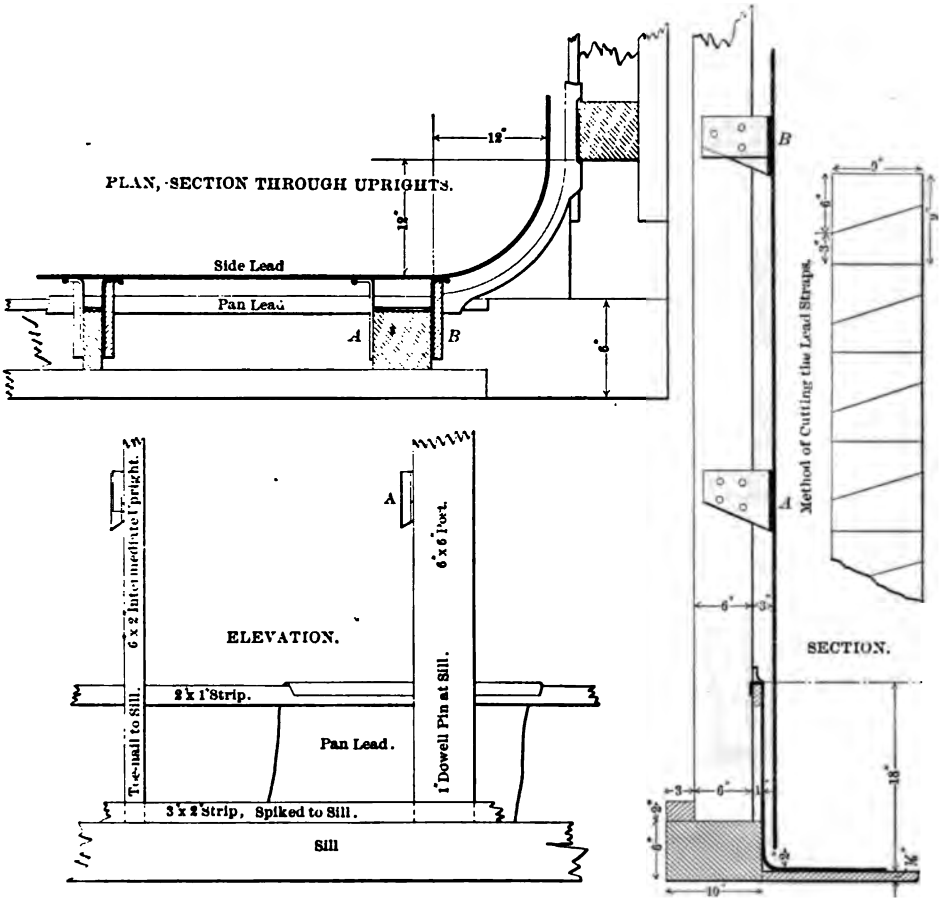


Details of Chambers; Acid Connections.
Scale: 1 in. = 2½ ft.

Details of chamber construction are shown in the accompanying engravings. The use of a cylinder of lead loosely set over each discharge pipe in the bottom of the chambers tends to keep a layer of 3 in. or 4 in. of cool acid on the bottom of the chambers and to prevent any scouring action of the acid on the chamber bottoms. Various forms of siphons are also used when it is not desired to have openings in the bottom of the chambers.

Functions of the Chambers.—The gas leaving the Glover tower consists of SO_2

with a large percentage of SO_3, N_2O_3, and water vapor. If the Glover tower be working efficiently the gas contains quite sufficient steam to complete the process for a considerable distance in the chambers. The amount of water vapor is directly proportional to the concentrating efficiency of the Glover tower, and is the water evaporated in concentrating the acid fed to the Glover to the strength at which it leaves the tower, and which may be calculated from the sulphuric acid tables appended.

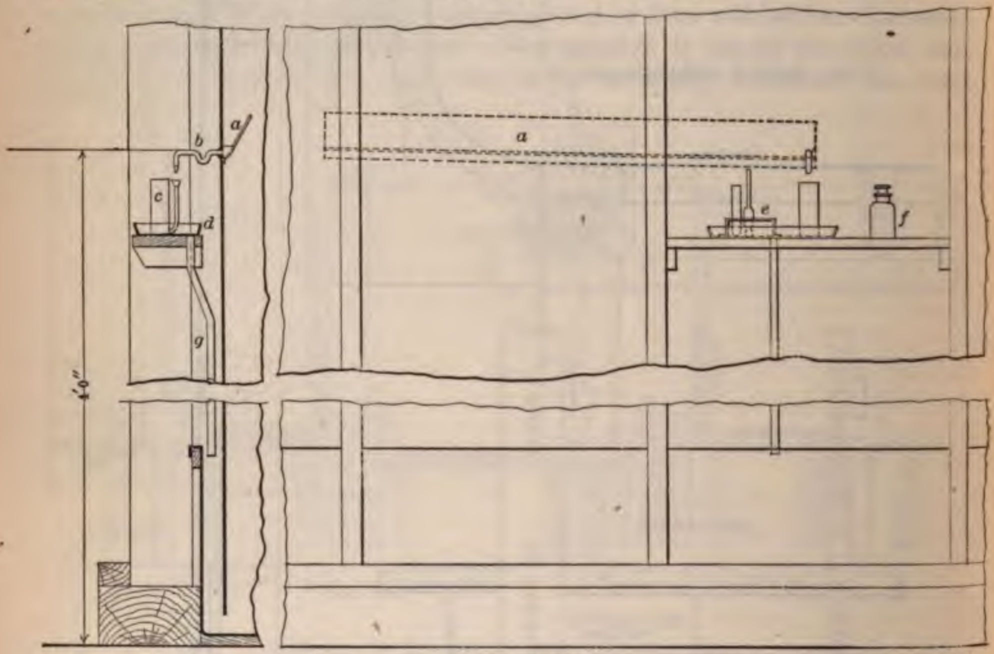


DETAILS OF CHAMBERS.

For chambers 20 ft. high burn four straps on alternate sides of each post and intermediate upright at heights respectively of 3 ft. 4 in., 6 ft 8 in., 10 ft. and 13 ft. 4 in. from chamber bottom. *A*, shows strap unfinished. Use four leaded nails 1·5 in. long and turn over 1 in. at the wider end for burning to side lead. Finish by spiking on a 0·75 in. board 4×6·5 in., as shown at *B*.

The functions of the chambers are: (1) To supply the gaseous mixture of SO_2, SO_3, and N_2O_3 with the necessary water vapor; (2) to afford the gas a space of comparative absence of motion and to give time for the various chemical reactions to take place; (3) to radiate the chemical heat produced by these reactions; (4) to afford surfaces to facilitate condensation of the sulphuric acid as it is formed and to receive and store such acid; (5) to mix the various gases, to completely convert the SO_2 into dilute sulphuric acid, and to dry, cool, and prepare the N_2O_3 for absorption in the Gay-Lussac towers.

Supply of Water Vapor and its Determination.—Since the Glover tower supplies the necessary water vapor for the first part of the process no steam should be added until the gaseous mixture requires it, or in other words, until this initial supply is exhausted. This will be shown by the drips, which are of two kinds. Curtain drips are troughs of lead burnt on the inner surface of the lead curtain, or side of the chamber, terminating in a luted pipe on the outside of the chamber. This trough collects the acid forming on the surface of lead above it and passes it outside into a "drip cup," where it can be tested by the hydrometer and steam added or taken from the chamber according to the hydrometer reading. Table drips are pans of lead set well within the chamber and away from the side (and connected by a lead pipe with the exterior of the chamber similarly to curtain drips). The rain of acid forming above the pan is thus collected and tested by the hydrometer.



CURTAIN DRIP.

(*a*) Curtain trough (4 ft. long); (*b*) luted pipe (¾ in.); (*c*) hydrometer jar (6 x 1 in.); (*d*) lead pan (6 x 12 in.); (*e*) hydrometer and test tube stand (3 holes); (*f*) solution of ferrous sulphate; (*g*) drip pipe (¾ in.).

Since on account of the radiation of heat from the lead sides of the chambers, they are always cooler than the interior, more water vapor condenses on the sides than in the interior, and the acid collected by curtain drips is always weaker than that of table drips. This difference is about 3° B. A curtain drip reading of 52° B. would represent a table drip reading of about 55° B. and vice versa. The amount of aqueous vapor contained in the gas can also be very well obtained by means of a luted pipe in the connections between the chambers.

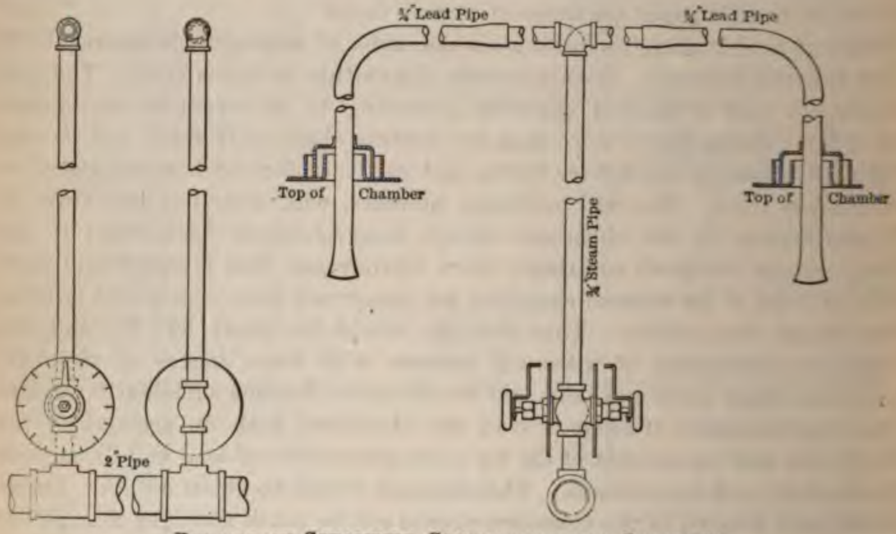
Sometimes the area of the pan and the proportion of length of trough to area of curtain drained by it are so arranged that the acid passed by them to the outside of the chambers forms some indication of the activity of the process, but this is inaccurate and useless. The only requisite is to have these drips sufficiently large to supply acid for observation purposes rapidly enough. Such drips should not be placed in immediate proximity to the points of admission of steam, but the distance of the drips should be constant in any one set of chambers, so that the observations which they afford may be of uniform conditions. If in one case they are 20 ft. forward of a steam jet, this distance and location should be uniform throughout the system. On account of structural difficulties table drips are rarely used, and consequently strength of drips referred to in this paper are those of curtain drips.

Different acid makers prefer to keep the drips of different chambers at different degrees Beaumé. Within certain limits this is immaterial. The first chamber, or part of the first chamber, according to its strength, or the first part of the process where more than one leading chamber is used, will be supplied with steam by the Glover tower, and will therefore be beyond control of the chamber man. The acid produced, however, will rarely fall below 52° B. The acid formed in the chambers should, however, never be allowed to get strong enough to absorb and retain more nitrous gas than is absolutely inevitable, at least so far as such chambers are concerned from which acid is withdrawn from the system. This strength would be about 52° B., and the tendency to absorption of niter will increase with every degree of strength. Nor on the other hand must the acid be allowed to become sufficiently weak to permit the formation of nitric acid in the chambers, both on account of the loss of niter and on account of the far more serious loss of lead and damage to the chambers and connections. This strength would be about 45° B. Therefore the acid formed in the chambers should not be much stronger than 52° to 54° B., nor weaker than 44° to 45° B.

Although the drips are highly useful adjuncts in controlling the chamber process, samples of the bottom acid should also be taken at proper intervals, and in each individual set of chambers the acid maker must refer the drip readings of the curtain drips to the actual strength of acid formed in said chambers, as shown by experience. A curtain drip reading of 50° B. means nothing unless it conveys to the acid maker the information gained from experience and frequent tests that this reading at this especial drip or set of drips means that the acid formed in the chamber in question is of a certain relative strength.

Admission of Steam to the Chambers.—The supply of aqueous vapor to the chambers is almost universally in the shape of steam at low pressure. From 10 to 40 lb. is a fair variation according to circumstances, 20 lb. being a good average to use. Exhaust steam may be used either wholly or in connection with live steam, but it should be kept well drained, so as not to inject occasional streams of water into the chambers.

It seems to be generally thought best to admit the steam from the roof of the chambers. Sometimes it is admitted at the connections between the chambers and in such a way as to act as an injector in case of insufficient draft. The principal consideration is to admit it in such a way that it will mingle most intimately with the gas in the chamber and remain in suspension until its function is consummated. Very good results are obtained by admitting the steam on the center line of the top of the chambers at points about 25 ft. apart and equidistant from one another and the ends of the chamber. Thus a chamber 100 ft. long would have the means of admitting steam 25 ft. from the entrance of the gas and at points 50 ft. on the center line of the chamber and 75 ft. or 25 ft. from the end of the chamber. In very wide chambers instead of placing the jets on the center line two jets abreast should be used. It is better to have too many rather than too few points of admission, reducing the quantity of steam admitted by the individual jets or reserving some jets for use in emergency or as may be found most advantageous. In starting up any set of chambers there should be the means of supplying steam at the entrance of the gas to the first chamber until the Glover tower is working efficiently. The various details connected with the supply of steam to the chambers are shown in the accompanying drawings.



DETAILS OF STEAMPIPE CONNECTIONS FOR CHAMBERS.
Scale: 1 in. = 1 ft.

For a given charge of sulphur, producing a constant supply of gas of a uniform percentage of SO_2, with a Glover tower supplied with a steady stream of nitrous vitriol and nitric acid of constant specific gravity and constant nitrosity, a constant supply of aqueous vapor will be required. Therefore after the steam supply is once regulated and means are taken to secure its admission at uniform pressure it will not require to be changed, and the fact of such change becoming necessary is *ipso facto* an alarm that something is wrong which should be found out and rectified at once.

Retardation of Current.—Considered as a reservoir of gas where the current is checked in order to afford time for the necessary chemical reactions, it is evident that the chambers should not be regarded as part of a flue and that any tendency to a direct current through the chambers should be avoided in construction. In no case should the same gas entering the chamber pass through and out as in an ordinary flue. In passing these gases will be diffused, partially condensed, reduced in volume, changed in chemical composition, in density, and in volume. The talk so commonly heard about "dead corners" is not rational. No such thing as a vacuum can exist nor in the case of gases of so nearly the same density is it likely that any layers or corners of inert nitrogen can be possible, although in the case of presence of carbonic acid such a condition might possibly arise. The passage of the gas through the chambers should be largely due to their diminishing volume owing to condensation and diminishing temperature and displacement by the incoming gas, and any attempt to force the gas through this form of condensing apparatus at any rate will only result in disaster when the Gay-Lussac towers are reached. The draft at the exit of the last chamber must be simply sufficient to remove the inert nitrogen and the oxides of nitrogen as they become sufficiently free from SO_2 and dry and cool enough to be absorbed in the Gay-Lussac, but before the formation of NO or N_2O_4, which cannot be absorbed. The last chamber can even be worked at a slight outpressure. In this respect the chamber space is analogous to a lake, with a slight current at the inlet and a slight current at the outlet and a great neutral zone of no apparent current.

Temperature Determinations and their Importance.—A consideration of the work done by the furnaces and Glover tower in concentrating acid will show that the heat derived from the oxidation of the sulphur and in the case of pyrites of the iron is almost if not entirely absorbed by the Glover tower. The heat of the gas, therefore, passing from the Glover tower into the chambers is largely if not entirely due to the oxidizing processes going on in the upper part of the tower in the presence of the liberated nitrous acid. This is shown in the case of the Freiberg chambers quoted by Dr. Lunge (2d Ed., Vol. I., p. 473), "where the cooling of the burner gas, in order to condense the arsenious acid, is carried on so far that the temperature is that of the outer air, and in winter the gases often enter the chamber at 0° C. (32° F.). In spite of this no inconvenience is felt and no special contrivances or precautions need be employed; the temperature within the first chamber soon rises to 60° C. (140° F.) and upward."

The rapid rise in temperature which an increase in the nitrous acid gas supplied to the chambers will cause also evidences this condition. If the leading chamber of a set is a small one it is possible to see the thermometer rise from 170° to 200° F. and over, in a few minutes, on an increase in the supply of niter. This being the case, it is evident that the temperature of a chamber is a rapid, sensitive and altogether valuable index of the activity of the process going on in any chamber. This is so much so that if any two points at sufficient distance apart be supplied with thermometers and the readings of these thermometers be taken at a time when the system is doing its best work, a variation of 2° F. in the difference of these two readings is sufficient indication of coming trouble to set a careful acid maker looking for the cause at once, and if such cause is found and remedied in a reasonable time, this slight variation of temperature may be the only indication that such trouble ever existed.

When chambers are working at maximum efficiency this initial temperature, registered by a bent thermometer inserted in the side of a chamber about 6 ft. from the floor and about 30 ft. from the entrance of the gas, will register about 200° to 210° F. In the side of the second chamber it will register from 150° to 160° F. A deviation of 2° from this difference of say 50° F. is by far the most sensitive and the earliest indication of a fault somewhere in the continuity and regularity of the process. It may be the strength of the burner gas, the strength or quantity of the supply of niter or acid to the Glover tower, the pressure or condition of the steam; but somewhere the fault exists and if it be not remedied trouble will come. By the time the gas is ready to leave the last chamber its temperature should be not over 80° to 90° F., varying according to the outside temperature.

Radiation of Heat from Chambers—After the oxidation which causes this heat has taken place, the more rapidly the heat is carried away the better. In chambers this is done constantly and regularly but more or less imperfectly by radiation from the surface of the lead. In construction nothing should be done which will not facilitate this radiation. The outside of the lead should be as free from timber framework or other non-conducting surfaces as possible and all chances of access of air in motion should be given around the chambers, not only on account of the process but also to prevent corrosion of the lead itself and the rapid destruction of the chambers. Many devices to carry away this heat more rapidly, and thus to increase the rapidity of the condensation of sulphuric acid mist, have been introduced. Undoubtedly the Lunge columns when properly erected are the most efficient and most scientific. Towers are also used filled with earthenware rings, quartz, etc. The Gilchrist columns built with air-cooled pipes are designed to bring comparatively thin strata of gas into contact with surfaces cooled by radiation. Long pipe connections between the chambers and especially between the last chamber and the Gay-Lussac towers are also very efficient. Whether the most efficient means of promoting surface radiation of heat and surface condensation of the minute vaporous particles of sulphuric acid in the chamber process has been arrived at, structurally considered, I think open to doubt; certainly the method of application of this means of getting the already formed acid out of the way is not often understood, and these towers are often placed where they will do more harm than good and are so proportioned as to obstruct the passage of the gas unnecessarily and interfere at a time when the gas requires rather to be left alone. The only possible function they can perform is to condense the sulphuric acid mist after it is formed and radiate the heat of formation. If it were possible to reduce the temperature of the first chamber to freezing point it would simply mean that no chemical action was going on, *i.e.*, no sulphuric acid was being made.

Admixture of the Gases.—In order to mix the various gases and to differentiate the various stages of the chamber process, the chamber space is generally divided into several sections. The process is at its greatest intensity immediately after entering the chambers, and this intensity is sustained for a distance in the chambers varying with the volume of gas admitted. After a certain distance, generally not exceeding 100 ft., the temperature and formation of acid diminish sharply. Up to this point in the Glover tower and chamber fully 70% of the SO_2 has been converted into sulphuric acid.

By causing the gas to pass through a comparatively small connecting pipe or pipes into a second chamber, the intensity of the process is greatly increased for a time. In this second chamber the bulk of the remaining SO_2 is formed into sulphuric acid, about 25% being gained. On passing the gas into the third chamber the last of the sulphuric acid is formed and in this chamber the nitrous gas is prepared for absorption in the Gay-Lussac. The principles underlying the division of chamber space into separate chambers therefore should be governed by the requirements set forth below.

Chamber Subdivision.—The division should be made approximately proportionate to the activity of the process. This is usually recognized in small sets of chambers, ranging from about 100,000 cu. ft. capacity to 150,000 cu. ft., where usually three chambers are used having the approximate ratio of 5 : 3 : 1. Such a set of chambers fairly well represents the requirements of the case. In the first chamber there is intense chemical activity, producing with the Glover tower upward of 70% of the whole formation of acid. In the second chamber there is renewed but much slower action, practically completing the formation of acid, while the third chamber simply dries the remaining gases and prepares the nitrogen oxides for absorption in the Gay-Lussac towers. But in larger systems, where four, five and even six sets of chambers are used tandem and sometimes of extreme length, the construction is hardly rational. To accommodate the greater volume of gas the first two or three chambers and indeed sometimes all the five or six are made of great length, even up to 200 ft., and in these chambers the chemical activity grows less and less, except where at the beginning of each new chamber it is more or less stimulated, until finally by sheer space and distance the sluggish process is completed, always in presence of the danger of loss of niter and great corrosion of the final chamber and connections, either by formation of nitric acid or by higher oxidation of the N_2O_3, which accidents do in fact occur.

A little consideration of the mechanics of the process will probably lead the way to an amelioration of these conditions. Suppose that for a given volume of gas entering the first chamber from the Glover a length of travel in any section of chamber of say 80 ft. is sufficient to condense say 60% of the sulphuric acid. Omitting consideration of the aqueous vapor, which may be held as remaining constant throughout the active part of the process, the gas, which upon entering contained approximately 8% SO_2 to 82% N, at a point 60 ft. beyond the inlet contains 3% SO_2 or less to 90% N or more. In other words for every foot of forward travel in the chambers the gas will contain a smaller percentage of SO_2 and a larger percentage of absolutely inert gas. No forcing of larger quantities of fresh gas through the same chamber space can prevent this constantly increasing dilution; the inert nitrogen steadily increases the moment the sulphuric acid begins to form in the chambers. At the same time the volume of the gas becomes less both on account of reduction in temperature and condensation, and the reduced percentage of SO_2 becomes more and more diffused, and the formation of sulphuric acid more and more sluggish, and the risk of formation of undesirable nitrogen oxides more pronounced.

The division of the chambers should also be proportionate to the volume or density of the gases. To secure such proportionate division it would seem rational to pass the gases as they enter the chamber system, and when they contain the maximum proportion of active to inert gas, and are at maximum expansion, either into a chamber of large sectional area and of such a length as the temperature line shows to be of greatest efficiency (although this method is not altogether satisfactory since it disregards the necessity for rapid radiation of the chemical heat), or to subdivide these gases into a number of leading chambers of ordinary sectional area and of a length approximating the greatest chemical activity. Such length will vary with the size of the system from 60 to 80 ft. or possibly 100 ft.

Such subdivision of the gas is of course easily and positively obtained by the use of fans, and I have used it very successfully in my own practice. We then have at the exit of two, three, four or in case of very large systems as many as six leading chambers, each of usual section and each from 60 to 100 ft. long, exactly the same conditions which exist in the leading chamber of the small set previously mentioned, which are (*a*) from 60 to 75% of our acid already made and condensed or ready to be condensed, and (*b*) gases containing a much smaller percentage of SO_2, much more diffused, and of much lower temperature of actual oxidation but containing much heat from the zone of previous activity of oxidation. These gases, therefore, are ready to be freed from acid already formed, cooled, mixed, and brought into more intimate contact, and after such cooling and mixing to be passed into chambers of much smaller sectional area.

Condensation of Acid Mist.—It is at this point that the great advantage of the Lunge and other forms of cooling and mixing apparatus is felt. By passing the gas at such point through such apparatus the mist of sulphuric acid which would otherwise float for a long time is removed from the process by condensation and the accumulated heat is also largely taken away, leaving the remaining SO_2 to the increased action of the N_2O_3. But such an apparatus, whatever its nature, must not interfere with the passage of the gas, and should also be of such capacity as not to unduly accelerate the speed of the gas current. It should contain the largest possible area of radiation surface. I do not think it is necessary to run acid or water down these towers, at any rate unless under careful observation. A properly constructed tower at this part of the system will work to great advantage when used dry, or when, as is the case with the Gilchrist columns, in towers packed with quartz and rings, etc., surface radiation is largely depended on. It will also be found advantageous in entering and leaving any such apparatus to increase the length of the connecting pipes. In fact I should recommend at this point to erect a system of lead standpipes as gas connections similar to the cooling arrangements in gas works. These pipes could then be cooled by water or air. Or the gas might be made to pass a series of refrigerated lead coils.

At any rate, and by whatever means the acid mist has been condensed and the previously formed heat of the gases removed, it will plainly not be necessary to preserve the same sectional area of chamber space. This may be safely reduced by one-half. In other words, the gas from two leading chambers may be carried into one second chamber. If at the end of this second chamber another tower or Lunge column be placed with a long return pipe to the Gay-Lussac it will not be necessary to erect a third chamber, if the columns are of sufficient size and proper construction. If, however, a larger number of leading chambers than two are used I frequently prefer using a small third chamber and a final column common to the whole system.

The above conclusions, which have been carefully worked out in practice during the last few years, conform to Dr. Lunge's statement* that it has been proved "to be quite feasible to cope with the task of reducing the space for generating sulphuric acid at the point where the greater part of the sulphuric acid has been taken out of the gases together with a corresponding quantity of oxygen."

Irregularities in the Chamber Process.—If the chambers are supplied with a gas containing a constant percentage of SO_2, N_2O_3 and a sufficient and constant supply of steam at a constant pressure, the process will remain constant and regular, as will be evidenced by the drip, color of the gas, and above all by the temperature readings of each chamber. Such constancy requires only reasonable care, more especially when the loss of niter is supplied by means of the Glover tower and nitric acid. When it is supplied by the more or less intermittent method of "potting" there will be a constant ebb and flow of the active zones of chemical action, which will be accurately and immediately registered by the thermometers. The chamber system is usually large and elastic enough, with the last chamber acting as a cushion, to compensate for these irregularities up to the point of entry of SO_2 into the Gay-Lussac tower, if there is only one, or into the second tower if there are two. When this is the case decomposition of the nitrous vitriol sets in, with loss of N_2O_3 into the air, and consequently to the system; and the process is "lost" unless it can be held by replacing the lost niter at the beginning of the process. When using nitric acid, or a large reserve of nitrous vitriol, if taken in time the difficulty may soon be gotten over. When "potting" is practiced the difficulty is often a very serious one. The remedy lies in reduction of charges, in increase of niter, in readjustment of the steam admitted to the chambers, and requires the thoughtful care and immediate attention of the acid maker, not only to recover his process, but also to protect the plant from serious injury. Such heroic measures as supplying the system with 30 to 40% or more

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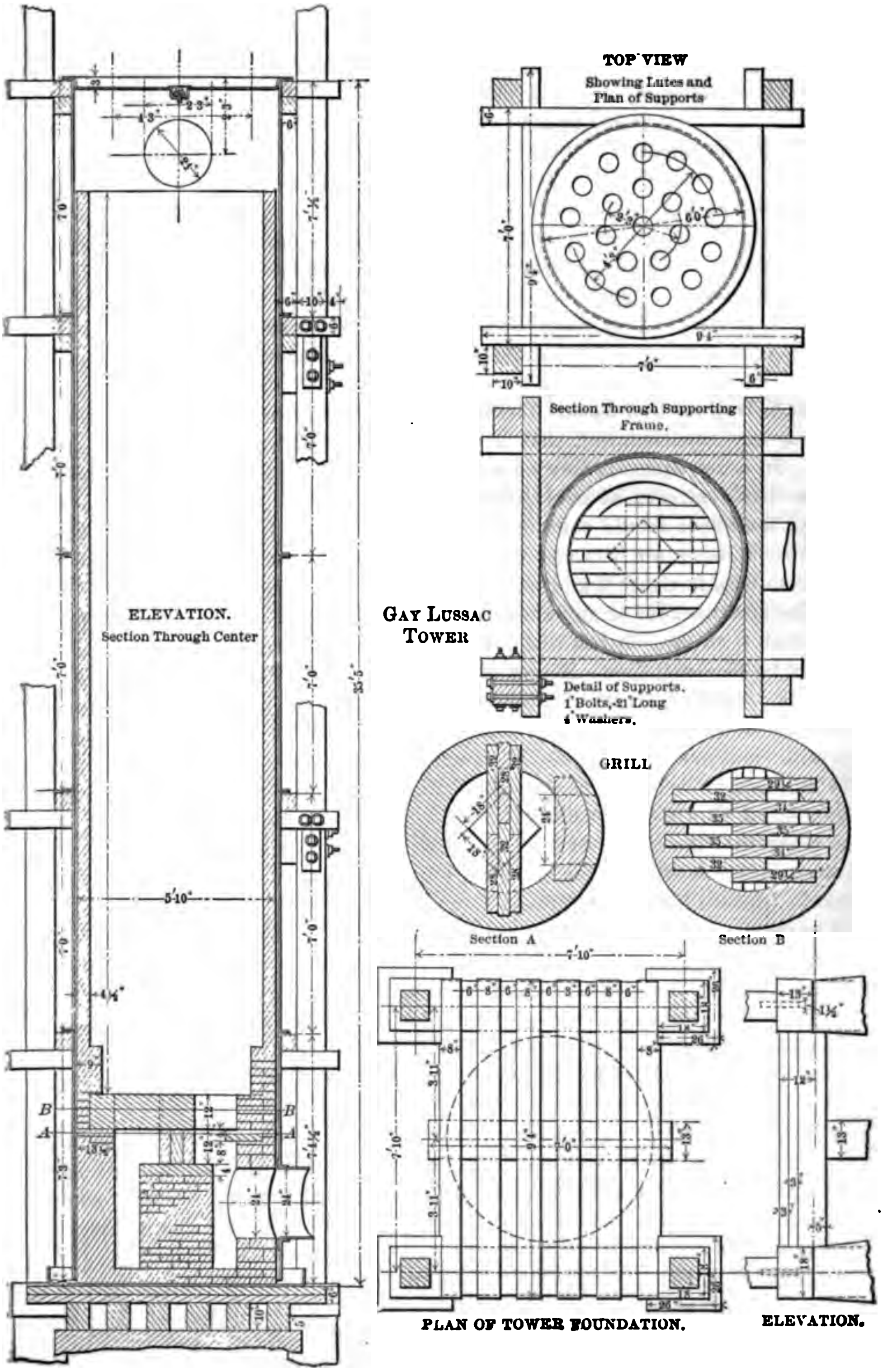
* THE MINERAL INDUSTRY, Vol. II., p. 129.

of niter are useless and most injurious. I have seen such work done when the chambers were full of aqueous vapor. Only one result can follow, and that is production of nitric acid, waste of niter, and great injury to the plant. The increased niter used should be just enough to make up for the deficiency in the nitrous vitriol owing to the loss at the Gay-Lussacs, just enough to put the gases entering the chamber into a normal condition, leaving the tail end of the system to clear itself in due course. Once the Gay-Lussac towers are lost, loss of niter and SO_2 are inevitable. Keep things going right in front and gradually matters will get normal at the end, until the Gay-Lussacs again absorb nitrous acid gas and the artificial supply can be gradually dispensed with. It need not be stated that, short of a breakdown in any of the apparatus, it is impossible to "lose" a system except by gross carelessness when the draft is positive, as is the case when using fans. With natural draft a heavy storm will sometimes draw the SO_2 into the Gay-Lussac towers, but dampers are usually provided, and it is the duty of the chamber man to attend to this point.

The points to be looked after in acid making are these, and it is the duty of some one to look after each of them, and his disgrace if his employer suffers loss by his carelessness: (1) Constant percentage of SO_2; the acid maker, as to the condition of the ore (sulphur contents, size, etc.), the weight of charge, the conditions of draft, the proper admission of air to secure the percentage of SO_2; the burner man, as to obeying instructions as to admission of air, proper charging and raking, shaking and discharge of cinder, and of the potting of niter if this method is practiced. (2) Constant percentage of N_2O_3. The regular work of the Gay-Lussac and Glover as to absorption of N_2O_3 and denitration and concentration. The regular supply of nitrous vitriol of uniform percentage of N_2O_3, and sufficient weak acid or water to secure its denitration, and the testing of the nitrous vitriol. The testing of the concentrated acid for density and nitrosity and its efficient cooling are usually looked after by the acid maker and the chamber man, with the help of the man pumping acid to the various towers. (3) The steam supply will also be attended to when necessary by the chamber man under directions from the acid maker.

Pressure Determinations.—In addition to the indications of the condition of the process afforded by the drips, and especially the thermometers, a manometer placed on the leading chamber about the zone of greatest activity (the first 20 or 30 ft.), and one placed near the exit of the last chamber, give very reliable assistance in acquiring information as to the tension of the gas in the chambers, and consequently of the condition of the process. These manometers should always be read by difference, which will always be the same under equal conditions. Having once learned the difference between the tension at the beginning and end of the system under the most favorable conditions, any deviation from this shows abnormal working of the process. A simple and sufficiently accurate form of this instrument, which can also be used for testing the resistance in any tower or plate column, is described in the section of this paper on instruments. If the resistance (the difference between normal pressure at the inlet and outlet) be observed when such tower



GAY LUSSAC TOWER

is known to be clean and in good working order, its subsequent fouling or partial stoppage for any reason can be readily detected.

THE GAY-LUSSAC TOWER.—The construction of the Gay-Lussac tower is shown in the accompanying engravings. The lining wall and arches or grills should be made only of good chemical brick, and where necessary these may be cemented by boiled sulphur or sulphur and clean sand. Coke is entirely inadmissible as packing material. Broken massive quartz forms a good packing, large pieces being used at the bottom and smaller at the top. Lunge (perforated) plates, chemical ware rings or small tiles laid transversely all form good packing. Quartz has one great advantage, however, and that is that it does not contaminate the acid with alumina. The towers should be placed in as cool and as easily accessible a position as possible, and for this reason do not require protection from the weather otherwise than from the direct rays of the sun.

Functions of the Gay-Lussac Tower.—Unlike the Glover tower the Gay-Lussac has only one function, and that is absorption of nitrogen oxides, principally N_2O_3. In order that this may be properly carried out, the tower must be fed with a steady stream of cool sulphuric acid of about 62° B. This acid on leaving the tower should contain about 2·5% of N_2O_3. In order that the tower may absorb this niter, the gas upon entering the tower must be practically free from SO_2, fairly cool and dry, and must not contain NO or N_2O_4.

If the final member of any chamber set is a Lunge or well-constructed tower or column used dry, that is without water or acid flowing down it, the gas will usually be cool and dry. Excess of NO or N_2O_4, as evidenced by the dark-red color, is caused by excess of niter in the last end of the process, or too sluggish a process in the rear end of the chamber system. Moreover, since in order to prevent the formation of these higher oxides of nitrogen, it is desirable to carry the sulphurous acid as near to the Gay-Lussac tower as possible without its entering this tower, where it would decompose the nitrous vitriol, it is necessary that the height of the Gay-Lussac tower should be considerable, so that any niter set free at the bottom by the introduction of SO_2 may be again absorbed before leaving the tower.

For the same reason and to avoid the expense of too high a tower, it is better to divide it into two of less height, and where several systems are near at hand the Gay-Lussacs of all may be combined. Any difficulty as regards the draft may be avoided most economically by the use of the fan. In case of new construction the towers may be of different diameters or they may be more loosely packed nearer the chambers, in this way taking advantage of the diminishing value of the gases.

Use of Two Towers.—When two or more towers are used the gas passes in at the bottom of the first tower and is taken from the top of it to the bottom of the second, and so on until it leaves the last tower at the top, where it should be discharged by means of a fan directly into the air. When this is done the system will be absolutely unaffected by outside meteorological conditions. On the other hand, the cool 62° B. acid is first run down the last tower, where it meets the gases weakest in N_2O_3, absorbing the last traces of N_2O_3, and is then pumped to the top of the next tower, or in a set of two towers (which is most usual) to that tower immediately after the chambers where it will absorb N_2O_3 until it reaches about 2·5% N_2O_3 and is ready to be pumped over the Glover tower.

The quantity of acid run over the Gay-Lussac towers is regulated by this final strength of 2·5% N_2O_3, enough acid being run over the tower furthest from the chambers to emerge from the tower nearest the chambers at 2·5% N_2O_3 after it has been successively run over any intervening towers. This quantity of nitrous vitriol at 2·5% N_2O_3 should be equivalent to about 20% of nitrate of soda ($NaNO_3$) on the sulphur burned. The regularity of this function depends on the (1) freedom of the entering gases from SO_2 and NO and N_2O_4, or in other words on the proper performance of the functions of the chambers, as mentioned above; (2) the proportion of the absorbing space and its efficiency; and (3) on the towers being supplied with acid of proper strength (62° B.) as cool as possible, which of course will depend on the temperature of the air and cooling water unless artificial refrigeration is possible.

As has been before stated, with all care it is not possible to maintain a rigid curve of activity in the chambers. There will be some ebb and flow in the chamber process, sometimes resulting in the admission of SO_2 into the towers. At such times the great advantage of two towers is evident, since any N_2O_3 escaping from the first will be absorbed in the second until the nitrosity of the acid leaving the second may be brought up to actual nitrous vitriol of 2·5% N_2O_3, the first tower in the meantime performing the functions of chamber or absorbing space to the extent to which the process has been allowed to "go back." The process will not therefore be lost until the SO_2 reaches the second tower, which should never by any possibility occur.

Capacity of the Gay-Lussacs.—It has been frequently stated that in order to perform their function in this way the proportion of absorbing space must be large when compared with the chamber space. I think, however, that this is rather exaggerated. Much, of course, depends on the construction and packing of the tower. Many old towers which have been packed with coke for a number of years do not perform 10% of their absorbing duty. The coke is broken down and often cemented together until it has to be broken with pickaxes to remove it, and the gas and acid simply travel through a few open channels, hardly being brought into intimate contact at all. In my own practice with a combined set of chambers aggregating nearly 700,000 cu. ft. and a consumption of actual sulphur burnt of over 55,000 lb. in 24 hours, two circular 8×50 ft. Gay-Lussac towers with quartz packing and fan draft were found ample (with an unusually long pipe connection to the last chamber), the acid flowing from the second tower about 1·5% N_2O_3 strong and from the first about 2·5% strong. The total consumption of niter averaged about 3% $NaNO_3$ on the available sulphur.

In another case a set of chambers of 100,000 cu. ft. was provided with a circular tower 30 ft. high, 4 ft. in diameter outside, much reduced by the brick lining, and in bad condition, coke packed, and its efficiency consequently very slight. This tower was repacked with quartz and a tower 8×40 ft.

quartz-packed was added, the old tower being the second from the chambers. It was found that the 8×40-ft. tower absorbed all the N_2O_3, leaving none for the smaller tower, when the combustion of brimstone was 9,600 lb. in 24 hours or 1 lb. S to 10·5 cu. ft.; and a consumption of 2·7% $NaNO_3$. This system has since been changed to pyrites fines (Virginia, 40% S) and is now making acid at the rate of 9,200 lb. sulphur available, or 1 lb. sulphur to 10·87 cu. ft., with a consumption of nitrate of soda of 2·60%. To increase the capacity it is proposed to change the connections so as to utilize the old 4-ft. tower as a drying tower, *i.e.*, without supplying it with acid, and doubtless the 8×40-ft. will be amply sufficient.

In these two cases, which are corroborated by many others, the proportion of Gay-Lussac capacity to chamber space (including all cubic capacity within the lead coverings, viz., lining, grill, entrance, packing and space unpacked on top) was in one case about 2,513 cu. ft. to 700,000 cu. ft., or about 0·36%; in the other it was 1,664 cu. ft. to 100,000 cu. ft., of which 1,250 cu. ft., or 1·25%, was found to be more than ample to do the work. This cannot be called an excessive proportion in either case. However, the proper absorption of niter in the Gay-Lussac tower depends much more on the condition of the entering gas and upon the efficiency of the construction of the tower than on any absolute ratio of cubic contents to chamber space, and the cases cited are rather as extreme instances than examples.

In the case of equal efficiency of packing, that is where each foot in the towers is calculated to bring about an equal contact between gas and downflowing acid, such as is the case in towers packed for instance with the mathematically correct Lunge plates, and for a given sectional area, say 8 ft. circular, I think that for the smallest systems a height of 30 ft. should be a minimum for the gas travel, and up to at least 100 ft. (in two towers of 50 ft. each) for systems of about 500,000 cu. ft. and 150 ft. in three towers of 50 ft. each for all larger systems. While these suggestions are largely the result of experience, they cannot be by any means considered exact. No exact proportions can be given for any one part of a sulphuric acid apparatus without consideration of the other parts. The relationship of the type and size of burners, the ore burned, the Glover tower, the chambers and other absorbing apparatus and connections, and the Gay-Lussac towers, as well as the purposes for which it is proposed to use the acid produced, are greatly interdependent upon one another.

THE MANUFACTURE OF SULPHURIC ACID BY CATALYTIC ACTION.

Probably the most interesting process now in evolution is the manufacture of sulphuric acid by catalytic action. This involves doing away entirely with lead chambers and in some cases almost all lead or platinum, and the production of sulphuric acid of all strengths from 50° B. containing 51% SO_3 to monohydrate containing 81·6% SO_3 and pyrosulphuric acid containing 89·89% SO_3 or even the sulphuric anhydride itself (SO_3). Robert Hasenclever in *Die Chemische Industrie*, of January, 1899, sums up the history of this development as follows: "Even dilute sulphuric acid, which for nearly 150 years has been produced in lead chambers, will be made in future to better advantage by means of contact substances (catalytic action), according to the latest reports of the Badischen Anilin und Soda Fabrik."

History of Catalytic Processes.—Up to the later seventies the manufacture of sulphuric anhydride was in the hands of the firm of J. D. Stark, of Prague, and owing to absence of competition the prices were high, ranging from 2 to 3 marks per kg. of distilled anhydride. It was only to be expected, therefore, that the chemical manufacturers should themselves undertake the manufacture of this product, which was done at about the same time by the following concerns: (1) The Fiskalischen Hütten, of Freiberg (Saxony); (2) Chapman, Messel & Co., of Silvertown, near London (England); (3) Dr. Jacob, of Kreuznach (Rhenish Prussia); (4) the Chemischen Fabrik Rhenania, of Aix la Chapelle (Rhenish Prussia); (5) Georg Karl Zimmer, of Mannheim (Baden); (6) W. Roth, of Plettenberg (Westphalia); (7) the Chemischen Fabrik Thann, of Thann (Alsace), and (8) the Chemischen Fabrik Einergraben, of Barmen (Rhenish Prussia). In 1883 the Color Works, formerly Meister, Lucius and Brüning, at Höchst (Hesse), bought the plant of Dr. Jacob, at Kreuznach, which were improved and extended by it. As early as 1887 the Badischen Anilin und Soda Works bought the patent of Schroeder & Hänisch, which they used in the manufacture of fuming sulphuric acid.

Winkler's Discovery.—The various processes by which these firms made their anhydride were all founded principally on the work of Prof. Clemens Winkler, published in *Dingler's Journal* in 1875, and entitled "Experiments on the Conversion of Sulphurous Acid into Sulphuric Anhydride by Means of Catalytic Action, in Relation to the Production of Fuming Sulphuric Acid." Up to this time no one had practically adopted this method, although various chemists, including Plattner (*Die Metallurgische Röst Processe*), and Phillips (English Patent No. 6,096, March, 1831), had called attention to the fact that sulphurous acid can be converted into sulphuric by the action of certain contact substances.

Winkler's first patent (D. R., 4,566) was taken out in 1878 on the production of platinized asbestos, in which "the coating of bulky and otherwise inert substances by the films of the so-called contact substances chemically deposited on them was protected." This patent was immediately bought by the Rhenania Chemical Co., of Aix la Chapelle, and the fiscal works of Freiberg, and in 1880 by the Société St. Gobain, Chauny & Cirey, and P. K. Ouschkoff & Co., of Moscow.

The Badischen Process.—In the meantime the Badischen Anilin und Soda Fabrik had developed a secret process and improved the manufacture of anhydride to such an extent that other manufacturers had to discontinue making it. The Badischen Anilin und Soda Fabrik describes its method as "an innovation in the process of manufacturing sulphuric anhydride and the apparatus used therefor." The following description appeared in the *Austrian Patent Bulletin:* "As is well known, the combination of $SO_2+O=SO_3$ produces a large amount of heat, which Hess (*Pogg. Ann.*, Vol. LVI., p. 471, 1842) and Momsen (*Ann. Çh. Ph.*, (4), XXX., p. 442, 1873) have measured accurately as 32·2 calories. As the reaction between SO_2+O occurs only at high temperatures, the gas, or rather mixture of gases, must first be heated before it will combine under the influence of the contact substances. This initial heat added to the heat of the reaction will, according to the volume of gas, assume great intensity, rising even to a white heat. It has been found that the practical carrying out of the process of manufacture of sulphuric anhydride is very injuriously affected in many ways by this excessive heat. The iron apparatus is rapidly destroyed by oxidation; the efficiency of the contact substance is deteriorated; the capacity of the apparatus is diminished, and above all, the rapidity of the process, which should be as great as possible, is greatly diminished."

The unsatisfactory course of the reaction is thus accounted for: SO_3 is partially decomposed into SO_2 and O at a temperature slightly higher than that favorable for its formation; while at the same time the combination of SO_2+O takes place much more rapidly when it first enters the apparatus, thus overheating the whole apparatus from the start. The more the apparatus is overheated the more SO_3 is decomposed, or in other words the greater the amount or concentration of the SO_2 the greater the heat and the greater the decomposition of the SO_3 formed, or the less perfect is the reaction. In either case much SO_2 leaves the apparatus unchanged, necessitating other provision for its utilization, such as lead chambers, or manufacture of sulphite, etc.

The Badischen process, which avoids these disadvantages, removes the harmful excess of heat from the contact substance and the apparatus by a readily controllable means of external cooling. By this method the contact apparatus is maintained at the temperature most favorable for the formation of SO_3, quite independently of the quantity or concentration of the SO_2 passing through it. Consequently it is possible to obtain an output of sulphuric anhydride in quantity comparable to that obtained by the chamber process, and, on account of the lower temperatures employed, to prolong the life of the contact mass and apparatus and greatly to increase their efficiency. This method can be varied in many ways and depends especially on the concentration of the gas employed. The cooling of the contact apparatus is generally effected by means of a current of gas whose velocity and temperature can be controlled, for instance, air or the gases themselves. Other means, however, can be used for cooling, such as liquid baths, especially metal baths whose temperatures can be controlled. In using the gases themselves for cooling the contact apparatus the stream of gas is passed partially or wholly into a chamber surrounding the contact tube in which chamber is absorbed the excess of heat from the contact tube, the gas being heated thereby to the proper temperature for the reaction before reaching the contact mass. This cooling or heating must be regulated according to the volume or concentration of the gas being used. To secure continuous use of the apparatus, and to prevent rapid deterioration of the contact substance, it is advisable to first purify the gas so as to free it from dust and volatile impurities such as arsenic, phosphorus, mercury and their compounds, etc.

The Belgium patent claims are as follows: No. 137,751. (*a*) "Improvements in the process of making sulphuric anhydride (or gas containing it) from sulphurous acid and oxygen or air or some other mixture of gases containing sulphurous acid and oxygen, by means of the usual contact substances, consisting in subjecting to a controllable cooling the apparatus containing the contact mass, into which the mixture of gases heated up to the proper temperature for the reaction and cleansed of injurious impurities, if necessary, is introduced for the purpose of abstracting the excess heat which is injurious to the process." (*b*) Relates to the construction of the apparatus used for this purpose which is described in detail, with examples of its application.

No. 137,752. (*a*) "New processes for producing and cleaning the gas mixtures for the production of sulphuric anhydride, which mixtures are obtained by burning sulphide ores or sulphur or otherwise, consisting in first thoroughly mixing the hot gases by means of a stream of gas or a jet of steam, and thereby completing the combustion of any sublimed sulphur. The introduction of this steam into the hot gases dilutes the concentrated sulphuric acid in them and changes the close combination of the sulphuric anhydride and impurities, precipitating the latter as a thin slime, and reduces the danger of injuring the apparatus and the formation of objectionable hydrogen compounds. Furthermore, the gases are gradually cooled and thereupon subjected to a washing process, until an optical and chemical examination shows the absence of dust, steam and volatile matter such as arsenic, phosphorus, mercury and their compounds. The gases are finally dried, if they have been washed with steam or water." (*b*) Relates to the recovery of the by-products obtained by the cooling and washing process, *i.e.*, arsenic, mercury, selenium, thallium, etc.

No. 137,753. "Improvements in the contact process, serving to produce sulphuric anhydride by passing technical gas mixtures, sulphur dioxide, oxygen, and possibly other gases, at ordinary atmospheric or even lower pressure through a contact mass which is distributed in the contact tube in many thin layers placed above each other on perforated plates, so arranged as to prevent the pressure of one contact layer on the other, and which neither diminishes the effect of cooling the contact tube nor allows the gases any other passage than through the contact mass."

The Farbwerke vormals Meister, Lucius & Brüning, of Höechst-am-Main, Germany, has patented* a process for the manufacture of sulphur trioxide by the combination of sulphur dioxide and oxygen, by "raising these gases to the temperature required for the reaction, by transferring to them heat from the produced sulphur trioxide, which has a high temperature, resulting from the combining reaction" referred to. The method of heat transference is effected through the agency of a simple counter-current apparatus. It is said that by the means and method described, complicated machinery and much fuel are spared, and that large apparatus, once the process is in action, "work entirely without supply of heat from outside."

The Grillo Process.—The latest development in this direction is disclosed by German patent No. 102,244, granted to the Actien Gesellschaft für Zinc Industrie, formerly Wilhelm Grillo, of Oberhausen, and Dr. Max Schroeder, of

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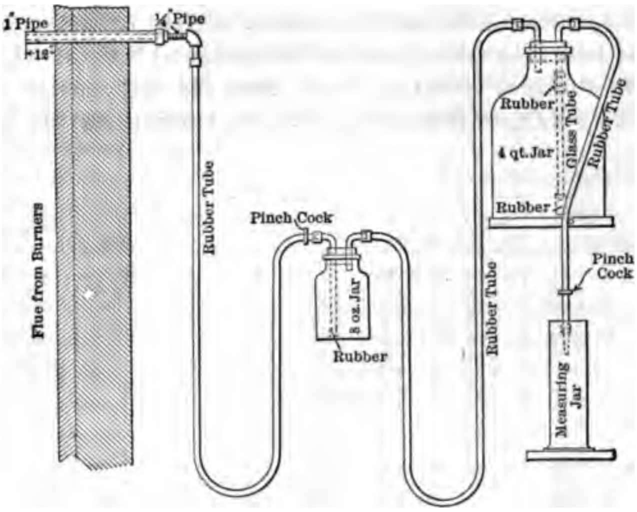
* English Patent No. 6,057, March 11, 1898.

Dusseldorf, which claims the discovery of cheaper and simpler methods of preparing the contact substance for the catalytic process. The inert or carrying substance for the active substance (platinum, etc.), consists of any salts soluble in water. When the contact substance has become inactive for any cause, such as the injurious action of fluedust, the substance can be readily dissolved, purified and regenerated, virtually without loss of platinum.

ANALYTICAL DETERMINATIONS IN SULPHURIC ACID WORKS.

Reich's Test for Sulphurous Acid Gas.—A solution of starch and a decinormal solution of iodine are required. These should be prepared as directed in Lunge & Hurter's *Alkali Makers' Handbook*, 2d Ed., p. 171. The iodine solution will keep well in a dark, cool place. The starch solution is more liable to spoil, even when saturated with salt, and should be renewed more frequently. It is important that the gas to be tested should be aspirated from a part of the flue which represents the gas actually leaving the burners. It may be taken through a 1-in. pipe projecting about 12 in. into the brick flue about 1 ft. below the cast-iron pipe leading to the Glover tower, or it may be taken direct from the cast-iron pipe itself, which is tapped so as to allow a ¼-in. pipe to be screwed through to its center. The 1-in. pipe built into the wall of the flue is outside of the latter reduced to ¼ in. and terminates with an elbow and nipple pointing downward to receive a rubber tube, the threads being filed smoothly off the nipple for about 1 in. The rubber pipe so attached should be of ample length so as to allow considerable freedom of movement. It connects with a bent $\frac{3}{16}$-in. glass tube, of which one leg passes through the rubber stopper of an 8-oz. white-glass wide-mouthed bottle nearly to the bottom of the latter. Another bent glass tube with equal legs is inserted in the other hole in the rubber stopper, just passing through the latter, while the outside leg is connected by a ¼-in. rubber tube of ample length to a bent glass tube with legs of equal length passing through a hole in the rubber stopper of a 4-qt. glass bottle. Through the other hole of this stopper a similar bend is placed and extended inside the bottle nearly to the bottom by a piece of glass tubing connected to it by a short piece of rubber tubing, and protected at its lower end also by a piece of rubber tube drawn over it. Outside the bottle a short piece of rubber tubing is attached, terminating in a short piece of glass tubing drawn down to about $\frac{1}{16}$ to ⅛ in. as a nozzle. This must be long enough to reach about 1 to 2 in. inside a 250 c.c. graduated jar. The rubber tubes between the flue and the 8-oz. bottle and the large bottle and the glass discharge nozzle are provided with strong spring pinchcocks.

To Make the Test: Fill the large bottle with water. See that the stopper is perfectly tight. Start the siphon by slight suction through the nozzle and close the pinchcock. Fill the 8-oz. bottle about one-quarter full of clean water from a wash bottle, slightly warmed in winter. Into this water pour about a teaspoonful of starch solution. Then by means of a pipette take 10 c.c. of the iodine solution and add to the contents of the 8-oz. bottle. Replace the rubber stopper tightly and close the pinchcock between the flue and the small bottle. Then open the pinchcock at the nozzle, allowing the water to waste. When the water ceases to run, proving the tightness of corks and connections throughout the apparatus, open the pinchcock between the flue and the small bottle. Take the small bottle in the left hand, keeping the right hand on the pinchcock at the nozzle, and shake the small bottle, not too violently, holding it to the light in such a way that any change in color can be readily noted. When a considerable change occurs in the color stop the flow of water with the right hand, and close the pinchcock on the tube between the flue and the small bottle. The tube between the flue and the pinchcock is now filled with the gas to be tested. Remove the stopper from the small bottle and add 10 c.c. of the iodine solution. Replace the cork tightly. Open the pinchcock between the flue and the small bottle. Then with the pinchcock at the nozzle in the right hand carefully waste water until the liquid in the glass tube, terminating the tube from the flue, is depressed to the bottom; or, in other words, until the tube is filled with the gas to its extreme end. Just before the first bubble of gas would escape and pass through the solution allow the water to commence running into the graduated jar. Shake the bottle as before and stop the water running the instant the color is discharged. The number of c.c. of water the jar holds at the exact point of the discharge of color from the small bottle represents the percentage by volume of SO_2 in the burner gas.



APPARATUS FOR THE DETERMINATION OF SULPHUROUS ACID.

Before making further tests the small bottle should be emptied and fresh water and starch solution used. Any intelligent foreman or workman can be taught to make the test, and after he has become accustomed may make such test in less than three minutes. He may either be provided with a percentage table or instructed to keep his gas between 128 and 138 c.c., or as may be required. The following table is useful for reference:

C.C. water in jar............	82	86	90	95	100	106	113	120	128	138	148	160	175	192	212
Per cent. SO_2 by volume....	12·0	11·5	11·0	10·5	10·0	9·5	9·0	8·5	8·0	7·5	7·0	6·5	6·0	5·5	5·0

*Permanganate Test for Nitrous Vitriol.**—This is made by running the nitrous vitriol to be tested from a 50 c.c. burette into a seminormal solution of potassium permanganate, prepared and standardized as directed by Lunge & Hurter, *Alkali Makers' Handbook*, p. 170. To make the test, fill the burette to zero point with the nitrous vitriol to be tested. Put about 50 c.c. of tepid water in a porcelain bowl, add 10 c.c. of the permanganate solution, and let the nitrous vitriol drop slowly from the burette into the bowl, stirring constantly with a glass rod. As the color commences to disappear add the nitrous vitriol more and more slowly until the color is entirely discharged. Sometimes a brown precipitate of hydrated manganese peroxide is formed, but will be dissolved before the test is completed. If the quantity of permanganate used is called x and the quantity of nitrous vitriol consumed in decolorizing it is called y, the quantity present, in grams per liter of acid, is $9{\cdot}5x \div y$ nitrous acid, $15{\cdot}75x \div y$ nitric acid, and $21{\cdot}25x \div y$ nitrate of soda (pure). When $x=10$ c.c. the following table gives the percentage of N_2O_3, HNO_3, and $NaNO_3$. (The strength of nitrous vitriol recovered from the Gay-Lussac tower should be from 2 to 2·5% N_2O_3, or from 2·8 to 2·2 c.c., reading on the burette when $x=10$.)

C.C. (a)	N_2O_3.	HNO_3.	$NaNO_3$.	C.C. (a)	N_2O_3.	HNO_3.	$NaNO_3$	C.C. (a)	N_2O_3.	HNO_3.	$NaNO_3$.
×=10	%	%	%	×=10	%	%	%	×=10	%	%	%
2·0	2·80	4·62	6·22	5·4	1·03	1·71	2·30	8·8	0·63	1·05	1·42
2·2	2·54	4·20	5·65	5·6	1·00	1·64	2·22	9·0	0·62	1·02	1·38
2·4	2·33	3·85	5·18	5·8	0·96	1·58	2·15	9·2	0·61	1·00	1·35
2·6	2·15	3·55	4·78	6·0	0·93	1·54	2·08	9·4	0·59	0·98	1·32
2·8	2·00	3·28	4·44	6·2	0·90	1·49	2·01	9·6	0·58	0·96	1·30
3·0	1·86	3·07	4·14	6·4	0·87	1·44	1·95	9·8	0·57	0·97	1·27
3·2	1·74	2·89	3·91	6·6	0·85	1·40	1·89	10·0	0·56	0·925	1·25
3·4	1·64	2·71	3·65	6·8	0·82	1·36	1·84	11·0	0·50	0·835	1·13
3·6	1·55	2·56	3·45	7·0	0·80	1·32	1·78	12·0	0·46	0·765	1·04
3·8	1·47	2·43	3·27	7·2	0·78	1·28	1·73	13·0	0·43	0·705	0·96
4·0	1·38	2·30	3·11	7·4	0·75	1·25	1·68	14·0	0·40	0·655	0·89
4·2	1·33	2·19	2·96	7·6	0·73	1·21	1·64	15·0	0·37	0·615	0·827
4·4	1·27	2·09	2·82	7·8	0·72	1·18	1·60	16·0	0·35	0.575	0·778
4·6	1·21	2·00	2·71	8·0	0·70	1·15	1·56	17·0	0·33	0·538	0·730
4·8	1·17	1·92	2·60	8·2	0·68	1·12	1·52	18·0	0·31	0·510	0·692
5·0	1·12	1·84	2·49	8·4	0·66	1·10	1·48	19·0	0·29	0·485	0·655
5·2	1·08	1·77	2·39	8·6	0·65	1·07	1·45	20·0	0·28	0·462	0·620

(*a*) Number of c.c. of nitrous vitriol consumed.

Specific Gravity Determinations.—The hydrometer as used in connection with sulphuric acid is simply an instrument for determining its specific gravity or density in comparison with distilled water at 60° F., and consequently with pure sulphuric acid, the percentage of monohydrate present. Impurities may of course cause a corresponding error in the hydrometer reading. Under any circumstances the use of the hydrometer must be limited to acids containing less than 96% monohydrate, and, in fact, all acids over 93·5% H_2SO_4 should be estimated alkalimetrically. Apparently it ought to be easy to make the hydrometric scale an exact basis of universal calculation, but in practice there are many different hydrometer scales and few accurate ones. One of the difficulties is the uncertainty as to the standard of maximum density. In Europe this is generally understood to be sp. gr. 1·842, at 15° C., or 66° B., although in England the Twaddell scale is used with a maximum sp. gr. of 1·850 or

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* Lunge, 2d Ed., pp. 177–180.

170° Tw. Even this would not be so serious if hydrometers of these scales were accurately graduated. This is, however, by no means the case, and the graduation is often based on impure commercial acid; 66° B. and 170° Tw. often represent acids of anywhere from 92 to 96% monohydrate or sp. gr. from 1·830 to 1·840. All intermediate graduations therefore are inaccurate. In the United States the Baumé scale is used with 66° = sp. gr. 1·835, or 93·5% monohydrate. In addition to these scales there are others, of varying degrees of inaccuracy, but for practical purposes it is only necessary to consider the following:

(1) Twaddell's scale: 0°=the immersion in distilled water at 60° F.; 170°= the immersion in rectified oil of vitriol of 1·850 sp. gr. at 60° F. Each intermediate degree represents a difference of ·005 in sp. gr. Thus 1° Tw.=sp. gr. 1·005 and 169° Tw.=1·845 sp. gr.

(2) The European Baumé scale is divided by the formula, $d=144{\cdot}3 \div 144{\cdot}3-n$, where d=sp. gr., and n the corresponding degree on the scale. In this case 0°=the immersion in distilled water of 15° C., and 66°=the immersion in rectified oil of vitriol of sp. gr. 1·842 at 15° C., the intermediate degrees being calculated by the above formula.

(3) The United States Manufacturing Chemists' Association scale is divided by the formula $d=145 \div 145-n$, values being as above. In this case 0°=the immersion of the instrument in distilled water at 60° F., compared with its immersion in oil of vitriol of 93·5% monohydrate and sp. gr. 1·835 at 60° F.

Each of these scales has its advantages. The United States Baumé has the double advantage of referring to an actual article of commerce, viz., 93·5% monohydrate, while its graduation covers all the strengths of acid for which a hydrometer is suited or accurate. The graduation of the United States hydrometer is openly referred to an article of commerce, instead of as in Europe being so referred in practice, but in theory referred to a strength and purity of acid seldom produced.

Since the graduation of hydrometers is referred to a temperature of 60° F. it is only possible to get a correct reading by carefully observing the temperature at the same time and making a plus or minus correction for differences of temperature. These allowances are 1° B. for every 46° F. in acid of 10° B. strength; 1° B. for every 31·8° F. in acid of 20° B. strength; and 1° B. respectively for every 30·25° F. in acid of 30° B., 31.46° F. in acid of 40° B., 34·69° F. in acid of 50° C., 40° F. in acid of 60° B., and 43·24° F. in acid of 66° B. For instance, in testing acids of about 60° B. strength an allowance must be made as follows: The hydrometer shows 60° B., the thermometer 140° F. (60° F. + 80° F.); since an allowance of 1° B. for every 40° F. is required, the reading on the hydrometer will be 60° + 2° = 62° B. If the thermometer reads 40° F. when the hydrometer reads 60° B., the hydrometer reading will then be 60° B. — 0·5° B., or 59·5° B.

Judging from the condition of hydrometers in many works it may not be amiss to say that not only to secure their endurance, but also accuracy, they should not be left floating in the acid, but after use should be washed and put aside in a convenient place.

COMPARATIVE TABLE SHOWING THE RELATION OF THE TWADDELL, EUROPEAN AND UNITED STATES BAUMÉ HYDROMETER SCALES TO SPECIFIC GRAVITY.

Sp. Gravity 60° F.=15° C.	Twaddell.	European Baumé.	U. S. Baumé.	Sp. Gravity.	Twaddell.	European Baumé.	U. S. Baumé.	Sp. Gravity.	Twaddell.	European Baumé.	U. S. Baumé.	Sp. Gravity.	Twaddell.	European Baumé.	U. S. Baumé.	Sp. Gravity.	Twaddell.	European Baumé.	U. S. Baumé.
1000	0·0	0·0	0·0	1150	30·0	18·8	18·9	1315	63·0	34·6	34·7	1500	100·0	48·1	48·4	1700	140·0	59·5	59·7
1005	1·0	0·7	0·7	1151	30·2	18·9	19·0	1318	63·6	34·8	35·0	1505	101·0	48·4	48·7	1705	141·0	59·7	59·9
1007	1·4	1·0	1·0	1152	30·4	19·0	19·1	1320	64·0	35·0	35·1	1510	102·0	48·7	49·0	1706	141·2	....	60·0
1010	2·0	1·4	1·4	1155	31·0	19·3	19·5	1325	65·0	35·4	35·6	1515	103·0	49·0	49·3	1710	142·0	60·0	60·2
1014	2·8	2·0	2·0	1160	32·0	19·8	20·0	1330	66·0	35·8	36·0	1520	104·0	49·4	49·6	1711	142·2	60·0	60·3
1015	3·0	2·1	2·1	1162	32·4	20·0	20·2	1332	66·4	36·0	36·1	1525	105·0	49·7	49·9	1715	143·0	60·2	60·5
1020	4·0	2·7	2·8	1165	33·0	20·3	20·6	1335	67·0	36·2	36·4	1526	105·2	49·8	50·0	1720	144·0	60·4	60·7
1021	4·2	2·9	3·0	1169	33·8	20·8	21·0	1340	68·0	36·6	36·8	1530	106·0	50·0	50·2	1725	145·0	60·6	60·9
1022	4·4	3·0	3·1	1170	34·0	20·9	21·1	1342	68·4	36·8	37·0	1535	107·0	50·3	50·5	1726	145·2	60·7	61·0
1025	5·0	3·4	3·5	1171	34·2	21·0	21·2	1345	69·0	37·0	37·2	1540	108·0	50·6	50·8	1730	146·0	60·9	61·1
1028	5·6	3·9	4·0	1175	35·0	21·4	21·6	1350	70·0	37·4	37·6	1542	108·4	50·7	51·0	1732	146·4	61·0	61·2
1029	5·8	4·0	4·1	1179	35·8	21·9	22·0	1355	71·0	37·8	38·0	1545	109·0	50·9	51·2	1735	147·0	61·1	61·4
1030	6·0	4·1	4·2	1180	36·0	22·0	22·1	1357	71·4	38·0	38·2	1547	109·4	51·0	51·3	1740	148·0	61·4	61·6
1035	7·0	4·7	4·9	1185	37·0	22·5	22·5	1360	72·0	38·2	38·4	1550	110·0	51·2	51·5	1745	149·0	61·6	61·9
1036	7·2	4·9	5·0	1188	37·6	22·8	23·0	1365	73·0	38·6	38·8	1555	111·0	51·5	51·8	1747	149·4	61·7	62·0
1037	7·4	5·0	5·1	1190	38·0	23·0	23·1	1368	73·6	38·8	39·0	1559	111·8	51·9	52·0	1750	150·0	61·8	62·1
1040	8·0	5·4	5·6	1195	39·0	23·5	23·7	1370	74·0	39·0	39·2	1560	112·0	51·8	52·1	1753	150·6	62·0	62·3
1043	8·6	5·9	6·0	1198	39·6	23·8	24·0	1375	75·0	39·4	39·6	1563	112·6	52·0	52·2	1755	151·0	62·1	62·4
1045	9·0	6·0	6·3	1200	40·0	24·0	24·2	1380	76·0	39·8	39·9	1565	113·0	52·1	52·4	1760	152·0	62·3	62·6
1050	10·0	6·7	6·9	1205	41·0	24·5	24·6	1381	76·2	39·9	40·0	1570	114·0	52·4	52·6	1765	153·0	62·5	62·8
1051	10·2	6·9	7·0	1208	41·6	24·8	25·0	1383	76·6	40·0	40·2	1575	115·0	52·7	52·9	1768	153·6	62·6	63·0
1052	10·4	7·0	7·2	1210	42·0	25·0	25·1	1385	77·0	40·1	40·4	1576	115·2	52·8	53·0	1770	154·0	62·8	63·1
1055	11·0	7·4	7·6	1215	43·0	25·5	25·6	1390	78·0	40·5	40·7	1580	116·0	53·0	53·2	1774	154·8	63·0	63·2
1058	11·6	7·9	8·0	1218	43·6	25·8	26·0	1394	78·8	40·7	41·0	1585	117·0	53·3	53·5	1775	155·0	63·0	63·3
1060	12·0	8·0	8·3	1220	44·0	26·0	26·1	1395	79·0	40·8	41·1	1590	118·0	53·6	53·8	1780	156·0	63·2	63·5
1065	13·0	8·7	8·9	1225	45·0	26·4	26·6	1397	79·4	41·0	41·2	1593	118·6	53·8	54.0	1785	157·0	63·5	63·8
1066	13·2	8·9	9·0	1229	45·8	26·9	27·0	1400	80·0	41·2	41·4	1595	119·0	53·9	54·1	1790	158·0	63·7	64·0
1067	13·4	9·0	9·1	1230	46·0	26·9	27·1	1405	81·0	41·6	41·8	1597	119·4	54·0	54·2	1795	159·0	64·0	64·2
1070	14·0	9·4	9·5	1231	46·2	27·0	27·2	1408	81·6	41·8	42·0	1600	120·0	54·1	54·4	1796	159·2	64·0	64·2
1074	14·8	9·9	10·0	1235	47·0	27·4	27·6	1410	82·0	42·0	42·2	1605	121·0	54·4	54.3	1800	160·0	64·2	64·4
1075	15·0	10·0	10·2	1239	47·8	27·8	28·0	1415	83·0	42·3	42·6	1610	122·0	54·7	54·9	1801	160·2	64·2	64·5
1080	16·0	10·6	10·8	1240	48·0	27·9	28·1	1420	84·0	42·7	42·9	1611	122·2	54·8	55·0	1805	161·0	64·4	64·7
1082	16·4	10·9	11·0	1241	48·2	28·0	28·2	1421	84·2	42·8	43·0	1615	123·0	55·0	55·2	1807	161·4	64·5	64¾
1083	16·6	11·0	11·2	1245	49·0	28·4	28·6	1424	84·8	43·0	43·2	1620	124·0	55·2	55·5	1810	162·0	64·6	64·9
1085	17·0	11·2	11·4	1250	50·0	28·8	29·0	1425	85·0	43·1	43·3	1625	125·0	55·5	55.8	1812	162·4	64·7	65·0
1090	18·0	11·9	12·0	1252	50·4	29·0	29·2	1430	86·0	43·4	43·6	1629	125·8	55·7	56·0	1815	163·0	64·8	65·2
1091	18·2	12·0	12·1	1255	51·0	29·3	29·5	1435	87·0	43·8	43·9	1630	126·0	55·8	56·1	1818	163·6	64·9	65¼
1095	19·0	12·4	12·6	1260	52·0	29·7	29·9	1436	87·2	43·9	44·0	1635	127·0	56·0	56·4	1819	163·8	65·0	65¼
1098	19·6	12·9	13·0	1261	52·2	29·9	30·0	1438	87·6	44·0	44·2	1640	128·0	56·3	56·6	1820	164·0	65·0	65·3
1100	20·0	13·0	13·2	1263	52·6	30·0	30·2	1440	88·0	44·1	44·3	1645	129·0	56·6	56·9	1822	164·4	65·1	65·4
1105	21·0	13·6	13·8	1265	53·0	30·2	30·5	1445	89·0	44·4	44·7	1648	129·6	56·8	57·0	1824	164·8	65·2	65·5
1107	21·4	13·9	14·0	1270	54·0	30·6	30·8	1450	90·0	44·8	45·0	1650	130·0	56·9	57·1	1825	165·0	65·2	65·5
1108	21·6	14·0	14·1	1272	54·4	30·8	31·0	1453	90·6	45·0	45·2	1652	130·4	57·0	57·2	1826	165·2	65·3	65·6
1110	22·0	14·2	14·4	1274	54·8	31·0	31·2	1455	91·0	45·1	45·4	1655	131·0	57.1	57·4	1828	165·6	65·4	65·7
1115	23·0	14·9	15·0	1275	55·0	31·1	31·3	1460	92·0	45·4	45·7	1660	132·0	57·4	57·6	1830	166·0	....	65¾
1116	23·2	15·0	15·1	1280	56·0	31·5	31·7	1465	93·0	45·8	46·0	1665	133·0	57·7	57·9	1831	166·2	65·5	65·8
1120	24·0	15·4	15·6	1283	56·6	31·9	32·0	1468	93·6	46·0	46·2	1666	133·2	....	58·0	1833	166·6	65·6	65·9
1124	24·8	15·9	16·0	1285	57·0	32·0	32·2	1470	94·0	46·1	46·4	1670	134·0	57·9	58·1	1835	167·0	65·7	66·0
1125	25·0	16·0	16·2	1290	58·0	32·4	32·6	1475	95·0	46·4	46·8	1671	134·2	58·0	58·2	1838	167·6	65·8	....
1130	26·0	16·5	16·7	1295	59·0	32·8	33·0	1479	95·8	46·7	47·0	1675	135·0	58·2	58·4	1840	168·0	65·9	....
1133	26·6	16·9	17·0	1297	59·4	33·0	33·2	1480	96·0	46·8	47·1	1680	136·0	58·4	58·7	1842	168·4	66·0	....
1134	26·8	17·0	17·1	1300	60·0	33·3	33·4	1483	96·6	47·0	47·3	1683	137·0	58·7	58·9	1845	169·0	....	....
1135	27·0	17·1	17·3	1305	61·0	33·7	33·9	1485	97·0	47·1	47·4	1686	137·2	58·7	59·0	1850	170·0	....	....
1140	28·0	17·7	17·8	1306	61·2	33·8	34·0	1490	98·0	47·4	47·7	1690	138·0	58·9	59·2				
1142	28·4	18·0	18·0	1308	61·6	34·0	34·1	1495	99·0	47·8	48·0	1691	138·2	59·0	59·3				
1145	29·0	18·3	18·4	1310	62·0	34·2	34·3	1498	99·6	48·0	48·3	1695	139·0	59·2	59·5				

TALC AND SOAPSTONE.

The production of fibrous talc, or agalite, in 1898 continued to come, as heretofore, from Edwards and Fowler, St. Lawrence County, N. Y. Owing to the increased requirements of the daily newspapers on account of the Spanish-American war the paper trade was stimulated greatly, and consequently a large amount of fibrous talc was required, so that the statistics for 1898 show an increase. The increased demand sustained prices at about $5@$6 per ton, although previously the business had been somewhat upset through the anxiety of some of the producers to make sales. The average cost of production is believed to be nearly $5 per ton and there is not much margin of profit to the miners and grinders when the ground talc fetches so little as that. On December 1, 1898, an agreement was made by the producers fixing the price at $7 per ton. It is questionable, however, if this increase will not affect unfavorably the consumption, since in the paper trade fibrous talc comes into competition with china clay and other loading materials which, although perhaps inferior to fibrous talc, can be substituted for it if the differential in price becomes too high. The production of St. Lawrence County, N. Y., was 54,807 short tons, valued at $285,759 in 1898, against 52,836 tons, valued at $283,685 in 1897.

The business in common talc was quiet throughout 1898. The supply of domestic mineral continued light, but certain of the North Carolina producers were reorganized with a view of working on a larger scale. There was a small decrease in production. Inquiry was made in various States for new sources of supply, and prospectors were particularly active in Virginia, but no important discoveries were made. American talc sold in New York at $10@$15·50 per 2,000 lb., according to quality. The supply of French and Italian talc continued as in previous years. The use of talc as a loading material, and for producing a satin-surfaced paper, has increased greatly in recent years. Although more expensive than china clay, it is said to be preferable for the finer grades of paper, where a better finish is required.

There was no important change in the soapstone business, except the practical exhaustion of the mine at Francestown, N. H. This quarry was worked in 1898 for a small output, but probably it will now be closed down, since the vein is said to be *entirely exhausted*. Formerly there was an output of 3,000 to 4,000 tons from this quarry, but the pit is now so deep and narrow at the bottom that only 10 or 12 men can find room to work, and there has been difficulty in securing men to work in it. The pit is about 1,500 ft. long, 70 ft. wide and 166 ft. deep, going down on an incline. Both the hanging wall and foot wall are solid. Continual pumping was necessary, since without it the pit would fill about 6 ft. in 12 hours until a depth of 40 ft. of water was reached, after which the level rose more slowly, the pit being wider. Labor cost $2 per day, and with the high cost of pumping operations became unprofitable. Stone quarried from the pit was carried to Greenfield, a distance of 7 miles, where it was loaded on cars for shipment to Nashua, at which place it was worked up into tables, sinks and other similar articles. The Albemarle Soapstone Co. opened a new quarry at Howardsville, Va., on a stratum of very fine green stone. Production went on in Vermont as heretofore. A new discovery was reported at Reedtown, near Sandusky, Ohio, but investigation showed that the material was not soapstone at all.

The Banning Co., of Los Angeles, Cal., did not produce any soapstone from its quarries on Santa Catalina Island in 1898, but sold a few tons left over from the previous year. The peculiar stone called "serpentine," which nevertheless has many of the characteristics of soapstone, and is so classed in our statistics, was sold to the value of $3,000, being used for slabs and other building work. The principal demand for this material at the present time is for general marble work, electric switchboards and tablet boards. A larger output will probably be made in 1899. The Santa Catalina serpentine is a peculiar material. It can be cut into wonderful shapes, *e.g.*, cylinders with walls only $\frac{1}{32}$-in. thick.

STATISTICS OF FIBROUS TALC AND SOAPSTONE IN THE UNITED STATES.

(In tons of 2,000 lb.)

Year.	Production.									Imports. (*b*)		
	Fibrous Talc.			Common Talc.			Soapstone.			Tons. *	Value.	Per Ton
	Tons.	Value.	Per Ton	Tons.	Value.	Per Ton	Tons.	Value. (*a*)	Per Ton			
1894....	50,500	$454,500	$9·00	$8,907	$80,163	$9·00	12,137	$121,370	10·00	622	$6,815	$10·95
1895....	40,000	320,000	8·00	9,063	86,099	9·50	13,070	130,700	10·00	3,165	26,843	8·48
1896....	51,816	256,080	4·94	7,098	63,585	8·97	14,350	143,500	10·00	1,950	18,693	9·60
1897....	52,836	283,685	5·37	10,164	90,908	8·94	16,904	169,040	10·00	799	8,424	10·54
1898....	54,807	285,759	5·21	9,112	78,645	8·63	18,862	158,635	8·41	445	5,326	10·79

(*a*) In reporting the value of their output of soapstone producers generally give the figures for the finished articles which they make. Since a varying proportion of labor enters into these, such figures are apt to be misleading. Few producers are able to name a value of the crude stone, or roughly dressed blocks. We have therefore valued the material arbitrarily at this stage at $10 per ton, except in 1898 when a large production of inferior stone reduced the average. (*b*) Talc, ground, powdered or prepared.

THE FIBROUS TALC INDUSTRY IN ST. LAWRENCE COUNTY, N. Y.

J. Nelson Nevius described the present condition of this industry in the *Engineering and Mining Journal*, February 25, 1899.* The companies engaged in the talc industry are the International Pulp Co., which is the largest operator, and which has absorbed the Adirondack Pulp Co., the Agalite Fiber

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* With respect to the geology and earlier operations reference should be made to THE MINERAL INDUSTRY, *Vols. I., II. and VI.*, especially the first two.

Co., the Natural Dam Pulp Co., the St. Lawrence Pulp Co., and the Asbestos Pulp Co.; the United States Talc Co., which is the second largest operator; the Columbian Talc Co., Keller Brothers, and the American Talc Co. There are about a dozen mines at Talcville, though half of them are now idle on account of the dullness of trade. They are situated in three groups, on the sides of low hillocks, along an approximately E. N. E. line. As the strike of the country rock is almost identical with the direction of the line between the mines, it is possible that the latter are all located on the same seam of talc, which may be continuous over the entire distance. The mines are similar in general characteristics. Mine No. 3 of the International Pulp Co. is opened to a depth of 300 ft. The hanging-and foot walls are nearly parallel and dip at an angle of 50°. The seam of talc varies from 15 to 25 ft. in thickness. So far it has been removed for a distance of 250 ft. along the strike of the vein and 60 ft. along the dip. Mine No. 5 of the same company, which is about 1,500 ft. from No. 3, is being exploited, but has not been opened to so great a depth as No. 3. The Columbian Talc Co. is mining a 15-ft. seam at Little York, in the township of Fowler. Two grades of talc are mined in this district: (1) "First quality fiber," which is tough, compact rock of variable texture and appearance; the two chief types are (*a*) the distinctively fibrous, with clusters of fibers ramifying in all directions, and usually of a grayish tint, and (*b*) the less fibrous in appearance, in which the fibers tend to run in one direction, the color being usually of a light greenish tint; (2) "second quality fiber," which is either (*a*) "gritty," when an otherwise first quality material contains some harder impurity, which is usually tremolite, or some other member of the amphibole group of minerals, or (*b*) "scaly," when it looses its fibrous structure and consequent tough character and becomes flaky and brittle. The last variety predominates more in the mines at Fowler than in those at Talcville. Second quality talc is useful only to a limited extent, since the trade requires a pulp with but little gritty or scaly material.

The pulverized talc is designated as "pulp." The pulp mills are located along the Oswegatchie River, between Talcville and Gouverneur. Some of the smaller mills are operated by water power, but the majority use steam power exclusively, or combine the two. The mills are several miles from the mines, and the Gouverneur & Oswegatchie Railroad transports the raw talc to the mills and carries the pulp to the main line of the Rome, Watertown & Ogdensburg Railroad. The operations are practically indentical in all the mills. At mill No. 3 of the International Pulp Co., situated at Hailesboro, about 1·5 miles east of Gouverneur, the material received from the mine varies in size from dust to masses 2 ft. or more in length. The large pieces are broken with sledges so that the maximum size is not more than 9 in. in diameter, then passed through a Blake crusher and next through a pair of slightly corrugated steel rolls 30 in. in diameter. The product of the rolls goes to a mill, consisting of two inclosed steel plates revolving rapidly in opposite directions, but not in contact, the object being to cause the talc to be pulverized by collision of its own particles, thereby avoiding admixture of particles of iron that would otherwise be worn from the plates and might discolor the pulp. A current of air is forced through the mill, blowing out the pulp as it becomes finer. The product of this mill goes to a series of Alsing cylinders, which are made of steel 6 ft. in diameter and 10 ft. in length, lined with enamelled porcelain brick and fitted with a manhole in the side for charging and discharging. These cylinders revolve horizontally about the long axis. The pulp is charged into them together with a quantity of water-worn quartz pebbles 2·5 in. in diameter, the cylinder being consequently a species of ball mill. After revolution for a sufficient length of time the manhole cover is replaced by a grating, with ⅜-in. spaces, and the cylinder is revolved again, whereby the pulverized talc is discharged, while the pebbles are retained in the cylinder. The pulverized product is filled automatically into paper bags holding 50 lb., or cloth bags holding 160 lb. One of the bag-filling machines requires two men to handle the bags and fills about 200 bags an hour. At the largest mill in the district, No. 6 of the International Pulp Co., experiments are being made with a new process by which grit and other impurities may be separated from the pulp.

The pulp is prepared in several grades, for which each company has its special names. The greatest demand for it comes from the paper industry. It is used as a filler in many qualities of paper, but the greatest consumption is in the manufacture of newspaper stock, for which purpose it is mixed with wood pulp. The talc pulp thus used is a very finely pulverized grade, designated "finished asbestine pulp" by the International Pulp Co., and "fine cylinder stock" by the United States Talc Co. A less finely pulverized ("fluffy") grade is produced by omitting the Alsing cylinder process above described. This grade is used with "asbestos" (chrysotile) fiber in the manufacture of asbestos paper, asbestos packing, etc., and is designated "special asbestine pulp" by the International Pulp Co., and "No. 1 buhr stock" by the United States Talc Co. The talc is used also in the manufacture of certain paints and wall plasters, and as an adulterant in soaps and similar articles.

TIN.

THERE was no production of tin from ore in the United States during 1898. Some prospecting for tin ore was carried on, without successful results. A small amount of tin was recovered from tin-plate scrap by the Vulcan Metal Refining Co., at Sewaren, N. J., the plant having been only a short time in operation. About 1·5% tin is recovered from the ordinary scrap. The process employed is an electrolytic one, similar to that which has been successfully used for some time in Germany and Holland, whither much American tin-plate scrap has been shipped. This material has realized about $5 a ton in bales delivered on the wharf in New York. After the recovery of the tin the steel is pressed into fagots for remelting. The Sewaren plant is said to have a nominal capacity of 10,000 tons of scrap per annum.

The year 1898 was marked by a further large development in the American tin-plate industry, and the organization of the American Tin-Plate Co., which practically controls this business. The companies which entered this combination were as follows: Ætna-Standard Iron and Steel Co., Bridgeport, Ohio, eight mills; American Tin-Plate Co., Ellwood, Ill., 20 mills, and Montpelier, Ind., six mills; Atlanta Steel and Tin-Plate Co., Atlanta, Ind., six mills; Baltimore Tin-Plate Co., Lisbon, Ohio, six mills; Britton Rolling Mill Co., Cleveland, three mills; Cincinnati Rolling-Mill and Tin-Plate Co., four mills; Crescent Sheet and Tin-Plate Co., six mills; Cumberland, Md., Steel and Tin-Plate Co., five mills; Ellwood Tin-Plate Co., Ellwood City, Pa., five mills; Falcon Tin-Plate and Sheet Co., Niles, Ohio, six mills; Great Western Tin-Plate Works, Joliet, Ill., four mills; Humbert Tin-Plate Co., Connellsville, Pa., six mills; Irondale Steel and Iron Co., Middletown, Ind., six mills; Johnstown Tin-Plate Co., Johnstown, Pa., four mills; La Belle Iron Works, Wheeling, W. Va., 10 mills; Laughlin Nail Co., Martin's Ferry, Ohio, 14 mills; Marshall Bros. & Co., Philadelphia, six mills; Monongahela Tin-Plate Co., Pittsburg, 14 mills; Morewood Co., Gas City, Ind., eight mills; Morton Tin-Plate Co., Cambridge, Ohio, five mills; National Tin-Plate Co., Anderson, Ind., six mills; National Tin-Plate Co., Monessen, Pa., 14 mills; Neshannock Sheet and Tin-Plate Co., New Castle, Pa., six mills; New Castle Steel and Tin-Plate Co., New Castle, Pa., 20 mills; Norton Bros., Maywood, Ill., 18 Norton machines; Norton Tin-Plate and Can Co., Baltimore, 16 sets; Ohio River Sheet and Tin-Plate Co., Remington, Pa., two mills; Pennsylvania Tin-Plate Co., New Kensington, Pa., six mills; Pittsburg Tin-Plate Co., New Kensington, Pa., six mills; Reeves Iron Co., Canal Dover, Ohio, four mills; Shenango Valley Steel Co., New Castle, Pa., 30 mills; Somers Bros., Brooklyn, three mills; Star Tin-Plate Co., Pittsburg, eight mills; Stickney Iron Co., Baltimore, two mills; St. Louis Stamping Co., St. Louis, eight mills; United States Iron and Tin-Plate Mfg. Co., Demmler, Pa., eight mills; Wallace, Banfield & Co., Irondale, Ohio, four mills; Washington Steel and Tin-Plate Mills, Washington, Pa., four mills. These companies were represented as having made a production of 7,633,556 boxes of tin plate in 1898 at an average profit of 35c. per box, or a total of $2,671,754. It was expected that the combination would effect a large reduction in operating expenses.

IMPORTS OF TIN INTO THE UNITED STATES.

Year.	Pounds.	Value.	Year.	Pounds.	Value.	Year.	Pounds.	Value.
1893........	40,184,556	$8,007,292	1895.......	54,252,045	$7,405,619	1897.......	55,172,571	$7,415,933
1894........	39,268,628	5,944,065	1896.......	44,639,324	5,848,933	1898.......	62,748,399	8,770,221

PRODUCTION OF TIN IN THE WORLD. (*n*)

Year.	Australia. (*a*)			Austria. (*b*)			Banka and Billiton. (*c*)	Bolivia. (*d*)	England. (*e*)			Germany. (*f*)		
	Met. Tons.	Value. (*o*)	Per M Ton.	Met. Tons.	Value. (*o*)	Per M Ton.	Metric Tons.	Metric Tons.	Met. Tons.	Value. (*o*)	Per M Ton.	Met. Tons.	Value. (*o*)	Per M Ton.
1893..	2,679	$1,115,695	$416	66	$36,824	$558	10,877	2,956	9,124	$3,928,705	$431	951	$348,135	$365
1894..	2,653	897,225	338	80	38,812	485	10,527	3,538	8,461	3,022,500	357	896	271,804	303
1895..	2,235	680,400	304	60	24,257	404	10,932	4,166	6,755	2,233,900	331	884	266,292	301
1896..	1,737	496,060	286	54	18,260	341	11,963	4,104	4,915	1,538,390	313	826	238,017	288
1897..	1,047	353,440	338	48	16,806	350	14,224	5,594	4,524	1,456,680	322	929	287,829	310

Year.	Japan. (*g*)	Mexico. (*h*)	Portugal. (*i*)	Russia. (*j*)			Singkep. (*k*)	Straits Settlements. (*l*)	Tasmania. (*m*)			Total Met. Tons.
	Met. Tons	Met. Tons.	Met. Tons.	Met. Tons.	Value (*o*)	Per M Ton.	Met. Tons.	Met. Tons.	Met. Tons.	Value. (*o*)	Per M Ton.	
1893........................	39	19	53	7	$2,501	$357	170	44,658	3,179	$1,301,095	$409	74,778
1894........................	39	42	26	4	1,440	360	265	51,100	3,195	1,348,000	422	80,826
1895........................	48	39	3	21	5,068	241	654	53,354	4,265	1,301,350	305	83,416
1896........................	50	5	6	2	476	238	852	53,964	4,759	1,327,369	290	83,237
1897........................	(*p*)	1	1	(*p*)	(*p*)	(*p*)	813	45,632	3,281	999,340	305	76,094

(*a*) From *Report of Secretary of Mines and Agriculture of New South Wales*, which is the only Australian colony that produces metallic tin. Tin ore is also exported from New South Wales. Tin ore is produced in Victoria, Queensland, South Australia, and Western Australia, but its metal contents are reported in the productions of other countries.

(*b*) From the *Statistisches Jahrbuch des K. K. Ackerbau-Ministeriums*. The tin production of Austria is derived partly from domestic ores, and partly from Bolivian ores and impure tin from the East imported for refining.

(*c*) Total sales in Holland and Java as reported by William Sargant & Co.

(*d*) Exports of tin and tin in ore from Bolivia to England and the Continent. Some of this tin may be produced in Peru. It is all exported through Peruvian and Chilean ports.

(*e*) From *Mineral Statistics of the United Kingdom*. These figures give the amount of tin estimated to be obtainable in smelting the ore of domestic production. They differ considerably from the figures used by William Sargant & Co., which are possibly obtained directly from the smelters.

(*f*) From *Vierteljahrs und Monatshefte zur Statistik des Deutschen Reichs*. By far the larger part of this production is from Bolivian ores.

(*g*) From the *Resumé Statistique de l'Empire du Japon*.

(*h*) According to export returns. Small amounts produced by natives for domestic consumption are not included.

(*i*) From official returns furnished by the Portuguese Government. The figures for 1895–1896 include tin ore.

(*j*) From Russian official reports.

(*k*) The figures for 1891-98 are for fiscal years, as reported by O. H. Van der Wyck in *Mineral Resources of the United States* for 1895; the figures for subsequent years represent sales in Holland, reported by De Mouchy & Havelaar and W. Sargant & Co.

(*l*) Shipments from the Straits to Europe and America, as reported by William Sargant & Co., and to India and China by Boustead & Co., except the figure for 1896, which is taken from the *Report of the Governor of the Straits Settlements.*

(*m*) From *Report of the Secretary of Mines.*

(*n*) It will be observed that this table differs from the statistics usually referred to for the world's production of tin, namely, those of William Sargant & Co., and of the *Metallgesellschaft Frankfurt am Main.* This is because (1) they reckon Australian shipments to Europe and America instead of the Australian production, whereby a good deal of Australian tin consumed at home is not included; (2) shipments from the Straits to India and China and the production of several minor countries are not included. A small amount of tin produced in Spain is necessarily omitted, owing to the absence of statistics for that country.

(*o*) In cases where the statistics in the above table have been taken from official sources, wherein values of product have been stated, a calculation of unit values has been made. This developed some amazing differences, which are undoubtedly due to making value calculations at different stages of the product; *i.e.*, more or less cost of carriage is included.

(*p*) Statistics not yet published.

THE PRINCIPAL TIN SUPPLIES OF THE WORLD. (*a*)

(In long tons.)

	1892.	1893.	1894.	1895.	1896.	1897.	1898.
English production	9,270	8,837	8,328	6,648	4,837	4,453	4,200
Straits' shipments to Europe and America	34,648	39,670	45,640	47,840	47,180	41,700	43,350
Australian shipments to Europe and America	5,972	5,579	5,824	4,992	4,320	3,466	2,420
Banka sales in Holland	6,300	5,244	5,626	6,221	6,735	8,900	9,038
Sales of Singkep in Holland	*Nil.*	*Nil.*	261	644	839	800	*Nil.*
Billiton sales in Java and Holland	5,560	5,462	4,735	4,539	5,040	5,100	5,342
Bolivian arrivals on Continent	....	....	....	....	210	1,208	1,000
Bolivian arrivals in England	2,819	2,909	3,482	4,097	3,829	4,298	3,464
Straits shipments to India and China	3,450	4,285	4,655	4,674	6,118	3,214	2,551
Totals in long tons	68,019	71,986	78,551	79,655	79,108	73,139	71,365
Totals in metric tons	69,107	73,138	79,808	80,930	80,374	74,309	72,507

(*a*) This table is based on the statistics compiled by William Sargant & Co., but the figures of English production are taken from the British blue-book, except for 1898, where Sargant's have been used. The Straits shipments to India and China are as reported by Messrs. Boustead & Co. This table does not include the production of Germany, Austria, Spain, Portugal, and various other countries.

CONSUMPTION AND STOCKS OF TIN IN ENGLAND, AMERICA, AND HOLLAND. (*a*)

(In long tons.)

	1893.	1894.	1895.	1896.	1897.	1898.
Consumption—						
Deliveries from London after deducting all shipments to America	18,663	18,050	17,222	19,015	17,118	13,472
Deliveries from Holland after deducting exports to London and America	7,853	8,207	9,029	10,150	11,680	12,130
English consumed at home	1,473	2,314	988	5,000	4,400	4,100
Exports of English, minus quantity shipped to America	6,554	5,686	5,530			
American consumption of all sorts	19,000	16,650	22,500	20,500	22,500	29,000
Billiton sent to other ports than Holland	1,919	862	1,532	1,603	1,370	2,120
Straits direct to Continent, less re-exports to America and England	8,900	11,725	7,622	11,400	9,170	12,740
Bolivian delivered from Liverpool	2,704	3,323	4,099	4,071	3,838	3,874
Bolivian delivered from Continent	....	....	....	210	1,208	1,000
Totals	67,066	66,817	68,522	71,949	71,284	78,436
Stocks, December 31—						
Stock of foreign in London	4,392	8,985	13,539	18,097	15,146	8,110
Foreign landing in London	817	1,189	1,025	1,174	673	165
Straits afloat for London, including wire advices	3,470	3,778	2,825	2,792	2,500	1,050
Australian afloat for London, including wire advices	909	650	825	525	600	400
Banka on warrants in Holland	319	1,006	1,154	1,616	2,877	2,228
Billiton in Holland	640	1,632	1,507	1,638	1,328	1,036
Billiton afloat for Holland	1,600	1,129	1,097	1,742	1,193	1,322
Straits stock in Holland	670	767	277	789	377	215
Straits afloat for Holland	....	....	....	950	100	560
Straits afloat for Continent	....	....	....	650	600	454
Bolivian in Liverpool	335	494	492	250	710	300
Total stocks	13,152	19,630	22,741	30,223	26,104	15,840
Estimated stock in America and quantity floating	2,730	3,820	6,823	3,925	4,500	3,800
Grand totals	15,882	23,450	29,564	34,148	30,604	19,640
Trading Company's reserves of unsold Banka stock in Holland	4,200	5,770	6,140	5,953	4,333	3,213

(*a*) *From the annual metal circular of William Sargant & Sons.*

EXPORTS OF TIN PLATES FROM THE UNITED KINGDOM.
(In tons of 2,240 lb.)

	1892.	1893.	1894.	1895.	1896.	1897.	1898.
To the United States........	278,479	255,583	226,880	222,901	113,049	85,472	65,337
To other countries...........	117,101	123,650	127,048	143,081	153,914	185,758	186,432
Totals....................	395,580	379,233	353,928	365,982	266,963	271,230	251,769

Australia.—A new tin field was discovered in 1898 in the vicinity of Beenah, Gippsland. The gravel, which is said to be from 6 to 8 ft. in depth, extends from the surface and can be worked cheaply, there being a good supply of water. Small parcels of the ore gave returns of 72 to 74% tin.

Bolivia.—Charles S. Pasley described* the tin deposits of Bolivia, all of which are situated on the eastern cordillera of the Andes, between latitude 21° S. and 16° S., the mines extending north and south over a distance of 300 miles and appearing in distinct groups. They are all to the eastward of the great Bolivian plateau, in which are the lakes of Titicaca and Poopo. No tin mines are known to exist in the western or coast Cordillera, nor are there any in Peru and Chile. There are four tin-producing districts: Oruro in the center, La Paz in the north, Chorolque in the south, and Potosi further to the eastward. Oruro is the most important, its production being two-thirds of the total of the Republic. The Catarecagua lode, in this district, worked by the Huanuni Co., is from 2 to 8 ft. in width. The sorted ore contains about 50% black tin. About 100 metric tons of concentrates are produced monthly. The methods of dressing employed in this and other mines of Bolivia are very crude. In general, the Bolivian tin ores occur in connection with silver ores. Some mines at Chorolque afford both tin and bismuth. The progress in the Bolivian tin-mining industry in recent years has been due to the completion of the railway from Antofagasta, which was opened to Orūro in 1892. This reduced freight rates from \$2·50 per 100 kg. (by mules) or \$1·25 (by llamas) to about 62½c.

Smelting of the ores is carried on at certain places on a small scale in water-jacket furnaces, but the product is inferior, being contaminated with antimony and other metals. Some of the ores would probably pay as well if shipped away as concentrates. The concentrates of some ores, however, are so dirty that no merchant will purchase them except at a very low figure, for which reason the smelters are able to buy them at a very low rate and make money by saving on freight. Coal, chiefly Australian, costs about £11 per ton delivered at Oruro, which is a prohibitive price for the smelters, who consequently are limited to charcoal and llama dung. In the case of some remote mines as those at Potosi, which are 150 miles from railway, it pays better to ship bar tin than concentrates, owing to the saving in transportation. The Bolivian government charges an export duty of 50c. (silver) per 100 lb. of bar tin and 35c. per 100 lb. of concentrates; at present rate of exchange, these duties being about 9d. and 6·5d. respectively.

According to our own advices the shipments of bar tin from Bolivia in 1896

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* *Institution of Mining and Metallurgy*, Dec. 21, 1898.

amounted to 2,000 metric tons; in 1897 they were also 2,000 tons. In 1896 there were shipped 5,470 tons of "barrilla," or concentrates, and in 1897 6,200 tons. The barrilla assays from 50 to 75% tin.

China.—According to the report of the French Commercial Mission, the Province of Yun-Nan now exports annually 2,500 metric tons of tin. This is obtained from alluvial deposits at Kotchiou, 20 miles from Moung-tse. The product is shipped down the Red River to Canton.

Congo.—Tin-bearing rocks have been noticed along the Ubangi and Uelle (Djabbir), and there is evidence of the existence of workable deposits.*

Great Britain.—Report of the directors of the Dolcoath Mine, Ltd., for the half year ended June 30, 1898, showed a profit of £7,266, against £3,399 17s. 7d. in the previous half year. The quantity of ore treated was 38,089 long tons, against 37,686, and the average yield of black tin was 67·07 lb., against 63·78; while the price realized for black tin was £39 16s. 4d., against £37 17s. 4d. In the deepest level of the mine, 455 fathoms, ore of the value of £35 per fathom has been encountered. This is especially gratifying, since, from the 400 to the 440-fathom level the ore was of low grade. During this half-year the trials with the new concentrating machinery were completed, with the result that 27 Frue vanners have been purchased, of which 20 have already been erected. It has been proved definitely that they effect a large saving in the cost of treatment. The following table gives the statistics of production since the present company was formed:

Six Months.	Tin Ore Crushed.	Block Tin Sold. T. c. q. lb.	Produce per Ton of Ore.	Average Value per Ton of Ore.	Average Price per Ton of Black Tin.	Amount Realized.
	Tons.		Lb.	£ s. d.	£ s. d.	£ s. d.
December 31, 1895	28,717	1,015 5 1 16	79·19	1 7 8·36	39 3 5	39,769 2 5
June 30, 1896	30,015	1,029 19 1 10	76·86	1 5 6·33	37 4 0	38,311 6 5
December 31, 1896	33,712	1,009 7 0 25	67.08	1 1 10·20	36 9 2	36,830 18 2
June 30, 1897	35,879	1,021 17 0 22	63·79	1 1 7·27	37 18 7	38,760 7 2
December 31, 1897	37,686	1,073 2 1 9	63·78	1 1 6·78	37 17 4	40,636 19 8
June 30, 1898	38,069	1,140 7 2 22	67·07	1 3 8·94	39 16 4	45,221 15 1

Java.—The exports of tin from Java and Madura, in long tons, are reported as follows: †

	1894.	1895.	1896.	1897.
Private	5,768	4,526	4,825	5,436
Government	6,789	6,950	6,707	8,137

Straits Settlements.—According to the administration report of Perak for 1897, there was a falling off in the export of metallic tin compared with 1896, the figures being 20,702 tons for the former year and 22,542 tons for the latter. The Warden of Mines names the low price of tin in the early part of the year and the smaller number of coolies employed in mining as the two principal causes of the reduced output, and is of opinion that the latter is

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* X. Stanier. "The Geology of the Congo," *Transactions of the Institution of Mining Engineers*, September, 1898.

†British Foreign Office Report, May, 1898, No. 2,095, Annual Series.

the more important. The area of land available for alluvial tin mining in the Larut district is rapidly diminishing, but there are still large areas available in the less accessible portions of the Kinta district, and also in Batang Padang, Kuala Kangsar, and Upper Perak.

According to the *Straits Times*, tin mining has made a fresh start in Malacca in 1898. Four mining leases for 299 acres of land in the Mukims of Ayer Panas and Kesang were issued to a syndicate of Chinese, who have obtained very good ore assaying as high as 75% tin. Other Chinese prospectors are now at work in the Ayer Panas, Chinchin and Durian Tunggal districts. M. François Raaymaakers, of the Singkep Tin Co., has obtained from the government a monopoly of the right to dredge for tin ore within 3 miles of the shore for a distance of 10 miles from Kuala Linggi southward for a period of five years.

W. H. Derrick described* lode tin mining by the Pahang Corporation, Ltd., at Kuantan, Pahang, one of the federated Malay States. Although alluvial tin has been found over a large area, the payable lodes appear to be limited to about 10 sq. miles. The lodes are from 50 to 2,000 ft. apart, vary in width from 2 to 10 ft., and yield from 1 to 15 lb. black tin per 2,240 lb. The country rock is slate underlain by granite. The lodes cut through the slate into the granite. In general the geology of the district is similar to that of parts of Cornwall. The lodes carry both tin and copper ore, the latter assaying 25 to 30 oz. silver per ton, but with depth it appears to be rapidly giving place to tin. The average cost of mining, exclusive of permanent shafts, is about 5s. per ton. The company has two mills of 40 and 20 stamps respectively. The stamps are of the usual California type, weighing 850 lb. and making 90 drops per minute; they crush about 2·25 tons of ore per day. About 2,500 tons are being treated monthly, yielding 85 to 95 tons of tin oxide, an average of about 3·5% per ton. In dressing, the usual concave and convex buddles and other apparatus used in Cornwall were employed at the beginning, but these have been replaced by Frue vanners with 4×12-ft. corrugated belts, which take the pulp without classification directly as it comes from the stamp. With ores carrying a large percentage of pyrites these tables give heads containing 25 to 45% tin oxide, which are sent directly to the roasting furnaces, and tails containing 3 to 5 lb. per long ton. Three tables are required for every 10 stamps. Ordinary reverberatory furnaces are used for calcining; with wood at 10s. per cord (128 cu. ft.) the cost of roasting is 4s. per ton of concentrates. The total cost of dressing is 5·5s. per ton of ore crushed. The black tin always exceeds 70% metal, the impurities being oxide of iron, a little silica, and less than 0·12% copper. Middlings are recrushed by means of two pulverizers, which work up about 250 tons per month. This product is redressed on Frue vanners, one for each pulverizer, with plain belts 4×12 ft., with middlings assaying 10% black tin, the tails assay only 6 lb. per ton, while the concentrates are very clean.

Tasmania.—The production of tin in this colony in 1898 was 1,972 long tons, the more part being obtained from Mt. Bischoff, as usual. The Mt. Bischoff Tin Mining Co. realized a profit of £25,271 during the half year ending June

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* *Institution of Mining and Metallurgy*, 1898.

30, 1898. The amount of ore smelted was 1,347 tons 7 cwt., yielding 926 tons of metal, of which 368 tons, yielding 259·5 tons of metal were purchased and the remainder was produced by the Mt. Bischoff mines. The average assay of the refined tin was 99·86% and the slag 5·3%. The mine produced 1,026 tons of dressed ore at a cost as follows: Mining, including new works, maintenance and other expenses, 2s. 3d.; hauling, filling and emptying trucks, 6$\frac{1}{2}$d.; crushing and dressing, maintenance of plant, etc., 1s. $\frac{7}{8}$d.; slime sheds, 2$\frac{1}{4}$d.; ringtail sheds, 2$\frac{1}{2}$d.; management and supervision, 10$\frac{1}{8}$d.; plant, including all machinery, 3$\frac{1}{6}$d.; development and progressive work, 1$\frac{3}{5}$d.; waterworks, $\frac{7}{8}$d.; ore bagging, $\frac{7}{8}$d.; contracts and sundries, 2$\frac{1}{2}$d.; stores, 5d.—total, 6s. 3$\frac{3}{10}$d.

Apart from the Mt. Bischoff mine, tin ore is found in the Ringarooma valley at Branxholm, Derby, Moorina, Pioneer and Mt. Cameron, where the Brothers' Home, Briseis, Brothers' Home No. 1, Arba and Ormuz mines are situated. The east coast deposits have been proved to be extensive, and a discovery of tin ore on the Great Mussel Roe River has recently been reported. The Anchor Tin Mining Co. did not in 1898 realize the success that was anticipated. Mr. A. Lee, chairman of the company, spent some time at the mine in the early part of the year and put through the mill more than 500 tons of ore taken at random from all the working faces. As a result he reported the average value of the ore to be in excess of 7s. per ton, and estimated the cost of treatment at 4s. Later work was suspended as being unprofitable, although in the period immediately preceding the closing down the cost of mining and dressing was only 2s. 11d. per ton. The Anchor dike was examined by George Thureau, formerly government geologist of Tasmania, and later by A. Montgomery. According to their reports the company formerly working the property had obtained an average of 0·973% black tin from 30,734 tons of stone crushed. Later returns showed an average of 1·09%. The results of the Anchor Tin Mining Co. gave only 0·02%.

THE TIN MARKETS IN 1898.

New York.—The year 1898 will long be memorable in the history of the tin market. In this one year the value of the metal recovered the loss of the previous five years. It is important to note that this movement was not the result of speculative operations, but was directly caused by the conditions of demand and supply. The stocks in London dwindled to an unprecedentedly low figure, and stocks in America were also depleted. Consumption in this country increased enormously and at the end of 1898 was estimated at almost 3,000 tons per month, or three-fourths of the world's production. In spite of the stimulus of the higher values, production was but slightly increased. This is due to the fact that the new tin fields have not been sufficiently prolific or accessible. The decrease in the production through the giving out of the old mines was therefore not compensated by the addition from the new sources. Besides coolie labor was limited, and owing to the doubled price of rice the cost of labor was increased. While the American market was, as before, dependent upon the London and Eastern markets, prices here did not in every instance follow those of Europe, owing to the large stocks at times accumulated here. The New York market opened at 13¾c., and during January fluctuated between that price and 14c. During the following four months it advanced, with occasional reactions, to 15c., and during June and July to 16c. In August and September it went to 16¼c. During the last three months it fluctuated considerably, but persistently recovered all reactions and moved upward, closing at 19c.

AVERAGE MONTHLY PRICES OF TIN IN NEW YORK.

Year	Jan.	Feb.	Mar.	April	May.	June.	July.	Aug.	Sept.	Oct.	Nov.	Dec.	Year
	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.
1893	19·99	20·30	20·71	20·81	19·96	19·76	19·15	18·81	20·14	20·84	20·61	20·67	20·15
1894	20·16	19·60	19·09	19·75	20·21	19·75	19·22	19·22	16·27	15·35	14·56	13·81	18·08
1895	13·25	13·35	13·20	14·00	14·65	14·15	14·40	14·35	14·45	14·65	14·40	13·91	14·05
1896	13·02	13·44	13·30	13·34	13·54	13·59	13·63	13·49	13·15	12·94	13·09	12·96	13·29
1897	13·44	13·59	13·43	13·34	13·44	13·77	13·89	13·80	13·98	13·88	13·79	13·71	13·67
1898	13·87	14·08	14·38	14·60	14·52	15·22	15·60	16·23	16·03	17·42	18·20	18·30	15·70

London.—The year 1898 opened with a large visible supply, amounting roughly speaking to about 31,000 tons, and the English stock—which is the principal medium of speculation—was practically in a few hands; shipments from the Straits continued to come forward in large quantities, and the position of the bull party did not seem at all enviable. The opening prices were £63 1s. 3d. for spot, £63 17s. 6d. three months, and throughout the month the fluctuations were confined to a narrow limit, the extremes being £62 12s. 6d. and £63 5s. cash; £63 6s. 3d. and £63 18s. 9d. three months, closing at £63 2s. 6d. and £63 15s. respectively. Consumption was on a good scale, and the Americans, whose stocks had become very bare, took considerable quantities direct from the Straits.

Statistics published at the commencement of February showed a further increase in European stocks of about 1,500 tons, and the market for Straits opened at £63 cash, £63 15s. three months, from which point they steadily rose to £67 5s. and £64 17s. 6d. A temporary reaction then took place, and values backed to £64 cash, at which figure the market remained almost stationary for a few days, and the public trading shrunk to a small volume. Toward the end of the month a better tone was manifested, and the closing figures were £64 12s. 6d. cash, £65 2s. 6d. three months. Consumption continued very good, especially in America, and deliveries were on a very large scale, causing a shrinkage in stocks of 1,718 tons, which was the first important reduction noticeable for a considerable time.

The month of March was fraught with trouble in the far East, and the crisis between the United States and Spain was coming to a head. This kept speculation from developing to any great extent, and the tin market remained quiet in company with other articles. The opening values were £65 cash, £65 10s. three months; there was then a sharp decline to £64 8s. 9d. and £65 1s. 3d., but the recovery was almost as rapid and prices rose to £65 7s. 6d. and £66, and closed with buyers at £65 5s. cash, £65 15s. three months.

Statistics compiled at the beginning of April made it evident that consumption was going ahead in a very healthy manner, and this attracted the attention of many outside operators; nevertheless, values moved slowly and after a temporary dip to £64 12s. 6d. cash, and £65 7s. 6d. three months, closed at £65 13s. 9d. and £66 7s. 6d. The chief incident during this month was the large shipments of Straits tin from London to America, where consumers had allowed their stocks to run low.

May opened with a reduction of 2,585 tons in European stocks, and the price of Straits was £65 13s. 9d. cash, £66 5s. three months, but after declining to £65 6s. 3d., took a decided turn for the better, and with almost daily improvement crept up to £67 15s. cash, £68 10s. three months, but closed at a decline of about 10s. from these figures.

Statistics published on the first of June revealed a further falling off in stocks of about 1,200 tons, but the prices under realizations took a downward turn to £67 5s. cash, £67 16s. 3d. three months. At this range there was considerable buying by speculative investors, who took up warrants rather freely, and as the demand from America continued good, and shipments from the East were meager, prices soon recovered and advanced rapidly to £68 18s. 9d. cash, £69 7s. 6d. three months. Determined attacks were then made by prominent dealers who were against the rise, but these large sales only depressed values slightly, and they had the doubtful satisfaction of selling large quantities at prices which soon showed handsome profits to their rivals. The month closed strong at £70 11s. 3d. spot, £70 10s. forward.

July commenced with cash tin ruling at £71, three months at £71 2s. 6d., but the bulls were able to obtain large quantities at cheap rates owing to the somewhat doubtful tactics indulged in by their opponents, who threw large lines on the market with the idea of breaking the prices, but these parcels were readily absorbed by those engineering the rise. The statistics again showed a decrease of 1,169 tons; it was therefore evident that consumption was rapidly overtaking production. The market soon rallied to £72 1s. 3d. and £72 2s. 6d., only to be depressed again to £70 11s. 3d. cash, £70 17s. 6d. three months, but when the ostentatious selling ceased prices again went ahead, and the final rates were £71 8s. 9d. spot, £71 16s. 3d. three months.

European figures made up at the commencement of September showed a decrease of 678 tons, and the London stock was only about 10,500 tons. Prices pursued their upward course, and the opposition seemed powerless to stem the tide until £75 was reached for cash, at which height many speculative holders, and even some of the bull clique, sold large quantities, and so the market gave way to £73 7s. 6d. cash, and £73 11s. 3d. forward. The prices in the Straits, however, were above the London parity, and a recovery was made before the end of the month to £73 17s. 6d. for all positions. The month of September opened at £73 12s. 6d. spot, £73 15s. forward, but as there was a decrease of only about 250 tons in the stocks, the public, whose appetite was thus somewhat checked, left the market for a time to the sport of the rival factions, who caused the fluctuations to be wild and varied; at times forward prompts were selling at a backwardation of 5s. per ton under the cash price; £72 3s. 9d. was the lowest price accepted for cash, and then a turn for the better came and values rose with only moderate variation to £74 5s. cash, and £74 10s. three months; these were the closing figures for the month.

October opened with a good consumptive demand, but with European stocks only down some 200 tons. However, the month was destined to be a record breaker, and on large trading prices went up by leaps and bounds. A rise of about £8 5s. per ton had been established, which showed an appreciation on the London stocks of close on £75,000, so that the bull operators certainly had every cause to be satisfied, and sold good lines, which sales in their turn caused a healthy decline from the highest point. One of the chief reasons for the sudden leap in price was the quantity of call options that had to be covered, and buyers had to dance to any tune the bulls played. The top price was reached on October 28th, when cash realized £82 10s. and three months prompts were worth £83. The greatest excitement occurred when values had reached about £80.

When the month of November opened prices again advanced to £83 15s. cash, and £84 5s. three months, but on the political outlook becoming very uncertain a good deal of selling took place on the part of those who wisely wished to secure the handsome profits which had been thrust upon them, and this caused another reaction to £82 cash, £82 10s. three months, but at this point the bull party came to the rescue and stayed the fall. The month, however, was marked with violent fluctuations, and after improving to £84 2s. 6d. cash, and £84 10s. three months, some of those chiefly interested in the rise sold very heavily, and it looked as if the market was going to pieces entirely. The selling at times was quite forced, and savored somewhat of a ruse to catch unwary bears, but when 2,300 tons had been disposed of and the value of cash stood at only £80 5s., a stronger tendency was evident, and a small recovery was made to £84 7s. 6d. cash, and £84 17s. 6d. three months; but before the month was out values had fallen about 15s. per ton, and closed at £83 11s. 3d. spot, £83 18s. 9d. three months.

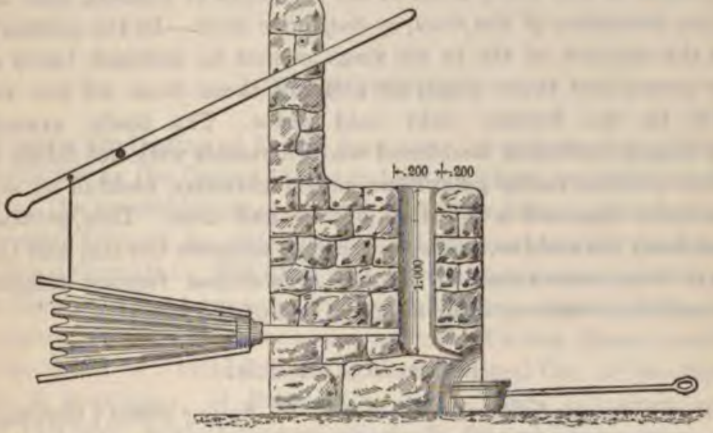
The statistics published at the beginning of December showed an increase in European stock of 132 tons, and the article was moved daily quite at the will of those controlling the stock; the fluctuations were frequent, and after opening at £83 10s. cash, £84 2s. 6d. three months, declined to £80 16s. 3d. and £81 7s. 6d., then recovered to £82 6s. 3d. and £82 18s. 9d., but eased off again to £81 and £81 13s. 9d. There was then a good deal of interest shown in this metal, and buyers having the market well in hand soon pushed prices again to £83 2s. 6d. for spot, and £83 17s. 6d. for three months at the close.

NOTES ON THE METALLURGY OF TIN.

BY HENRY LOUIS.

THERE is practically nothing new to be chronicled in the metallurgy of tin, the position of which has remained unaltered. Attention has been directed chiefly to the better dressing of the ore as a necessary preliminary to metallurgical treatment, and modern machinery is at last being used for this purpose. Frue vanners have, for instance, been used at Dolcoath, with satisfactory results, and other Cornish mines appear to be also experimenting in similar directions. On the Continent the electromagnetic separation of tinstone and wolfram is said to have been worked out satisfactorily, but the process has not yet been used in practice.

The tin belt that traverses the Spanish district of Galicia and the adjacent Portuguese province of Tras-os-Montes is still being explored, although the results so far are not satisfactory; it would seem as though some of the mines on the Portuguese side are now considered the more promising. There is no smelting carried on now in this region, although the ore used to be treated locally in small furnaces, a section of one of which, kindly supplied by Mr. A. H. Bromly, is here shown. These furnaces were from 3 to 5 ft. high and 8 to 10 in. inside diameter. They were worked by small hand-bellows, often circular, and about 2 ft. in diameter, the blowing being done by one man. These furnaces were capable of producing about 336 lb. of tin per day, or about 2,240 lb. per week, consuming during the latter period about 20 large sacks of charcoal made from brushwood and roots—trees being very scarce throughout the district. The slag was always very foul, carrying up to 30% of tin, and had to be crushed and resmelted, and the smelting operation was always somewhat troublesome, due perhaps to the poor quality of charcoal available; unless the tuyère was kept very low down in the furnace the tap-hole was apt to "freeze up." This method of smelting seems to have fallen into disuse about 30 years ago. The owners of smelting furnaces usually paid the miners (tributers) a fixed price for the clean black tin produced, this price ranging from 10c. to 15c. per kg. ($5 to $7·50 per cwt.). The black tin was obtained by carefully breaking and hand-picking the richer portions of the veinstuff, while other parts were crushed small by hand with hammers and washed in small wooden tyes. The process was obviously primitive in the extreme, and the scale of operations seems always to have been very small.



Tin Residues, Electrolytic Treatment of.[*]—The total loss in the manufacture of tinned iron is stated to amount to thousands of tons annually, the difficulties

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[*] *Ind. Electro-Chim.*, 1898, II., pp. 2-3; *Science Abstracts*, 1898, I. (7), p. 434.

in the treatment of the residues arising from their large bulk and from the presence, under the layer of tin, of an alloy of tin and iron. In some works caustic soda is used as electrolyte; in this way the tin is dissolved, but the iron left has little value, owing to the relatively large proportion of tin admixed with it. In a recent process the scraps of metal are packed in wooden frames, which are used as anodes, while the tin is deposited upon cathodes of tinned copper. Dilute sulphuric acid is used as the electrolyte, and the iron is eventually completely dissolved. Goldschmidt, at Essen, Germany, recovers electrolytically tin from scrap, which contains an average of 2% Sn. An electrolytic process is also being tried at the tin works at Tostedt, near Hamburg.

Tin Slags, Extraction of Tin from, by Sulphuric Acid.—In the ordinary furnace processes the amount of tin in tin slags cannot be reduced below about 4%. E. Bohne granulates these slags by pouring them from off the surface of the metal in the furnace into cold water. The finely granulated or powdered slag is treated in lead-lined wooden vessels with hot dilute sulphuric acid, which yields a partly gelatinous, partly granular, residue of silica, containing a little tin, and a solution of tin and iron. This solution, after separation from the residue, is electrolyzed to separate the tin, and the mother liquors are then evaporated to furnish crystallized ferrous sulphate. The silicious residue is used up in making the bed of the furnaces.*

ESTIMATION OF TIN.

Titration of Stannous Salts with Iodine.—S. W. Young states† that the reaction between stannous chloride and iodine, by which the former is oxidized, takes place smoothly in acid solution, and may be utilized for the determination of tin. The iodine solution is standardized against potassium bichromate with the intervention of a stannous chloride solution of known value. The statements are made that "thiosulphate cannot be satisfactorily used in acid solution to titrate against iodine," and that "the standard of iodine by thiosulphate is likely to be a trifle high;" but no authorities are quoted nor experimental data offered in support of these assertions.

Electrolytic Determination of Tin in Tin Ores.—According to E. D. Campbell and E. C. Champion,‡ tin ores are decomposed by fusion with sodium carbonate and sulphur, the resulting sulphostannate transformed to the double oxalate of tin and ammonia, and the latter electrolyzed. The transformation to the double oxalate is accomplished as follows: The sulphostannate solution is neutralized with hydrochloric acid, and sodium peroxide is added. This oxidizes the tin to stannic chloride. The sulphur is then filtered out, and ammonia and ammonium oxalate are added to the solution. From the solution so obtained the tin is satisfactorily deposited by a current of 0·1 ampère and 4 volts.

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* *Berg- und Hüttenmannische Zeitung*, 1898, LVII, 203.

† *Journal American Chemical Society*, XX, p. 809.

‡ *Ibid.*, XX., pp. 687-690.

TUNGSTEN.

There was a production of 78·865 long tons of tungsten ore, valued nominally at $17,398 in the United States in 1898, of which 30 tons were from South Dakota and the remainder from Arizona and New Mexico, chiefly the former. The increased demand for this mineral led to considerable prospecting and several new discoveries. The domestic production was employed chiefly in the manufacture of ferro-tungsten for use in the preparation of tungsten steel. Ferro-tungsten is made by three concerns in the United States, namely, Stein & Boericke, Ltd., of Philadelphia, the Chrome Steel Co., of Brooklyn, N. Y., and Ash & Deininger, of Phœnixville, Pa. There are three commercial grades of this alloy, containing respectively 37, 50 and 75% tungsten. Metallic tungsten is made by one firm which reported a product of 33,200 lb. in 1898. There were wide fluctuations in the market price of tungsten and ferro-tungsten, the former selling at 60c.@$2 per lb. during the year, and the latter, containing 37% tungsten, at 27@56c. per lb.

There was an equally wide range in the price of tungsten ore, which varied from $3 to $6 per unit of tungstic acid, according to grade and quantity offered. At the beginning of the year the ore mined in Arizona, assaying 70% WO_3, brought $100 per long ton f. o. b. cars; by June the price had risen to $450; but later it fell off. These prices were considerably in excess of the offers of European buyers, one of whom was willing to take from 300 to 500 long tons, delivered at the rate of 30 to 40 tons per month, assaying at least 60% WO_3, at $1·80 per unit at port of shipment.

The domestic production of tungsten ore in 1898 was derived chiefly from Arizona. Small amounts were mined in New Mexico and South Dakota. The Connecticut mines produced about 400 tons of low grade ore, which is held at the mines awaiting dressing and is not included in our statistics. The new dressing works will probably be completed in 1899. The Arizona mines occur in the Dragoon Mountains, about 6 miles from Dragoon Summit, to which point the ore is carted for shipment by railway. The principal mines at this place have been purchased by Stein & Boericke, Ltd., of Philadelphia. The wolfram deposits of South Dakota occur in the Archean schists near Hill City and Custer in the Black Hills. At the former place the Tungsten Mining Co. of Omaha gets huebnerite, assaying 60% WO_3. About 5 miles N. W. of Custer the mineral is found in the Astor and Astor No. 3 mines. Early in January, 1899, new deposits of the mineral were found in the Harrison and Durango mines at Lead City, and 17 tons of low grade ore were shipped to a mill at Idaho Springs, Colo., for a test as to the practicability of concentrating it. Wolfram ores were found in small amounts at various places in Colorado in 1898, but no shipments were made. Among other places it was located near Summit, in the southern part of the State, and near Salida, where huebnerite was found in a vein which is being exploited for copper.

THE OCCURRENCE AND PRODUCTION OF WOLFRAMITE IN ARIZONA.

BY W. P. BLAKE.

THE discovery of the existence of wolframite-bearing veins in Arizona was made and reported by me in 1896, mention having been made of the deposits at Arivaca, south of Tucson, in my report as director of the School of Mines for that year. In May, 1898, I reported the discovery of other and more extensive deposits in quartz veins in the granitic rocks of the Dragoon Mountains at Russellville, about 6 miles north of Dragoon Summit, on the Southern Pacific Railroad. At both of these localities the mineral occurs in workable quantities sufficient to form the basis of a tungsten mining industry, although the quantity is by no means unlimited, and there is no danger of the supply exceeding the demand. The Arizona wolframite is the manganiferous variety originally described by Riotte as hübnerite,* and is quite free from admixture of noxious minerals. The color is brownish red, but by long-continued weathering the surface becomes less deeply colored, and assumes a bronze-like luster, like the dull oxidized surface of copper. The sp. gr. is 7·140. The prevailing form of occurrence is in large tabular prismatic blocks, or thick plates, often somewhat radial, penetrating the solid gangue of white quartz. These masses vary in size from a fraction of an inch to 6 in. in diameter. This massive character, and the ease with which the mineral can be separated from the gangue, permit the hand-sorting of considerable quantities for shipment, especially from the numerous weather-worn masses detached from the veins by long-continued weathering. There are, besides, many loose blocks of the hübnerite lying in the soil on either side of the croppings, from which several tons have been already secured for shipment. One mass weighing 500 lb. has been reported.

The veins are of the simplest type of quartz veins, nearly vertical, and generally traverse the granitic gneiss in the direction of the rude structural bedding. They vary in width from a few inches to 3 ft., and possibly even more in some places. There is much branching and irregularity. While the veins are continuous for considerable distances, in one case for the length of two or three claims, the hübnerite is not so persistent, the mineral being abundant in places for short distances, and then thinning out. Insufficient work has been done yet to show whether the croppings of the mineral are the ends of chutes

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* *Reese River Reveille*, Nevada, 1865; also, H. Credner, in *Berg- und Hüttenmannische Zeitung*, No. 24, p. 370, 1865.

or chimneys, or merely lenticular bodies. It can hardly be said that there is any general prevailing form of distribution of the hübnerite in the substance of the vein. It occurs in patches or bunches, sometimes centrally disposed, with quartz on each side, sometimes disseminated from side to side, and again disposed in layers or bunches contiguous to the walls, leaving a central mass of white quartz. The whole aspect of the veins is that of the segregated type, rather than of gradual accumulation upon the walls of an open fissure.

Hübnerite compares most favorably with any wolframite as a source of tungsten. It is yet to be shown whether, as an addition to steel, it is not far superior to the ordinary wolframite. Much of the wolframite from other localities, notably that from Cornwall, England, besides its close association with cassiterite, is more or less mixed with sulphides and arsenides, from which noxious elements it must be freed before it can be added to steel. It does not yet appear that the clean manganese tungstate has been produced in sufficient quantity, commercially, to establish a distinctive place for itself in steel-making. Its value, as compared with ordinary wolframite, has yet to be studied and made known.

The Russellville veins have been purchased in part by Messrs. Stein & Boericke, Ltd., of Philadelphia, and in part by the Chrome Steel Co. of New York. Both of these concerns are now actively developing the veins. The first-mentioned has shipped several carloads of the crude ore to the Arizona School of Mines at Tucson for experimental demonstration of the efficiency of jigs for concentration, and to determine the best arrangement of crushing and concentrating for working this peculiar ore. The experiments have been entirely successful. By the preliminary coarse crushing and jigging the bulk of the coarse-grained hübnerite was separated in the first compartment of top discharge jigs, while the fines were best secured in bottom discharge jigs, a thick bedding of clean, coarse ore being used on the beds. Both sizes of product ran very high in tungstic acid, the first grade containing about 70%, while the second grade from the second and third top discharge averaged about 50%. The coarse tailings contained less than 1%. There was, of course, a considerable amount of quartz containing some attached ore, "chats," which require recrushing to liberate the wolframite, and a subsequent concentration on some form of concentrator, such as the Frue vanner, or Wilfley table. The deportment of this wolframite ore under concentration is very much like that of cassiterite, and the methods and appliances in use in Cornwall for dressing the stone are applicable in this case. So far, however, the wolframite, not being associated with any sulphides or arsenides, does not require calcining.

It is known, however, that in one or two of [illegible] wolframite-bearing veins there are portions where iron pyrites occur accompanied by gold. This is true particularly of the Arivaca veins where they were originally located and worked for gold. In these veins the problem is presented of securing the free gold from the surface or oxidized ores and collecting the finely crushed wolframite as a second product, and at one operation. When the lower levels on the veins are reached and the auriferous pyrites presents itself in its original condition, the separation will be more difficult and expensive. Here, however, magnetic separation may be employed. Samples of the hübnerite sent from the Arizona School of Mines to the Wetherill Separating Co., of New York, have been tested with remarkable results. Under the intense magnetic energy of the Wetherill magnet the manganese tungstate responds like the tungstate of iron and may be lifted easily from the inert pyrites. From the trials so far made it would appear that magnetic separation may be resorted to successfully, not only with mixed ores, but with ores of tungsten in a quartz gangue, especially in localities where water for jig separation is scarce.

Deeper work on the veins at Russellville has shown increasing amounts of the lime tungstate (scheelite), which is so mingled with the quartz as to escape casual observation. Fluorspar of a deep purple color has also appeared in small quantity in one of the veins.

The clean product of the concentration was sacked and shipped by Stein & Boericke to their establishment in Philadelphia. In addition to these shipments the owners of claims at the Dragoon localities have been able to cull some clean high-grade ore from the surface croppings, masses weighing from 10 to 100 lb. or more have been found with but little attached quartz. This source of clean ore is practically exhausted, though in mining there is a small amount of massive ore broken out which can be separated by hand.

In regard to prospective quantity of ore and commercial supply it must be understood that the wolfram occurs in detached bunches and pockets here and there, not in continuous sheets, and in mining it is necessary to break considerable stetches of barren ground to reach the ore-bearing portions. Thus the cost of mining per ton of clean ore is large. At the time of the discovery about $80 per ton of 2,000 lb. was offered for clean 70% ore, but as this did not bring out much of a supply the price was advanced to $120 to $200, and even as high as $400 per ton for small lots.

Mention should be made of a locality at Separ, N. M., from which several shipments were made in 1895 and 1896. The ore was the manganiferous variety of good quality, and was sold for $80 per ton. This locality is regarded as exhausted.

ZINC AND CADMIUM.

By Walter Renton Ingalls.

In The Mineral Industry, Vol. VI., I referred to the fact that the manner in which zinc smelting is carried out in Missouri and Kansas, where excellence in the metallurgical practice is not so much striven for as large capacity at a small outlay of capital, made it easy to build new plants, and in view of these circumstances it was only to be expected that the high price ruling for zinc during 1898 would lead to the erection of many. These have sprung up on every hand, especially in the natural gas field of Kansas, where by May 1 there will probably be not less than 12,000 retorts. George E. Nicholson built a 600-retort plant at Iola, Kan., which went into operation in January, 1899, and 600 more retorts are to be added before summer. W. & J. Lanyon built new works at Iola. S. C. Edgar put up a 1,200-retort plant at Cherryvale, Kan., which is expected to be in operation during the summer of 1899. The Robert Lanyon's Sons Spelter Co., which has a 2,400-retort plant at Iola, built a new plant of 1,200 retorts at La Harpe, Kan. The Midland Coal and Smelting Co. started new works at Midland, Kan. The L. T. McCrae Co. was organized to build a 1,200-retort plant between Iola and La Harpe. The Cherokee Smelting Co., a new concern, leased from the Cherokee-Lanyon Spelter Co. its works No. 12 at Cherokee, Kan., which had been idle two years, and put them in operation. The Girard Smelting Co. took over the two works of the Girard Zinc Co. and the Kansas Zinc Mining and Smelting Co., at Girard, Kan., which had also been under lease to the Cherokee-Lanyon Spelter Co.

Outside of Kansas and Missouri there were also large additions to the productive capacity. The Fairmount Zinc Co., of Fairmount, Ind., began operations in January, 1899, with 300 retorts, and expects to add 300 more before the end of the year. James La Tourette built a new plant at Marion, Ind., which, together with the old plant of the Marion Zinc Co., is now operated by the Columbia Zinc Co. The Swansea Vale Zinc Co., of Sandoval, Ill., to which the citizens of that place gave a bonus, started up one furnace and made a little zinc, but closed down in January, 1899, having lost money. The Humphrey Spelter Co. rebuilt the works at Upland, Ind., formerly owned by the Indianola Zinc Co.

Large additions were made to the plants of many of the older producers. W. & J. Lanyon added 600 retorts in February to their Iola works and 600 in

November, making a total of 1,800 in the plant. The Empire Zinc Co. added one block of retorts to its Joplin works and a new furnace to its North Chicago works, doubling the capacity of the latter. This company is also installing a Wethey roasting furnace at each of its works, these furnaces embodying some new features. The Robert Lanyon's Sons Spelter Co. installed a new Ropp furnace at its Iola works, and now has two of them in operation at that place. The works at Wenona, Ill., formerly owned by Humphrey, Chipman & Co., were taken over by the Wenona Zinc Co. The Collinsville Zinc Co., of Collinsville, Ill., increased its capacity by 50%.

There is prospect of a further large increase in the productive capacity in 1899. Besides the new works mentioned above as being under construction, the Palmer Smelting Co. is planning to erect a plant at Iola. The Cherokee-Lanyon Spelter Co., having dismantled its works No. 6 at Scammon, Kan., is rebuilding them at Iola, at which place it will be making spelter in 1899. Works No. 5 and No. 11 of this company at Pittsburg, Kan., which some time ago suspended operations owing to the competition of the plants in the natural gas district, resumed work early in 1899, stimulated by the increase in the price of spelter. The Prime Western Spelter Co. has already put in operation a 600-retort plant at Iola, Kan., and expects to build 600 additional retorts later in the year.

A noteworthy feature of the early part of 1899 has been the organization of the Lanyon Zinc Co., with a capital of $3,000,000, which took over the gas smelteries of the Robert Lanyon's Sons Spelter Co. at Iola and La Harpe, the gas smeltery of W. & J. Lanyon at Iola, and the coal smeltery of the same concern at Pittsburg, Kan.; also all the gas lands and leases owned by the Lanyons and those owned by the Palmer Oil Co., the total amounting to over 118,000 acres. The Palmer Oil Co. is largely interested in the new zinc company. It owned a large part of the gas and oil lands between Iola and La Harpe, on which it had drilled 18 wells, all of which are gas wells, one of them being said to have a flow of 12,000,000 cu. ft. per 24 hours. The management of the new company will be in the hands of W. & J. Lanyon.

In the Eastern districts there was less activity in 1898 than in the West, the producers of New Jersey, Pennsylvania and Virginia being makers of a very high grade of spelter, which, or at least a considerable part of it, commands an extra price in the market. For example, in August, 1898, while prime Western spelter was obtainable in New York at 4·5c. per lb., New Jersey high-grade spelter was fetching 6·75c. The New Jersey Zinc Co., which controls this business, is also the principal producer of zinc oxide in the United States, and in 1898 commenced, near Hazard Pa., nine miles east of Mauch Chunk (establishing there a new town to be called Palmer), the erection of a new zinc oxide plant, on a fairly large scale, with which it is proposed to take care of new business. The works at Jersey City and Newark, N. J., and Bethlehem, Pa., will continue to be operated, but it is contemplated to make eventually the new works the center of operations. Some experimental spelter furnaces are also to be erected at Palmer. Aside from the New Jersey Zinc Co., which in 1898 made zinc white at Jersey City, Newark and Bethlehem, this pigment was produced by an allied company at Mineral Point, Wis. Page & Krause, of St. Louis, Mo., also make a small amount of oxide, not more than 500 tons per annum, while the Standard Oil Co. recovers a little as a by-product at Williamsburg, Brooklyn, N. Y. The only other producer in the United States is the American Zinc-Lead Co., of Cañon City, Colo., which makes a zinc-lead pigment. The average value of zinc oxide at the works in 1898 was 3·4c. per lb., against 3·21c. in 1897, the improvement being accounted for in part by better freight rates and discounts.

A zinc product made in the United States in 1898 for the first time was zinc sulphate, of which 145 short tons were recovered by the Consolidated Kansas City Smelting and Refining Co. as a by-product from zinky silver-lead ores treated at its works at Argentine, Kan. There is likely to be a successful future for this industry. Although the consumption of zinc sulphate in the United States at present is not large, a considerable quantity of zinc chloride is consumed as a preservative for wood, one railway company reporting a consumption of 3,000 short tons per annum for this purpose.

The sheet zinc rolling industry was also prosperous in the United States in 1898, the Matthiessen & Hegeler Zinc Co., of La Salle, Ill., which is the most important factor in this business, having begun the installation of improvements and additions to its plant, which will greatly augment its capacity, to meet the increase in the consumptive demand.

PRODUCTION OF SPELTER IN THE UNITED STATES.

States.	1893.	1894.	1895.	1896.	1897.	1898.
Illinois and Indiana	29,725	28,948	33,748	31,656	38,680	46,693
Kansas	22,085	26,060	25,916	36,566	33,395	38,543
Missouri	13,737	12,000	10,726		18,412	21,063
South and East	10,708	6,996	11,468	9,415	9,900	7,805
Total tons of 2,000 lb.	76,255	74,004	81,858	77,637	100,387	114,104
Total tons of 2,240 lb.	68,525	66,074	72,088	69,319	89,623	101,879
Total metric tons	69,159	67,135	74,245	70,432	91,071	103,514

PRODUCTION OF ZINC OXIDE IN THE UNITED STATES.

Year.	Quantity.		Value.		Year.	Quantity.		Value.	
	Short Tons.	Metric Tons.	Totals.	Per Short Ton.		Short Tons.	Metric Tons.	Totals.	Per Short Ton.
1895	22,690	20,498	$1,588,300	$70·00	1897	26,262	23,825	$1,686,020	$64·20
1896	15,863	14,391	1,189,725	75·00	1898	32,747	29,708	2,226,796	68·00

IMPORTS OF ZINC AND ZINC OXIDE INTO THE UNITED STATES. (IN POUNDS.)

Year.	Sheets, Blocks, Pigs, and Old.		Manufactures.	Total Value.	Oxide.	
					Dry.	In Oil.
1894	512,932	$17,271	$12,342	$29,613	2,850,771	113,540
1895	864,113	29,352	12,182	41,545	4,546,038	120,343
1896	856,044	25,904	15,728	41,632	4,572,781	311,023
1897	2,557,341	95,883	19,431	115,314	5,564,753	502,357
1898	2,741,361	109,626	13,447	123,073	3,342,235	27,850

EXPORTS OF ZINC AND ZINC ORE FROM THE UNITED STATES. (IN POUNDS.)

Year.	Ore and Oxide.		Plates, Sheets, Pigs, and Bars.		Manufactures.	Total Value.
1894............	....	$5	3,621,934	$144,278	$99,418	$243,701
1895............	48,000	1,008	3,141,285½	155,975	50,126	207,109
1896............	4,648,000	47,408	20,300,169	1,013,620	51,001	1,112,029
1897............	*a* 22,220,907	315,490	28,490,662	1,356,538	71,021	1,743,049
1898............	*b* 31,413,859	552,064	20,998,413	1,033,959	141,932	1,727,955

(*a*) Includes zinc oxide, 3,718,507 lb. ($104,140). (*b*) Zinc oxide, 7,849,059 lb. ($252,194).

(By A. Heckscher): "The zinc industry during 1898 was active and prosperous, especially in this country, although the efforts made in 1897 to maintain prices on a remunerative basis, which failed at the time, were not renewed. Gradually it is becoming apparent that our matchless resources in the way of raw materials, our transportation facilities and the enterprising spirit of our people must give us a commanding influence in the exploitation of many industries—and zinc is one of them. At present (January 7, 1899), not so much by reason of more economical processes in use here, but rather because of the price advance abroad, following upon the exhaustion of stocks and the scarcity of fresh supplies, we are in a position to export zinc, zinc ore and zinc oxide.

"The foreign works have always followed the practice of making ore contracts for years in advance on a sliding scale. Even this has not availed in the present contingency. The impetus given to our market by the advancing foreign price for spelter has been reinforced by large requirements here, and the exceptional expansion which has taken place in our production has been readily absorbed. Ore prices in the Joplin field have been high and out of reach at times. Had they not fluctuated so constantly, exports would have been larger.

"Outside of Joplin the only present great producer is located in Sussex County, N. J. Its zinc ore has been largely exported, being strictly high grade and not directly a competitor of the larger Western mining district. In Wisconsin the sulphide ores are often pyritic; the tonnage mined is not large, and we have no present knowledge of any single great mine in that field; the carbonates are used for the manufacture of zinc oxide, and their tonnage, if not actually smaller than in the past, shows no signs of growth.

"Good progress has been made in perfecting the ingenious devices invented by J. Price Wetherill for the electro-magnetic separation of ores and other substances heretofore deemed incapable of magnetic separation, because of their low degree of magnetic attractability. Generally speaking, the new process is best adapted for ores which gravity separation, either by reason of the minuteness of the component mineral particles or by reason of too slight differences in specific gravity, cannot be made to treat successfully. The Wetherill invention first made possible the separation of the zinc, iron and manganese minerals, constituting what is known as franklinite ore, and added great value to the unique deposits of Sussex County, N. J. It is now found that spathic iron ore can be separated from blende and iron pyrites without any preliminary process other than crushing. The invention will ultimately open a wide field in the separation and utilization of ores heretofore deemed worthless. In Germany and Spain mines abandoned years ago are being reopened. The present scarcity of good furnace material enforces the necessity of rendering available every source of ore supply.

"Probably the most interesting feature to which the zinc industry can point in 1898 is the shifting of the center of manufacture from the older spelter works, scattered through Kansas, Missouri and Illinois, to the Iola gas field, where several large plants are now partly in operation, partly in course of construction, gas being used exclusively as fuel. Naturally enough, it must be an attractive proposition to be able to build furnaces of almost any size. The natural gas can be introduced at any point for the reduction of the ore by merely carrying a system of pipes under and along the front of the furnaces. Coal and ashes no longer cumber the ground. If the gas supply be continuous, the new works will supplant the old, if the latter do not introduce corresponding economies. At present the gas is furnished to all comers free of charge; the supply appears to be ample. When it shall fail, the older locations, by reason of their proximity to the ore and coal fields, will again have the advantage. By that time, however, the plants themselves may be antiquated and useless.

"The sheet zinc industry was satisfactory in price and volume in 1898, while in 1897 it was slow and halting. The sales of high-grade spelter were quickened by ordnance requirements, not only in this country but abroad. The galvanizing industry, the chief consumer of ordinary metal, has been very active, and the many avenues through which zinc enters into consumption for electrical purposes, such as telegraph and storage batteries, have increased their calls, and are likely to continue to do so.

"For oxide of zinc, while prices remained stationary, the demand was large. An exceptional business developed for export by reason of the price advance abroad and the knowledge of the fact, now generally recognized, that the American product is uniformly purer than the foreign. The export business is new; at least it has never before reached so large a volume. With the price at 5½c. per lb. in London, it is not to be wondered at that our product, which is offered much lower, should find a ready sale, and, by reason of its purity, lasting favor. In the United States, also, sales of zinc oxide were large.

"To those who remember the conditions obtaining 15 or 20 years ago, the present status must afford satisfaction. At that time we were largely dependent on foreign importations for zinc in blocks and sheets. All efforts to produce zinc oxide by the French process had failed. Exports of ore were not even attempted. To-day ores and metal are largely exported; more French process zinc white is produced and sold here than was ever imported, and, indeed, it may fairly be stated, as not the least satisfactory outcome of a sound trading situation, that foreign works for many years to come will derive greater profit from our exports of raw materials than they are likely to lose through the competition in finished products."

Colorado.—A small amount of zinc blende concentrates was shipped in 1898 from a mine at Creede. In general Colorado zinc ore carries too much iron to be desirable for zinc smelting.

Iowa.—The zinc mines of Iowa were active in 1898, and a more energetic effort was made to win the reserves of blende known to occur in the region. In recent years the buyers have preferred the oxidized ore and there has been little demand for the sulphide, but the rise in prices sent buyers to Dubuque and led to the mining of some blende. It is believed that better methods of mining and dressing will lead to an increased production from this district, which although small contains, according to H. F. Bain, several promising properties and much undeveloped territory.

Missouri and Kansas.—In 1898 zinc ore reached a higher price than ever before in the Joplin district, and the year was remarkably prosperous. There was a large increase in the output and a good deal of new prospecting work. The establishment of the large smelters in the natural gas field of Kansas, where spelter can be produced more cheaply than at the former centers, enabled the new concerns to make better prices for ore, which the smelters in the coal fields were compelled to meet. Some of them were unable to do this successfully and their works were closed down, the Cherokee-Lanyon Spelter Co., in especial, turning back to the owners early in July several of the works previously leased by it. The high price of ore led to the development of several new camps, while a large number of old mines which had been unprofitable at lower prices were reopened. All the old camps greatly increased their output, Oronogo in particular being the scene of renewed activity and becoming one of the greatest producers in the district. There were large sales of mines and mineral lands, and 76 new mining and dressing works were erected. With the advance in the price of ore the mine operators increased wages to $2 per day for ordinary labor and $2·50@$3 for foremen.

The production of the Joplin district, including the various camps in Kansas and Missouri, was 235,071 short tons, against 177,975 in the previous year. The year opened with zinc ore selling at $23 per ton, at which figure it remained steady during January. In February it dropped to $22, but it commenced to rise during March and advanced steadily up to December 3, when $40·50 was quoted; at the end of the year it had fallen to $29·50. The average monthly prices are given in the subjoined table:

Year.	Jan.	Feb.	March	April.	May.	June.	July.	Aug.	Sept	Oct.	Nov.	Dec.	Average
1896…………	$24·00	$23·50	$23·00	$23·00	$21·50	$21·00	$21·50	$21·00	$20·00	$20·50	$23·50	$25·50	$22·33
1897…………	22·125	21·50	21·00	21·125	21·60	21·875	22·50	22·50	22·625	22·75	23·50	24·25	22·28
1898…………	23·00	22·50	23·00	24·62	26·50	28·50	28·00	28·37	31·00	33·70	36·25	37·00	28·44

The average price of spelter in New York in 1896, 1897 and 1898, respectively, was 3·94c., 4·12c. and 4·57c. per lb. The above quotations for zinc ore at Joplin are for concentrates containing 60% zinc, or 1,200 lb. zinc per short ton; consequently the zinc in this ore was worth 1·86c., 1·86c. and 2·37c., respectively, the difference between these figures and the price of zinc in New York being 2·09, 2·26 and 2·20c., respectively;* the value of zinc in New York is about 0.25c. per lb. more than at the works in Kansas or Missouri. The miners have long been protesting that they have not been realizing a proper

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*It is necessary to bear in mind that these differences have to cover not only the cost of smelting, but also the freight on the ore and loss of metal in smelting.

price for their ore, and numerous attempts to form combinations to force better terms from the smelters have been made. Owing, however, to the conditions under which mining was prosecuted in the district, the more part of the output being made by small operators, concentrating their ore by means of hand jigs, it was impossible to unite a sufficient number of them. Recently there has been a change in the conditions, a large number of the mines having passed into the hands of strong companies, which have erected machine dressing works, with the result that at present only a small part of the output is concentrated by means of hand jigs. The latter, it is said, can be used profitably only with ores assaying 12 to 20% Zn, while some of the new plants are dressing 4% ore at a profit. Under these altered conditions it was considered feasible to form a successful combination, and consequently on December 21, 1898, the Missouri & Kansas Zinc Miners' Association was organized, claiming to control 75% of the production. Early in 1899 this association established a price of $42 per ton for 60% ore, when zinc was quoted at 6c. per lb. at St. Louis, with $1 additional for each unit above 60% and $1 less for each unit below that figure. The smelters at first declined to pay this price, but the association having provided sufficient capital to enable the miners to hold their ore out of the market, and having also received offers for large amounts of ore from abroad, occupied a firm position, and the smelters were finally obliged to buy at the figures named. However, this contest will be continued undoubtedly throughout the present year. The average cost of making 2,000 lb. of spelter at the Cherokee-Lanyon works at Scammon, Kan., in 1897, is said to have been $22·80, while the natural gas smelters at Iola are said to be making spelter now for $15 per 2,000 lb., not including cost of the ore. The merits of the miners' case appear in the comparison between the prices of spelter and prices of zinc in ore cited above. For details of the practice in dressing zinc ores in the Joplin district, see the paper by R. H. Richards, on "Progress in Ore Dressing," further on in this volume.

New Jersey.—The ore body at Franklin Furnace is now known to be over 3,500 ft. long and in places over 200 ft. thick. At the south end of the outcrop the ore is to be worked by an open cut, and the overlying limestone is now being removed. The ore now mined is raised through a shaft 950 ft. deep. The ore body has been opened by numerous drifts and raises, but there are no stopes yet nor has any definite system of mining been laid out, the shape of the ore body in the deep levels being undetermined. The ore is broken by Blake crushers to 0·5 in. size, dried in an Edison tower, crushed to 60 mesh and separated into five sizes, each of which is run separately over a Wetherill magnetic concentrator, which produces concentrates containing manganese and iron, utilized for making zinc oxide and spiegeleisen, and tailings, a mixture of willemite, zincite and calcite, which are jigged in the ordinary manner to remove the calcite, yielding a concentrate which is used for the production of spelter, a good deal of this product being exported to Germany.

Virginia.—Developments were made at the Clark mine, across the New River from Allisonia, Pulaski County, where calamine and blende have been found over a considerable area. The country rock is limestone interstratified with dolomite, in which large beds of mineralized rock are said to occur. A test of a carload showed that 3·12 tons of rock would yield one ton of concentrates assaying 48% zinc and 16% lead. Another test of ten tons of rock gave 2·45 tons of concentrates assaying 40·52% zinc, 16·05% lead, 2·75% iron, 20.54% sulphur, and 1·8 oz. silver per 2,000 lb.

PRODUCTION OF ZINC IN THE WORLD. (IN METRIC TONS.)

Year.	Austria. (*a*)	Belgium. (*b*)	England. (*c*)		France. (*b*)	Germany (*b*)	Russia. (*d*)	Spain. (*b*)	United States. (*e*)	Totals.
			Native Ores.	Foreign Ores.						
1893......	5,870	95,665	9,585	19,244	22,419	142,956	4,522	5,752	69,159	375,172
1894......	6,810	97,041	8,260	24,318	23,387	143,577	5,014	5,100	67,135	380,642
1895......	6,456	107,664	6,760	23,207	24,200	150,286	5,029	5,845	74,245	403,692
1896......	6,888	113,361	7,224	18,054	45,585	153,100	6,264	6,133	70,432	417,041
1897......	6,236	116,067	7,162	16,643	38,067	150,739	5,852	6,244	91,070	438,081
1898......	7,229	....	27,625	....	24,149	....	5,664	8,500	103,514	467,749

(*a*) The statistics for Austria are taken from the official reports of the Mines Department, except for 1898, for which the figure reported by Henry R. Merton & Co. has been used. It is to be noted that Merton's figures for Austria are generally higher than the official figures notwithstanding that the Austrian statisticians include the make of zinc gray with the spelter. Thus Merton reported 9,332 metric tons as the Austrian production in 1897, while the official statisticians gave only 6,236. The discrepancy appears to be due to the Galician works, which give higher figures to Messrs. Merton than to the government, those for the three remaining works in Austria agreeing closely.

(*b*) Official statistics, except for 1898.

(*c*) The statistics for England are arrived at by deducting the zinc produced from domestic ores, as reported in the official Blue Books, from the total output of the smelting works as stated in the reports of Messrs. Henry R. Merton & Co.

(*d*) From official reports, except the figures for 1896, 1897 and 1898, which are reported by Henry R. Merton & Co.

(*e*) Statistics compiled from direct returns by the producers to THE MINERAL INDUSTRY.

(*f*) According to Henry R. Merton & Co., Belgium, Holland and the Rhine district of Germany in 1898 produced 191,836 metric tons of spelter, against 187,406 in 1897; and Upper Silesia produced 99,232 (95,550).

PRODUCTION OF ZINC ORE IN EUROPE. (IN METRIC TONS.)

Year.	Algeria	Austria	Belgium.	Bosnia.	France.	Germany.	Great Britain.	Greece. (*b*)	Italy.	Norway. (*a*)	Russia.	Spain.	Sweden
1892	21,907	33,944	12,260	16	69,236	800,237	27,311	27,695	129,731	576	4,369	74,265	54,981
1893	24,400	30,531	11,310	*Nil.*	74,400	787,911	23,880	22,589	131,767	(*c*)	4,501	62,616	46,623
1894	29,703	28,491	11,585	*Nil.*	80,065	728,616	22,170	20,830	132,777	200	1,440	58,964	47,029
1895	14,300	25,862	12,230	*Nil.*	72,989	706,423	17,758	24,031	121,197	(*c*)	(*d*)	54,109	31,349
1896	17,587	26,887	11,630	*Nil.*	81,346	729,942	19,629	22,700	118,171	750	(*d*)	64,828	44,041
1897	32,269	27,463	10,954	*Nil.*	83,044	663,850	19,587	30,906	122,214	(*d*)	(*d*)	73,848	66,686

(*a*) Zinc-lead ore. (*b*) Including blende and calamine, calcined. (*c*) Not reported in the government statistics. (*d*) Statistics not yet published.

Canada.—The Grand Calumet Mining Co. is exploiting a vein of zinc blende ore near Rossport in the Algoma district, which was discovered in 1898. The ore is said to average about 50% zinc. The same company is operating several mines on Calumet Island, from which it shipped 1,500 short tons of ore, valued at $30,000, principally to Antwerp, during 1898.

Greece.—The increase in the price of zinc in 1898 had a highly beneficent effect on the production of calamine at Laurium, which rose to 30,650 metric tons, or 8,000 more than in 1897; the remainder of the Grecian production came chiefly from Antiparos. There is prospect that the exports of calamine from Laurium will soon attain the former high figures, but the calcined ore at present contains only a little more than 40% Zn, against 60% formerly. (E. Grohmann.)

New South Wales.—The shipments of zinc blende concentrates from Broken Hill amounted to 37,907 long tons in 1898, these being shipped chiefly to England, where they have been reduced by the Smelting Corporation, Ltd., and other new process owners, chiefly by the former. An Anglo-Continental syndicate represented in Australia by the Australian Metal Co., of Melbourne, has acquired the rights to the Wetherill magnetic machines in New South Wales and is erecting a plant at Broken Hill for the treatment of tailings.

Spain.—According to Señor Adriano Contreras, in the annual statistical number of *Revista Minera*, the exportation of zinc ore from Spain amounted to 65,333 metric tons in 1898, against 41,040 in 1897; and of spelter 4,551 metric tons, against 2,170 in the previous year. The estimated production of zinc ore in 1898 was 90,000 metric tons, against 62,000 in the previous year. The zinc works at Arnao, the only one in Spain, reported a production of 8,500 tons of spelter in 1898.

THE SPELTER MARKET IN 1898.

New York.—The course of the spelter market during 1898 was exactly the reverse of that of 1897. Then the attempt made by a combination of some Western smelters to hold the market up by artificial means, such as the holding back of large quantities and the exporting of round lots at 10 to 12% below the domestic price, proved futile. In 1898, notwithstanding a large increase in the production, the market advanced beyond all previous limits. This was due primarily to the tremendous consumption in this country. In all lines the demand for the metal was enormous. The wire industry absorbed more spelter than ever before, and the exports of galvanized wire greatly increased. Galvanizers of iron sheets also did a larger business, though the margins in this line were unsatisfactory. Galvanized pipes were in greater demand both here and abroad. The brass consumption was enormous and that of sheet zinc as large as before.

Exports of spelter were about the same as in 1897, but unlike that year, the sales for export were not made at a sacrifice, but at top prices. It was the European demand, that, at the end of the summer, when consumption here was very large, developed the strength of this market. In spite of the fact that spelter sold at unprecedentedly high figures during this year, complaints were heard that smelters were not able to secure a fair margin, owing to ore prices being driven up to unremunerative figures by strong competition.

In January the market ruled dull, and not until the end of February was there any perceptible increase in the demand. Then, however, and through March and the first part of April, values steadily improved, reaching 4·10c. St. Louis, and 4·30c. New York. About the end of April the demand fell off, buyers having covered their wants; and at the increased rate of production stocks quickly accumulated and the market became weak. However, consumption continued to expand, stocks were soon depleted, and the demand at the end of May and in June was simply enormous. At this juncture the burning of one of the smelteries increased the already acute scarcity of spot and nearby metal, and prices advanced rapidly to 4¾@5c. St. Louis and 5⅛c.

New York. In July, buyers having bought more than their requirements, held off, and a reaction set in, the metal selling down to 4·35c. at St. Louis and 4·50c. at New York. In August some sales were made at 4·25c and 4·40c. respectively, but again stocks in the hands of manufacturers had been used up, and with liberal buying the market turned and continued to advance with few reactions until toward the end of the year. In Europe there was a great scarcity of metal, due to the decreased output in the large Silesian mines, and the market at the end of September was £22, while here it was 4·70c. St. Louis and 4·90c. at New York. In October and November foreigners were forced to purchase round quantities in this country, both for prompt and future shipment, and with the increasing consumption on this side, the metal became exceedingly scarce. In November the advance culminated with New York selling at 5·40c. and St. Louis at 5·22½c. During December the price of ore declined to $30 per ton and producers were anxious to make sales. Buyers had been apathetic, and, becoming frightened at the concessions offered, held off, in consequence of which prices suffered considerably, declining to 4·70 at St. Louis. Toward the end of the month, however, the demand improved somewhat, and the market closed at 4·85c. St. Louis and 5c. New York.

AVERAGE MONTHLY PRICES OF SPELTER IN NEW YORK.

Year.	Jan.	Feb.	Mar.	Apr.	May.	June.	July.	Aug.	Sept.	Oct.	Nov.	Dec.	Year
	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.
1893	4·39	4·39	4·28	4·38	4·41	4·27	4·13	3·89	3·69	3·68	3·65	3·80	4·08
1894	3·56	3·85	3·89	3·62	3·47	3·40	3·43	3·38	3·44	3·45	3·36	3·43	3·72
1895	3·28	3·20	3·23	3·30	3·50	3·65	3·75	4·15	4·30	4·10	3·55	3·49	3·63
1896	3·75	4·03	4·20	4·09	3·98	4·10	3·97	3·76	3·60	3·72	3·99	4·14	3·94
1897	3·91	4·02	4·12	4·13	4·21	4·21	4·32	4·26	4·18	4·17	4·08	3·89	4·12
1898	3·96	4·04	4·25	4·26	4·27	4·77	4·66	4·58	4·67	4·98	5·29	5·10	4·57

London.—The market opened quiet but firm in January, with ordinaries quoted at £18@£18 2s. 6d. and special brands at £18 2s. 6d.@£18 5s., but after consumers had bought freely at these figures, the tone became easier and prices declined about a half-crown or so. February opened quiet at £17 7s. 6d.@£18 for ordinaries, and the tendency was dull thereat. For the first two weeks of the month the Continental makers would not meet the market, and the Americans were decidedly higher in their views; buyers had consequently to meet the higher rates asked, and the prices improved to £18 2s. 6d.@£18 5s.

When March came in it was found that the Continental producers had sold out for some months ahead, and in America there was such a good home demand that better prices could be obtained there. Values in London consequently stiffened, and buyers eagerly paid the advanced prices. The month closed with buyers of ordinaries at £18 8s. 9d. April commenced with the scarcity of sellers more pronounced, and most of the dealers were sold out. Values continued to advance almost daily, and as consumers did not believe in the reported scarcity, they only bought rather sparingly; nevertheless, before the month was out ordinaries were worth £19.

May saw a further advance on the Continent, and large lines were sold at good prices. Consumers in England, being temporarily covered, abstained from making fresh purchases, and the market became dull. This state lasted only for a short period, and the price again rose to £19 5s., which was the figure ruling at the commencement of June. Toward the middle of the month large purchases were again made on the Continent, and that restricted the quantity for Great Britain. America was still practically out of the market, and remained only a passive factor until well into the last quarter of the year. Prices in the United States at this time took a sudden jump, and shippers bought back parcels sold for shipment to England. This was immediately reflected on the English market, and prices commenced to soar upward, and before the month was over there were eager buyers at £20 for ordinaries.

July opened with consumers showing considerable anxiety to cover themselves against sales of manufactured stuff, and they readily paid the advancing prices; £20 7s. 6d. was reached before any break occurred, but then some second-hand holders showed an inclination to make sales for forward delivery, and a retrograde movement was made to £20. August witnessed another steady rise, and it became apparent that it was no use expecting any help from America. Values, therefore, rose to £20 12s. 6d., with stocks on the Continent standing at a very low level.

September found buyers very short of supplies for delivery over the remaining months of 1898, and they came in and bought with rather more pluck, and prices advanced quickly to £22. There was then a halt, but in October, after a few holders of second-hand lots had sold at a sacrifice and weakened the market, the consumptive demand grew keener than ever, and prices rose at almost every sale until at the end of the month the lowest price was £24 for ordinaries, special brands fetching 5s. per ton more. Very little spelter was offered at the commencement of November, and the Continental makers had sold for a long time ahead. Spot stuff was very scarce, and there was good buying until £24 15s. was reached. At this point consumers had satisfied their immediate wants and left the market alone. There were then reports of American offers, and this frightened some London dealers, who sold down to £24. This level attracted buyers, and £24 10s. was again established, but when December came in consumers were very reserved, and it was difficult to make sales. Buyers deeming it better to hold off until after Christmas, prices therefore dwindled gradually to £23 5s.@£23 10s. for ordinaries and £23 10s.@ £23 15s. for specials.

Breslau.—The zinc industry in Upper Silesia was unusually prosperous in 1898, due to the favorable statistical position of the metal and the urgent demand by consumers, especially the electrical manufacturers. The price rose steadily and attained a position analogous only to those in 1873-75 and 1890-91. Prices ranged as follows, per 50 kg. f. o. b. Breslau: January, 17·50, 17·80, 17·60 marks; February, 17·60, 18, 18·15; March, 18·30, 18·40; April, 18·50, 18·75; May, 19, 19·15, June, 19·75, 20·10; July, 20, 20·25; August, 20·50, 20.80; September, 21, 22·50; October, 23, 24·25; November, 24·25, 24·75, 25; December, 24·75, 25, 24. The official averages reported by the Royal Mining Office at Breslau are shown in the following table:

	1893.	1894.	1895.	1896.	1897.	1898.
First quarter	16·00	14·50	12·50	13·50	16·00	16·50
Second quarter	16·00	14.00	13·00	15·50	15·50	17·50
Third quarter	16·00	14·00	13·50	15·50	15·50	19·00
Fourth quarter	15·50	13·00	13·50	11·00	16·50	22·00
Average	....	13·87	13·12	15·12	15·87	18·75

In Upper Silesia there were 24 works in operation during the year.

The sheet zinc business in Germany was also very satisfactory in 1898. The price, which at the beginning of the year was 40·5 marks per 100 kg., rose in accordance with the price of spelter and closed at 54 marks, basis Oberhausen. The export trade was less favorable, on account of the diminution in the demand from Japan, Great Britain and Italy. The sale of this product was managed, as previously, by the Sales Bureau of the Schlesischen Actien-Gesellschaft für Bergbau und Zinkhütten-Betrieb in Berlin. The works in operation were those at Lipine, Oblau, Hohenlohe and Bielahuette. The Kunigunde works were idle. The total production of sheet zinc in Germany in 1898 is estimated at 40,350 metric tons. The associated Austro-Hungarian zinc rolling mills continued to make their sales through the central office at Privoz in Moravia. The companies belonging to this association are Tlach u. Keil, Donnersmarkhuette in Privoz, Oswiencim, Dziedik und Maiken (Bacz).

The zinc white business was less satisfactory in 1898 than other branches of the zinc industry on account of previous contracts made before the rise in spelter. Better results are expected in 1899, since a further increase in the cost of the raw material is not expected. The proportionately low prices of the year's contracts led to a large increase in the demand from the consumers and the works were occupied actively. The competition of American zinc white was felt during the year, this product not being subject to an import duty and being offered more cheaply than the German.

The demand for zinc ore was very active, the price rising in correspondence with that of zinc. Importations were made from France, New South Wales and the United States.

The demand for zinc dust in the latter half of the year was everywhere weak, and producers worked at a loss. This situation was brought about by several causes. The active demand which existed from March to November, 1897, led several works, which had previously discontinued the production of this substance, to resume, with the result that there was soon overproduction. On the other hand the demand from the dyers fell off, clear hyposulphite liquor being substituted more and more for indigo dye. In the third quarter of the year Japan left the market completely, owing to an imposition of a new duty of 10% ad valorem on importations into that country. The demand from the United States also fell off. New uses were found for this material, but their demand much less than offset the decrease in the old channels of consumption. The production of zinc dust in Upper Silesia in 1898 amounted to about 2,900 metric tons, against 1,650 in 1897, 1,750 in 1896, and 2,060 in 1895. Better conditions are looked for in 1899, since some of the works will probably discontinue the manufacture.

Cadmium, which began the year at 1,400 marks per 100 kg. at Breslau, fell steadily in value, and at the end the quotation was only 625@800 marks, according to quality and quantity; consequently this metal has reached its former low level. The important demand which came from the Imperial Pyrotechnical Laboratory and the Royal Saxon Artillery Direction in Dresden from April, 1896, to the end of 1897, caused an increase in price from 600 to 2,100 marks per 100 kg. and a corresponding increase in the production. This demand ceased entirely in 1898 and no other outlet for the excess in production developed, wherefore the metal declined rapidly in price, the only demand coming from the manufacturers of alloys and certain chemical preparations as previous to 1896. The production in 1898 amounted to 15,000 kg. against 15,527 in 1897, 10,666 in 1896, and 6,847 in 1895.

DEVELOPMENTS IN THE METALLURGY OF ZINC AND CADMIUM.

The most important recent development in the metallurgy of zinc has taken place in America, and has been not the result of technical study and experience, but the taking advantage of the natural gas resources of Kansas. However this has led to such a large reduction in the cost of smelting (this reduction being variously estimated at 0·35 to 0·5c. per lb. of metal as compared with the cost in the nearby coal fields) that the United States has been placed in the position where it can export zinc at a profit. Natural gas is also used as fuel at several smelteries in Indiana, but these are comparatively small works which are less advantageously situated as to ore supply. The utilization of natural gas in Kansas and in Indiana has not yet led to any radical change in furnace type and operation from those fired with artificial gas elsewhere in the United States, but it is more economical inasmuch as the labor in handling coal to the gas producers and cinder away from them is saved, and for the present at least the natural gas is supplied free to the smelters.

There have been no recent improvements worthy of mention in detail in the metallurgy of zinc in Belgium and Germany. In Silesia the percentage of blende smelted with the calamine is increasing and this is leading to the introduction of the Rhenish-Belgian furnaces and retorts made by machinery, such an installation having been made recently at the Antonienhütte, where the charge now consists of 70% roasted blende and 30% calamine. At the new Bernhardihütte there are eight regenerative gas Silesian furnaces each with 80 muffles, the largest previously in use in Silesia having only 72. In England, where the practice has been rather backward, improved Belgian furnaces have been built at some of the works, and arrangements have been made to remove the fumes of zinc oxide escaping from the condensers, this being done chiefly to improve the health of the workmen. Certain English works have also begun the fabrication of sulphuric acid from the waste fumes of the blende roasting furnaces. In Germany, where the dispersion of sulphurous gases in an objectionable manner is forbidden by law, a great overproduction of sulphuric acid has taken place, and there has been difficulty in disposing of the product.

Recovery of Sulphurous Acid.—A new suggestion as to the utilization of the roasting furnace gases was made by A. Harpf, of Przibram, at the International Chemical Congress at Vienna in 1898, who proposes to employ them in the preparation of sulphite pulp for the paper industry. Sulphurous gas for this purpose is already recovered at some works in Upper Silesia by the Haenisch & Schroeder process. In a process invented by Porak Bros., of Kienberg, Bohemia, the gas is enriched if necessary by burning sulphur in it, and after cooling is brought through a lead pipe to a water injector which carries the gases down to a tank 20 ft. below. The liquor is raised by a centrifugal pump again to the injector, and so on until the pressure of the gas and air above the water in the tank, which can absorb only a certain proportion of sulphurous anhydride, forces the gas from the tank over into a system of vats filled with milk of lime and provided with stirrers. The lime dissolves as calcium sulphite, and finally a clear acid sulphite solution is drawn off.

Composition of Silesian Spelter.—The average composition of Silesian spelter at the present time (August, 1898) is as follows: Zinc, 98·80 to 98·91%; lead, 1·03 to 1·08%; iron, 0·017 to 0·023%; cadmium, 0·015 to 0·034%; arsenic, 0·002 to 0·008%; antimony, traces.

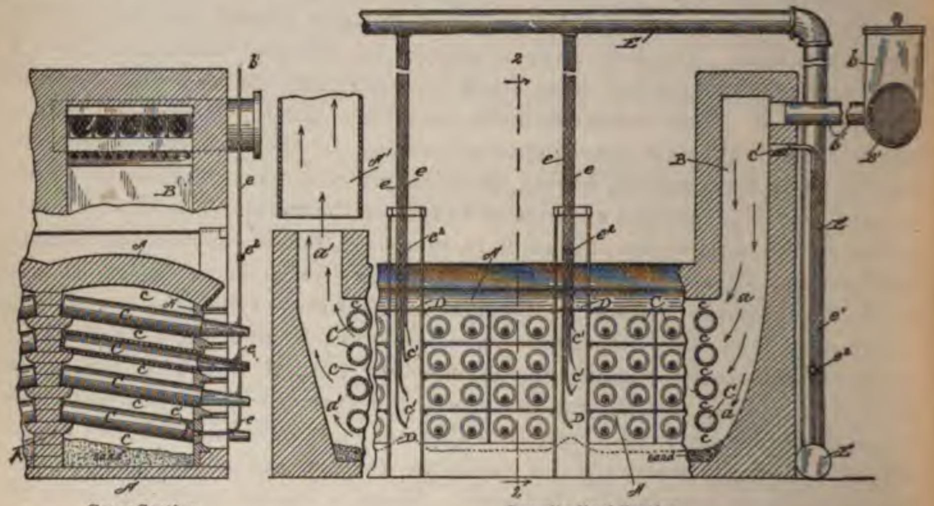
Cost of Producing Zinc in the United States.—The chief factor in the cost of producing zinc is the price of the ore, the average value of which at Joplin in 1896, 1897 and 1898 was respectively 1·86, 1·86 and 2·37c. per lb. of zinc contained, the corresponding averages of spelter in New York (where the price is about 0·25c. per lb. higher than at the works in Kansas and Missouri) being 3·94, 4·12, and 4·57c. per lb. The value of the zinc in the ore is calculated on the assumption that the latter contains 60% zinc, or 1,200 lb. zinc per short ton, this being about the average of the concentrates turned out by the dressing works in the Joplin district. These averages, however, represent the value of the ore reduced to terms of all the zinc contained in it, but the smelters do not save all this metal, and consequently pay not only for what they lose in the process, but also the freight on this loss to their works. The best works in the United States recover probably as much as 87·5% of the zinc contained in the ore purchased; it is doubtful if the older works in Kansas and Missouri recover more than 80%, allowing for all losses in handling, roasting, and distillation. In view of the high cost of the ore in making a ton of zinc, it is obvious that an additional saving in the recovery of metal is of the greatest importance in reducing the cost of production.

In order to make 10 tons of spelter a day in a works in the Kansas gas field, assuming that 80% of the metal purchased in the ore will be recovered, 20·83 tons of ore are required. On the basis of $42 per ton, when zinc sells at 6c. per lb. at St. Louis (the price fixed by the Missouri & Kansas Zinc Miners' Association), the cost of this amount of ore at the mines is $874·86; the freight on the ore and the moisture contained in it raises the cost at the smeltery to $891·66. The cost of labor of all kinds in smelting this ore, at a works of the capacity named, is about $100, or $10 per ton of metal produced. The cost of the clay (about 1·5 tons) and the coke for reduction of the ore (about 7 tons) and miscellaneous supplies, repairs, superintendence and general expenses bring the total cost up to \$1,037·16, or 5·19c. per lb. of metal produced, not allowing anything for depreciation of plant. The total cost at a works of the same capacity in the Kansas coal field would be \$1,123·56, or 5·62c. per lb. The gas smeltery has the advantage of not requiring any coal for steam raising, ore roasting, or in firing the distillation furnaces, and a saving in labor, since no coal or its ashes has to be handled; on the other hand, the coke for mixing with the ore in the retort must be transported to the works, and costs a little more than in the coal fields. The economy in gas smelting is therefore large, but that it is not larger—the gas being supplied free—is due to the fact that the coal used by the zinc smelters is obtained at a low cost, and as compared with the cost of ore and labor in zinc smelting, the cost of fuel and all other supplies is small.

The gas smelters in Indiana are at the disadvantage that freight from the Joplin district to their works is \$2·80 per ton, against only 73·33c. to the works in Kansas; but this is partially offset by their greater proximity to the Eastern markets. The two large smelteries in Illinois, which also have a freight rate of \$2·80 per ton from Joplin, employ furnaces fired with gas from very cheap coal. They make sulphuric acid from the fumes of the roasting furnaces, which effects a very important saving in the cost of the ore, and they roll their own spelter, having practically a monopoly of the sheet zinc business, which gives them an increased profit on the metal. The metallurgical practice at the large works of Illinois is, moreover, better than at the average in Kansas and Missouri, and a higher percentage of zinc is recovered, so that it is doubtful if even free fuel is sufficient to offset these economies.

Improvements in Distillation Furnaces.—Edward C. Hegeler, of La Salle, Ill., patented a furnace for burning gas, artificial or natural, in which the air is admitted at one end and the gas through pipes inserted in the front of the furnace at proper intervals toward the chimney end. This is a reversal of the ordinary conditions where the gas enters the furnace at the end, and the air through ports at intervals in its façade. When a concentrated gas, such as natural gas, is used for heating the latter type of furnace, the regulation of the draft becomes difficult in consequence of the comparatively small volume of the gas, while the deposition of soot on the outside of the retorts obstructs the draft and makes them less permeable to the heat. The improved furnace has at one end of the combustion chamber a flue through which the whole amount of air to be consumed in the furnace is admitted, enough gas being introduced and burned in this flue to raise the temperature to the required height at the first group of retorts. Further on in the combustion chamber, at different points, enough additional gas is admitted to produce the required temperature in all parts. The admission of gas is regulated so as to consume all the oxygen admitted at the end of the furnace, an excess of gas making itself known by the appearance of flames at the exit end of the combustion chamber.

The construction of this furnace is shown clearly in the accompanying drawings. *B* is the air flue, the portion of it which immediately joins the combustion chamber being made of brickwork, and the other portion *B′* being made of sheet iron and provided with a regulating valve b, the sheet iron portion being preferably connected with the brick portion by means of a series of small sheet-iron pipes b', entering through separate openings in the brickwork. The retorts in the combustion chamber are arranged in groups with open spaces at D between them. E is a large gas pipe, conveying the supply of gas under suitable pressure. This pipe is connected with the combustion chamber by means of small pipes e which enter through holes c' in the wall of the retort chamber into the open spaces at D; $e'e'$ are similar small pipes (but somewhat larger than $e e$) which enter the brick portion of the air flue. Each of the small pipes $e e'$ is provided with a valve by which the amount of gas passing through may be regulated. The openings c' through the wall of the air flue and the front of the chamber are made somewhat larger than the outside of the gas pipe, the latter being made tight by luting with clay. The gas pipes are sufficiently flexible to be pulled out of the openings and turned to one side for looking into the furnace. The cross sectional area of the fire brick part of the air flue should not be much less than the cross sectional area of all the open spaces between the retorts. Only the requisite amount of gas is admitted into the combustion chamber through the air flue and the sections DD, etc., to heat the group of retorts immediately in advance, so that free oxygen may be met by the gas entering the chamber at each section. In this manner a uniform temperature is maintained throughout the chamber, and the lowermost retorts are heated as easily as the upper. The air, which may be either hot or cold, is forced through the flue by means of a fan.*



Cross Section. Longitudinal Section.

THE HEGELER GAS-FIRED DISTILLATION FURNACE.

Richard Schneider, of Dresden, Germany, patented a distillation furnace for the reduction of zinc ores containing lead, and the recovery of the latter metal as well as the zinc. This furnace consists essentially of a single combustion chamber in which are placed three rows of retorts, the latter being clay

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* United States Patent No. 612,104, Oct. 11, 1898.

cylinders open at each end, placed in the furnace at a small inclination. A condenser is connected with the higher end in the ordinary manner. A similar condenser is connected with the lower end, but opens into the lower half of the retort instead of into the upper half as in the case of the ordinary condenser. The idea is that the zinc will be volatilized and condensed in the usual manner, while the lead reduced will trickle down the incline and collect in the condenser at the lower end.* There is no novelty in this idea, and there are various structural difficulties in building such a furnace, not to speak of difficulties in its operation. George M. Holstein, of Pulaski, Va., has designed a special retort for the same purpose. This retort is placed in the furnace in the ordinary manner. The clay plate which closes the end under the condenser is provided with a tap-hole, through which any lead accumulated in the retort can be drawn out.†

Robert H. Lanyon and William Lanyon, of Iola, Kan., patented a zinc smelting furnace, of which the front is built up of clay pillars, with clay plates resting horizontally upon them, in such manner as to divide the façade of the furnace into a series of pigeon holes, each of which accommodates two retorts.

THE LANYON DISTILLATION FURNACE.

The plates resting on every alternate pillar abut closely, while a space is left between the meeting ends on the intermediate pillars. This results in the formation of a series of port holes into the furnace, after the front has been closed by the retorts and the surrounding clay in the ordinary manner. Corresponding to each pillar there is a buckstaff, the alternate ones being made hollow, with rectangular openings or thimbles on one side, which fit into the ports in the furnace. At their upper ends the hollow buckstaves are connected with a central air supply pipe, the object of this design being to supply the furnace with air under pressure, through the hollow buckstaves and ports as described above.‡

Herman Kämmerling, of Girard, Kan., patented a modification of the ordinary Belgian zinc smelting furnace, wherein there are two combustion chambers, separated by a central longitudinal wall, as in the usual type of double furnace, of which only one is provided with a grate. The gases of combustion pass from the fire on the grate between the retorts of that side of the furnace, and then through ports arranged at intervals near the top of the

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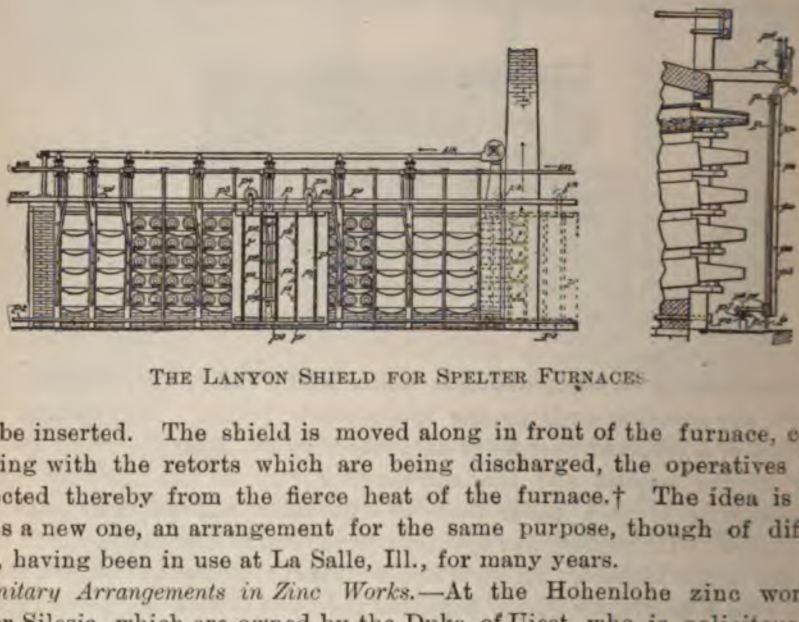
* United States Patent No. 605,802, June 14, 1898.

† United States Patent No. 554,185, Feb. 14, 1896.

‡ United States Patent No. 616,475, Dec. 27, 1898.

dividing wall, and down between the retorts of the other side of the furnace, escaping through openings in the floor of the latter, thence passing through suitable conduits to the chimney.* This type of furnace is not new.

William Lanyon and Josiah Lanyon, of Pittsburg, Kan., patented a shield for use especially in connection with zinc-smelting furnaces, which consists of an iron curtain, or plate, properly stiffened by ribs of angle iron or in any suitable manner, hanging from trolley wheels, running on a rail, parallel and on a level with the spring of the arch of the furnace, and just beyond the vertical plane of the outer ends of the condensers. The bottom of the shield is also provided with wheels running on a rail, to prevent swinging. In the shield there are small oval holes, coinciding with the retorts, through which the steam pipe used in Kansas zinc works for the discharge of residues from the retorts, may be introduced. Between two sections of the shield there is a vertical opening, properly stiffened by angle iron, with transverse lazy bars, through which rakes, for the discharge of the residues, or other working tools, may be inserted. The shield is moved along in front of the furnace, corresponding with the retorts which are being discharged, the operatives being protected thereby from the fierce heat of the furnace.† The idea is by no means a new one, an arrangement for the same purpose, though of different form, having been in use at La Salle, Ill., for many years.



THE LANYON SHIELD FOR SPELTER FURNACES.

Sanitary Arrangements in Zinc Works.—At the Hohenlohe zinc works in Upper Silesia, which are owned by the Duke of Ujest, who is solicitous as to the health of his employees, numerous improvements for their advantage have been introduced lately. Especial attention has been given to the ventilation of the buildings which cover the distillation furnaces, and in these buildings the floor is kept well sprinkled with water to avoid dust. Tanks have also been placed near the furnaces so that the men during their work may cool their faces, hands and feet. Women, who are employed generally in certain departments of the zinc works throughout Upper Silesia, are no longer allowed in the Hohenlohe works.

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*United States Patent No. 610,540, Sept. 13, 1898.

† United States Patent No. 621,577, March 21, 1899.

THE TREATMENT OF MIXED SULPHIDE ORES.

Progress has been made in the treatment of mixed sulphide ores, and at least two companies, one in England and one in Germany, both employing the Fry process, are now prepared to purchase these ores on a large scale, while one electrolytic process is being worked successfully, although in a small way, in England. There has been already a large exportation of zinky tailings from New South Wales, for treatment by these processes, although the output of metallic zinc from this source has not yet been large. Owing to the greater cheapness of executing such metallurgical work in England and the comparatively low cost of carriage from Australia, it appears more economical to export the crude ore from Australia rather than to attempt its reduction there. A highly interesting new development in this problem is the contemplated application of the Wetherill process of magnetic separation. It is claimed that a sufficiently clean separation between the galena and blende can be effected by these machines, and if this hypothesis is borne out in practice, the mixed sulphide ore problem will be a long way toward solution. The galena product will be smelted on the spot in the ordinary manner, while the blende will be sufficiently enriched to be capable of reduction to spelter by roasting and distillation. This process will not of course insure the recovery of the silver contents of the blende, but if the latter be important they may be won by smelting the residues from the retorts.

The Fry Process.—The Smelting Corporation (successor to the Burnham Syndicate) which controls the Fry process of smelting mixed sulphide ores with sodium sulphate flux, closed down the experimental works at Swansea, after treating 20,000 long tons of Australian ore, and commenced the construction of a new plant on the Manchester ship canal. The new plant, of which only one unit (100 to 150 tons per diem capacity) has been begun, was designed by W. B. Devereux, a well-known American metallurgist, and embodies the best features of American lead smelting practice, including a large water-jacket blast furnace. A large part of the machinery for the new works was obtained in the United States. There are several novel features in the design, especially the general use of electrical transmission of power throughout the works, each machine being driven by its own motor. Besides the purchase of mixed lead sulphide ores the new works will carry on a general smelting business, for which it is well favored both by its location and its design. The first unit will probably go into operation about June, 1899. In the treatment of 20,000 tons of ore at Swansea about 500 tons of zinc oxide were made, the more part of the zinc contents of the ore being left in the slags for subsequent treatment at the new works near Manchester. The 500 tons of oxide produced was sold to a spelter manufacturer. It contained about 55% zinc, but it is expected that in regular operation the tenor of this product will be raised to 60%, the impurities being largely lead, soda and sulphur. A good deal of difficulty was experienced at first in properly condensing the fumes, but this is now done successfully by means of water towers. The Anhaltische Blei- und Silberwerke of Silberhuette, Anhalt, is using the Fry process with some modifications which have been made by Dr. Foehr, the director of the company, but have not yet been described publicly. This company reports that the process is successful, but it lacks an adequate supply of ore.

According to the latest patent of H. E. Fry and Robert Addie the zinkiferous slags are mixed with from 10 to 20% of carbonaceous material, preferably non-bituminous coal or coke, in a fine state of division. The richer the slag, the more carbonaceous matter required. The whole is charged into a regenerative gas-fired furnace, preferably of the Siemens type, the furnace hearth or bottom being composed of a material, neutral or basic, capable of withstanding the action of the basic slag, or should the zinkiferous material be acid, an acid or neutral bottom should be put in; for treating basic zinkiferous slags, a basic bottom is preferably used, consisting of calcined ground dolomite mixed with anhydrous tar as a binding material, and the bottom must have a neutral division between the acid brickwork of the furnace and the basic bottom, the material usually employed being chrome ore, containing a high percentage of Cr_2O_3. The gas and air ports of the furnace, valves, regenerator spaces, flues, etc., are of specially large area in order to prevent choking and deal with the increased volume of fumes over and above that from the combustion of the gases from the coal used for heating purposes. The charge is melted down at a fairly high heat, say 2000° to 2500° F., the bulk of the zinc being eliminated during the melting period. After melting the charge is well stirred or rabbled at intervals, to promote further elimination of the zinc, or it may in addition, and preferably so, be stirred with green wood poles for the same purpose. After the zinc has been eliminated as far as desired, as shown by the fumes of ZnO becoming less dense, the molten slag is tapped out and a fresh charge put in. The fumes of zinc oxide may be collected by suitable means, either wet or dry, and the draft of the furnace maintained either by means of a chimney, or by means of a fan or other suitable appliance.[*]

The Ellershausen process was experimented with further by the Sulphides Reduction (New Process), Ltd., which erected larger works at Llanelly, South Wales, and treated ores from Broken Hill, Tasmania, and France. The rights to the process in the last country have been acquired by a French syndicate which proposes to erect works in Angouleme, where large bodies of low-grade mixed sulphides are said to exist. The process has undergone important modifications from the original scheme.[†] Smelting in a cupola blast furnace was first substituted for reverberatory furnace smelting, the zinc-lead fume being collected in the same manner, the zinc being dissolved from this fume by means of sulphurous or sulphuric acid water and precipitated from the solution by means of sodium sulphide, the zinc sulphide being filter pressed and roasted to oxide. The details of the present process have not been described, but it is said that the smelting is performed in blast furnaces with only the ordinary fluxes, and although the zinc is recovered from the fume as sulphide and then oxide, sodium sulphide is no longer used as the precipitant. Silver-lead is tapped from the crucible of the furnace. In treating Broken Hill ore assaying 20% Zn, Ellershausen claims a recovery of 90% of

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[*] *English* Patent No. 4,911 of 1898.

[†] THE MINERAL INDUSTRY, Vol. VI., p. 674.

that metal and avers that the furnace runs without fouling. This process seems to be a development of the Bartlett process in use at Canyon City, Colo.

Electrolytic Zinc Production.—No material progress was made in 1898 in processes for the electrolytic production of zinc, and so far as I am aware the plant of Brunner, Mond & Co., at Winnington near Chester, England, employing the Hoepfner process, was the only one which made electrolytic zinc direct from ores. Its output was about 600 long tons. The Nahnsen process was used at Lipine, in Upper Silesia, for refining the impure spelter produced there. The works at Fuehrfort in Germany employing the Hoepfner process were idle on account of internal difficulties in the company. The works at Duisburg, which formerly made a small amount of electrolytic zinc by the Dieffenbach process were also idle, previous operations having been at a loss, it is said. The Hoepfner people are taking steps to acquire the Duisburg works, which are well equipped and well situated. The Hoepfner process is to be put in operation by the Erste Soda Fabrik, at Hruschau, in Austrian Silesia, in May, 1899, beginning with 400 h. p. although the works are planned for 4,000 h. p. Hoepfner works are also to be built near Brussels, in Belgium, near Turin, in Italy, and at a place in Southern France. According to Dr. Hoepfner, his process as developed at present yields from 4 to 5 kg. of zinc and 12 to 15 kg. of bleaching powder per horse power day. The works at Winnington produce about 5·5 tons of bleaching powder and 2 tons of zinc per day, the latter assaying 99·93 to 99·98% Zn. The zinc is said to be obtained as a perfectly homogeneous deposit.

The Ashcroft electrolytic zinc process, which was tried on a large scale by the Sulphide Corporation, Ltd., at Cockle Creek, New South Wales, was finally abandoned at the beginning of 1898. The causes of the failure were related in a paper by E. A. Ashcroft before the Institution of Mining and Metallurgy, June 22, 1898, which was criticized by me in the *Engineering and Mining Journal* of October 22, 1898. The Ashcroft process, described in the text books and in numerous technical periodicals, consisting in leaching the roasted ore with a solution of ferric chloride, whereby zinc chloride and ferric hydrate are formed, and the subsequent electrolysis of the zinc chloride, with iron anodes, wherein the zinc is deposited and ferrous chloride is formed, the latter being then converted into ferric chloride, the original lixiviant, was never used except in an experimental way, and the practice at Cockle Creek resolved itself into leaching with dilute sulphuric acid and electrolysis of the zinc sulphate solution, or substantially the old Létrange process. Difficulties were encountered in leaching the ore on account of the gelatinization of the silica, which made filtration costly, presses being required for this purpose, while in decomposing the solution much trouble was experienced by the formation of mossy zinc. The chief obstacle, however, to this as to all other similar electrolytic zinc processes was the large amount of power required for precipitation of the zinc. At Cockle Creek 3,000 kw. were required to produce 1,000 kg. of zinc per hour, which means 4,730 h. p. at the fly-wheel of the engine, assuming an efficiency of 85%. At 3 lb. coal per horse power hour there would be a consumption of 6·45 metric tons of coal per 1,000 kg. of spelter, or nearly twice as much as is consumed in distilling an ore with 50% zinc in Belgium. The cost of coal, however, is only a part of the total cost of steam power. In the United States, in the development of 250 h. p. by a compound condensing engine and consumption of 2·3 lb. of coal per horse power hour, with coal at $3 per ton, the fuel came to about 50% of the total cost. With coal at the same figure and a triple expansion 500 h. p. engine running continuously a cost of 0·7c. per h. p. has been noted. On the last basis the 4,730 h. p. hours required to produce 2,204·6 lb. of zinc would cost $33·11 or 1·5c. per lb. of spelter. It does not cost so much to reduce zinc oxide in the ordinary manner in good practice, either in Europe or in the United States, but in electrolysis there is the cost of anodes, cathodes, various supplies and labor, besides the power, and the last is not likely to be more than 33 to 50% of the total cost of production. Under certain favorable conditions, especially the availability of cheap water power, it may be possible to obviate the technical difficulties in the electrolysis of zinc sulphate and successfully carry out the process, but otherwise it does not yet appear that any electrolytic zinc process can be made to compete with the ordinary distillation process, if no by-products are recovered together with the zinc.

F. Foerster and O. Guenter have studied the causes for the deposition of mossy zinc in the electrolysis of zinc chloride solution, with the conclusion that this is due especially to the presence of precipitated zinc oxide on the cathodes.* The oxygen of the air easily oxidizes zinc in presence of zinc chloride, and a certain quantity of the oxide formed is dissolved by the solution, forming basic zinc chloride. Deposition of mossy zinc occurs only when the excess of oxide begins to make the electrolyte turbid, and if this precipitation be prevented by the periodic addition of hydrochloric acid good deposits of zinc can be obtained. According to Foerster and Guenter, the electrolyte should contain a quantity of free hydrochloric acid corresponding to a 0·04 normal solution. The acid may be increased until it represents 0·1 normal acid, but it should not be allowed to fall below 0·004. If this slightly acid electrolyte be used in conjunction with an arrangement for liberating free chlorine, such as a cell with two small platinum anodes and a large zinc plate, serving as cathode for both pairs of anodes, still better deposits of zinc are obtained. When the solution contains 0·2 g. of free chlorine per liter the hydrogen evolved at the cathode does not remain sticking to the electrode long enough to impair the surface of the deposit. A circulation of the electrolyte is necessary to avoid the formation of a layer in contact with the cathode which has been deprived of its free acid, although the main body of the electrolyte may remain slightly acid. The addition of 1% of a perfectly neutral solution of ammonium persulphate to a neutral solution of zinc chloride furnished good deposits of zinc by electrolysis.

Proposed New Processes, Chemical and Electrolytic.—A large number of new processes for the treatment of mixed sulphide ores were proposed in 1898, but none of them passed beyond the experimental stage, and most of them remained simply on paper. Many of these proposals are merely repetitions of former

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* *Zeits. f. Electrochem.*, 1898, V., (1) pp. 16-23.

ideas, and it is astonishing not only that the inventors should have taken the trouble to patent them, but also that patents should have been allowed. These processes fall into several classes: (*a*) Smelting for recovery of the zinc as oxide or metal; (*b*) liviviation and precipitation by chemical reagents; (*c*) lixiviation and precipitation by electrolysis.

A. M. G. Sebillot, of Paris, has patented a special furnace for the reduction of zinc ores in the blast furnace. It is a bottle-shaped water jacket, with an air-tight top, the ore being charged through a special apparatus. The furnace has two rows of tuyères, one a little above the crucible, and the other near the top of the ore column. Midway between the two series of tuyères is an outlet for the gases, which pass first through a chamber containing incandescent charcoal, then through a flue wherein condensation to metal takes place, and finally to a dust chamber for the recovery as oxide of any zinc escaping condensation.* It is claimed that the oxidization of the zinc, the great obstacle to the recovery of zinc as metal by blast furnaces, is overcome in this apparatus, but no account of its practical operation is given, and it is referred to here more for the purpose of putting it on record than from any idea as to its practicability.

G. H. Blenkinsop, of Swansea, proposes to smelt zinc-lead ores in blast furnaces, with or without flux and using a hot blast, taking care that the gases drawn off from the throat of the furnace are always at a temperature above that at which zinc vapor condenses. The volatilized zinc is to be collected in suitable condensing chambers.† There is nothing novel or useful in this idea.

Julius Asbek, Niederfischbach, Germany, described‡ experiments with mixed sulphide ore from Australia. The ore assayed 27·5% lead, 34% zinc, and 810 g. silver per metric ton. The gangue consisted chiefly of quartz, together with some alumina, iron and manganese. The ore crushed to 3 mm. size was subjected to a sulphatizing roasting in a reverberatory furnace. The time of roasting was 12 hours, 2·5 tons being roasted at a charge. The roasted ore was leached with water containing 2·5% sulphuric acid in wooden vats with stirring apparatus. After one hour all the soluble zinc was in solution. The charge was then drawn off to another vat, whence it was put through a filter press. In leaching the ore lost 40% in weight. The residue assayed 45% lead, 8·3% zinc, and 1,225 g. silver. In roasting there was a loss of silver amounting to as much as 10%, which took place at a moderate temperature. A Swedish ore with 18·5% lead, 27·7% zinc and 70 g. silver was roasted sulphatizingly and leached, yielding a residue with 33·5 lead, 5·2% zinc and 150 g. silver. The solution contained 6% zinc and some copper, iron and manganese. The copper was precipitated with zinc dust. Iron and manganese were then oxidized with chloride of lime and precipitated by caustic lime. Zinc was then precipitated by sodium carbonate. The precipitate after drying had a snow-white color and assayed 58% zinc, but after calcination it retained a yellowish tinge and was unsuitable for use as a pigment although it had good covering power. Precipitating the zinc as oxide with lime gave a product with 33% zinc. Precipi-

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* English Patent No. 7,904, 1898. † English Patent No. 8,990, April 8, 1897. ‡ Der Gnom. Nov. 17, 1898.

tating with magnesia 90% pure gave a product with 54% zinc. Dolomite gave one with 35% zinc.

G. de Bechi, of Paris, proposes to treat mixed sulphide ores by roasting with common salt, lixiviation of the zinc chloride formed thereby, and precipitation as hydroxide by means of milk of lime; the gases evolved in the roasting are condensed in suitable towers, the acid water from which, after neutralization of sulphuric acid by means of calcium chloride, is conveyed to the leaching vats.[*] This idea is not novel and there are numerous difficulties in carrying it out. The amount of salt required in the roasting is large, probably as much as 1 lb. per lb. of zinc present in the ore. The zinc is partially converted to chloride, which is easily volatile, and partially to oxychloride, which is insoluble, while a large part of the lead is also converted into chloride and goes into solution with the zinc. In precipitating the zinc from the solution a portion goes down as oxychloride. The recovery of silver in this procedure is also complicated. It has been attempted to volatilize the whole of the zinc as chloride, and condense the fume in water, from which it may be recovered as above described, but the formation of oxychloride, which is not volatile, in the roasting is an obstacle to this process.

H. A. Frasch proposes to treat oxidized zinc ores by lixiviation with ammonia water of 26° B. (containing about 28% NH_3), precipitation of copper by metallic zinc, and then precipitation of the zinc by boiling off the ammonia which is recovered for further use after the manner of the ammonia soda process. The loss in ammonia is claimed to be very small. It is estimated that 1 ton of ore containing 10% ZnO will require 1·5 tons of ammonia water of the above strength. A similar process was patented by Henry Howard, of Brookline, Mass.[†]

L. P. E. Jacob, of Paris, patented a process for the treatment of mixed zinc-lead sulphide ores by subjecting them to a sulphatizing roasting, leaching with water and fractional precipitation of the iron and copper by means of zinc or magnesium oxide, the zinc itself being recovered by magnesium oxide. Magnesium sulphate is obtained from the solution.[‡] This is a repetition of the Marsh & Storer process.

E. A. Ashcroft patented[§] the use of insoluble anodes (carbon, for example) instead of iron, in the electrolysis of zinc sulphate solution. The solution from the leaching vats enters the cathode compartment where metallic zinc is precipitated. The weakened solution then moves on, becoming mixed with ferrous sulphate, and enters the anode compartment, where the ferrous salt is raised to the ferric state. A third of the solution is drawn off to the leaching vats to dissolve more zinc, while the remainder passes into a receptacle containing scrap iron, where reduction to ferrous salt takes place. The solution is then led back to the anode division. Polarization is prevented by the presence of ferrous salt in the anode division. While there is little likelihood that the Ashcroft process will find application again, it is worth while to put the above method on record as a contribution to patent literature. As a metallur-

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[*] German Patent No. 100,242, July 1, 1897; English Patent No. 285, Jan. 5, 1898.

[†] United States Patent No. 623,154, 1899.

[‡] German Patent No. 100,425, Jan. 6, 1898.

[§] English Patent No. 7,532, March 23, 1897.

gical process it is without value. These petty improvements cannot overcome the huge consumption of power which the Ashcroft process involves.

S. O. Cowper-Coles proposes to treat mixed sulphide ores by roasting to sulphate and oxide, which is facilitated by using a mechanical furnace with a hollow rabble through which air may be blown. After roasting the ore is leached with water and dilute sulphuric acid so as to obtain a solution containing approximately 1 oz. of free acid and 20 to 30 oz. zinc sulphate to each gallon of water. This solution, having been freed from copper, is electrolyzed in vats provided with disk cathodes of aluminum and anodes of lead plates. The cathodes are revolved slowly during the electrolysis. The current density employed varies from 15 to 20 ampères per sq. ft. A convenient voltage at the terminals of the electrical generator is said to be 100 to 150 and at the terminals of the cells from 1 to 2. The residue remaining in the leaching vats after washing is further leached with a solution containing 30 oz. of caustic soda per gallon of water, which is passed through the ore a number of times, to dissolve the lead as sodium plumbate. The solution of this salt is electrolyzed for deposition of the lead or it may be used for the manufacture of lead white. Repeated use of this solution causes it to become partly converted into sodium silicate and carbonate, consequently an amount of freshly burned lime is added to reproduce caustic soda, but care must be taken not to add too much lime.[*]

This process has been experimented with on a rather large scale at Hayle, Cornwall, by the British Broken Hill Co., using ore from Broken Hill, containing 30% Pb, 24% Zn, 4% Fe, 19% S, and 0·06% Ag. The residue after leaching is said to have been brought down to 4·5% Zn. In these experiments the zinc has been deposited intentionally as moss, which is removed from the cathodes by wooden scrapers. It is proposed to use this as a substitute for zinc dust. An assay of the product showed 54·88% Zn, 12·55% O, combined with the zinc as oxide, 10·05% SO_3, 1·05% SiO_2, 0·80% CuO, 0·07% PbO, 1·20% Al_2O_3, 0·60% Fe_2O_3, 0·45% CaO, and 13·35% water and organic matter—total, 100%. To insure the zinc being deposited as moss a low current density is employed in connection with a weak solution. The lead obtained by leaching with caustic soda the residue after zinc extraction is also deposited on revolving disks and the deposit scraped off from time to time. The lead thus precipitated can be melted down by a simple operation. An analysis showed 0·08% Cu, 0·0004% Sb and Sn, 0·0117% Fe, 0·015% Zn, and by difference 99·8029% Pb. Silver is recovered from the residue after lead extraction by leaching with potassium cyanide and precipitation on silver sheets. Litharge is obtained by evaporating the sodium plumbate solution at a temperature not exceeding 80° C., any sodium silicate that may be in solution having been precipitated previously as calcium silicate by means of calcium oxide. White lead is precipitated from the plumbate solution by means of CO_2, obtained by decomposing $CaCO_3$ by SO_2, from the roasting furnaces, condensed in water. The CO_2 which is mixed with SO_2 is passed into a vessel where lead sulphite is precipitated by the latter, the purified CO_2 then being conveyed to another vessel where lead carbonate is precipitated.[†] This process is ingenious, but it

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[*] English Patent No. 5,943, of 1898.

[†] *Electrical Review*, London, Nov. 25, 1898.

appears complicated. The difficulty in obtaining a good deposit of zinc is evaded, but it is worth while to remember that the world's requirement of zinc dust is comparatively small and is easily supplied by Upper Silesia. Cowper-Coles gives no data as to the cost of his process or its efficiency.

John Jones, Melbourne, Australia, proposes to treat mixed sulphide ores by obtaining a sulphate solution of the zinc, which is subsequently purified from iron by precipitation in a tower with gases containing ammonia. From 10 to 25% of sulphuric acid is then added to the purified solution of zinc sulphate, which is led to a second absorbing tower, where it meets gas containing ammonia, developed by an ordinary producer, which is absorbed by the sulphuric acid, a solution containing both zinc sulphate and ammonium sulphate being drawn off from the tower. This solution is conducted to electrolytic vats, each cell of which is divided into two compartments by means of a porous diaphragm, one compartment containing a lead anode and the other a zinc cathode. The mixed solution flows into the anode compartment, while either acid zinc sulphate or acid zinc sulphate and ammonium sulphate flows into the cathode compartment, the added ammonium sulphate being to produce ammonium ions in the catholyte and also to harden the zinc coat as it is deposited. On the passage of the current zinc is deposited on the cathode and ammonium is liberated, the latter combining with the free acid of the catholyte, whereby free acid is prevented from accumulating near the cathodes. A certain amount of free acid must be maintained, however, in the cathode compartments. The sulphuric acid which is developed in the anode compartment may be used for the lixiviation of further quantities of ore.*

F. M. Lyte, London, proposes† to treat mixed sulphide ores by roasting and obtaining a sulphate solution of the zinc, which is converted into chloride by treating with an alkaline chloride. The zinc chloride is concentrated, dehydrated, and electrolyzed in molten condition with an anode of carbon and cathode of fused metallic zinc, both the Zn and Cl of the $ZnCl_2$ being recovered. In the electrolysis of this salt when melted there is danger of boiling over and explosion, as mentioned by Lorenz and others. Merely heating does not serve to drive off all the hygroscopic water, which can only be accomplished by a long smelting with solid or, better, molten zinc. The process can be hastened by electrolysis with a current in reverse direction, by which the molten zinc is the anode and the carbon rod is the cathode. The current decomposes the water whose oxygen goes to the zinc anode and there forms zinc oxide while the hydrogen is developed at the cathode. With sufficiently high tension kernels of metallic zinc are separated there also and assist in the decomposition of the water. Instead of the carbon electrodes, copper or another metal whose melting point is higher than zinc may be used. The zinc oxide formed by the reversed electrolysis swims in the molten zinc chloride and in part is dissolved therein. After complete dehydration the charge may be subjected to electrolysis, with the carbon as anode and the molten zinc as cathode, in accordance with the inventor's English patent No.

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* English Patent No. 4,296, of 1898.

† English Patent No. 11,190, 1897; United States Patent No. 618,575, 1899, and numerous others.

15,813, of 1895, without danger of boiling over or explosion. In this process the oxide is first decomposed and after it the chloride, which appears through the development of chlorine at first mixed with oxygen. The electrolysis of molten zinc chloride has been an attractive subject of experiment, owing to the possibility of employing currents of greater density, and because of the higher conductivity of the fused salt the less electromotive force required as compared with the electrolysis of the aqueous solution, while the formation of mossy zinc is not to be dreaded, and the process can be made continuous. On the other hand, molten zinc chloride is an exceedingly corrosive substance, and so far there has been no apparatus devised on a practical scale which will withstand it.

Bernhard Mohr's improved process* consists in treating the roasted ores with a solution of sodium bisulphate,† thus obtaining a solution of zinc sulphate saturated with neutral sodium sulphate. By electrolysis of this solution metallic zinc is deposited at the cathode and sodium bisulphate is regenerated at the anode. It is claimed that this method furnishes a solution of zinc sulphate which is entirely free from iron, while the solution of zinc thus obtained, being saturated with neutral sodium sulphate, is easily electrolyzed and the formation of mossy zinc is entirely avoided, owing to the fact that the SO_4 liberated at the anode does not form free sulphuric acid, but is directly converted into sodium bisulphate, while the regenerative character of the process makes the cost of chemicals low. This process has been taken up by some large zinc works in Belgium.

J. Swinburne, London, has introduced a process‡ wherein the crude ore is stirred into a bath of molten zinc chloride (anhydrous) heated from beneath. A little molten lead is then added to take the gold and silver from the ore, and thus enriched is tapped off, after which an electric current is passed through the bath, first depositing lead, which is drawn off as soon as this part of the reaction is completed; the electrolysis is then continued to decompose the zinc chloride, the deposited zinc being also drawn off as metal. After all the zinc sulphide has been decomposed the gangue is raked out and heated to liquate out any zinc chloride remaining in it, the chloride being conveyed back to the bath. The reactions which take place in this process are complex. No chlorine, or only a little, is given off, but sulphur vapor appears at the anode, whence it may be recovered. Probably chlorine is set free at the anode and immediately decomposes the metallic sulphides, lead first and then zinc, with the formation of chlorides and liberation of sulphur, the temperature being too high to permit the formation of sulphur chloride. The same criticisms apply to this process as to the Lyte process, previously described. The Swinburne process, which has not yet been worked outside of the laboratory, is being exploited by the Phœnix Syndicate, Ltd.

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* United States Patent No. 588,076, 1897.

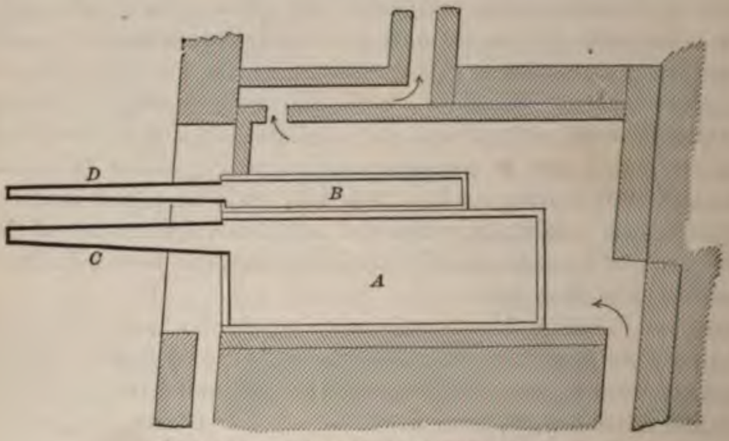
† In the original process, described in THE MINERAL INDUSTRY, Vol. VI., the ore was fused with solid sodium bisulphate.

‡ English Patents No. 10.829 and No. 10.829A, both of May 1, 1897.

THE METALLURGY OF CADMIUM.

Herr Otto Saiger, director of the works of Georg von Giesche's Erben, in Upper Silesia, which furnish about two-thirds of the cadmium produced in that province, equivalent to saying two-thirds of the world's production, has furnished the following notes as to the present practice there:

"Cadmium being more volatile than zinc passes over first in the distillation of zinc ores which contain it, and escaping condensation, enters the large pipes leading away the fumes from the zinc furnaces employed in Upper Silesia, where it is deposited as oxide, this deposit being richer in the parts more remote from the furnaces. This condensed fume, which at the Wilhelmine works averages 3 to 4% cadmium, and sometimes rises to 5%, having been collected by itself, is mixed with fine coke and charged into the retort *a*, shown in the accompanying engraving, of a special furnace. This retort, or muffle, which is 1·8 meters in length, takes about 115 kg. of the cadmium-bearing oxide, mixed with about 17 kg. of coke. It is charged about 8 o'clock in the morning and the distillation is finished about 6 o'clock the next morning. The temperature in the muffle is maintained at a medium red glow, whereby only the cadmium oxide is reduced. The metal collects in the sheet-iron condenser *c*, partly in the form of powder. That which is molten is so pure that it requires only a remelting before casting into the small sticks (Stangen) which are known in the trade. The powder is collected by itself, and every three days is reduced in the smaller retort *b*, the temperature of which is maintained somewhat higher than that of *a*. The metal obtained from the condenser *b* is over 99·5% pure, and is likewise cast into sticks. The residual zinc oxide from the retort *a* is returned to the zinc distillation furnaces."



RETORTS FOR CADMIUM DISTILLATION.

PROGRESS IN ORE DRESSING IN 1898.

BY ROBERT H. RICHARDS.

THE literature of the year includes a number of interesting machines and devices for improving concentration and mill work. It also contains a number of useful discussions upon milling practice, among which may be mentioned the effect of temperature on gold amalgam; the relations of sorting, concentration, and cyaniding on the Rand; the importance of laboratory tests to check mill tests; and the equipment of concentrating mills for small mines.

BREAKERS AND FINE CRUSHERS.

The "Invincible" Crusher.[*]—The Ottawa Gold Milling and Mining Co.'s mill at Keewatin, Ontario, is equipped with some novel machinery. The crusher is an "Invincible." In this the jaw has a nearly parallel motion. The usual arrangement is reversed, the outside jaw being movable, with the result that the common massive frame is done away with, which brings down the weight of the 10×18-in. machine from 12,000 to 18,000 lb., as in the Blake or Dodge, to 9,000 lb. The jaw moves over an elliptical path, the long axis of which is so inclined that, during the moment of crushing, the swinging jaw plate moves directly toward the stationary plate, and at the end of the stroke moves down and back to let the material pass through. The tension is taken by four wrought-iron rods, which run through to the back of the frame. The inclination of the moving jaw to the fixed jaw can be altered to suit the ore.

Sturtevant Roll-jaw Crusher.[†]—The union of a coarse and a fine crusher in one machine has not, as a rule, succeeded, owing to the great difference in capacity of the two parts; either the coarse crusher would be waiting for the fine to catch up, or, if fed up to its capacity, the fine machine would be so completely choked as to stop, or to back up into the coarse crusher. The machine here referred to has a larger capacity for fine than for coarse work, so that choking is improbable if not impossible. The movable jaw is so linked to the frame that the lower portion, perhaps three-quarters of the length, has a rolling motion upon the fixed jaw, while the upper portion simply approaches and recedes from the fixed jaw after the manner of the Dodge crusher. This

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[*] Frederick T. Snyder, *Canadian Mining Review*, February, 1898, p. 48; Federated Canadian Mining Institute, 1898.

[†] *Engineering and Mining Journal*, Jan. 29, 1898, p. 184; made by the Sturtevant Mill Co., Boston.

upper portion is the coarse breaker, while the lower acts like a pair of rolls of great surface, but forced against the rock by toggle links.

*Sectional Cushioned Rolls.**—The purpose of these rolls is to prevent the finer particles of ore from passing through uncrushed when a large lump forces the rolls apart. This is accomplished by making one of the rolls in several sections or rings about 2 in. wide, only one of these rings being forced back when the hard lump arrives. For each ring there is a stiff rubber cushion, which makes a snug fit both on the shaft and in the bore of the hub. The cushion is from ¾ to 1 in. thick, according to the size of the machine. The sections of the roll have skeleton cores, and are driven by two steel arms in the form of lathe dogs, one on each side. These arms are keyed to the shaft, and each passes through and presses against the spokes of half the sections.

Iron Stamp Guides.†—Cast-iron stamp guides are a common feature of New Zealand mills. They are very deep (about 18 in.), with lubricant recess at the top. For dry crushing they are simply bored out, and for wet crushing are frequently run in with white metal. They do not wear the stem appreciably, and are generally preferred to other kinds.

Tappet Keys with Steel Ends.‡—Iron tappet keys being soft, take good hold, but in the frequent resetting of the tappets the ends of such keys become burred, and time is lost in driving them out and reshaping them. Steel keys, if soft enough to take hold well, do not contain enough carbon to permit hardening the ends. A good remedy consists in using iron keys with ends of drill steel. No particular advantage results from putting steel on the large end, but the advantage gained by the steel toe soon pays for the extra cost.

Improved Blanton Cam.—In Fraser & Chalmers' circular No. 26 is shown a new form of Blanton cam, which is attached to the cam shaft without keys or wedges. The shaft has ten wedge-shaped flutes or grooves, and the hub of the cams are bored with corresponding flutes. This construction allows the ten cams, all of which are alike, of two adjoining batteries, to take their positions as required.

The Barbour cam fastener is in the form of a curved wedge, on the under side of which is a spline to fit the longitudinal grooves in the cam shaft. It is used on the principle of the earlier form of Blanton cam fastener. The positions of the splines on the five wedges of a set differ 18° from one another. This, in connection with the two grooves, 90° apart in the shaft, gives all the combinations necessary for the five positions of the cams. This fastener is made by the Risdon Iron Works, San Francisco.

Order of Drop for Gravity Stamps.§—When the order of drop 1, 5, 2, 4, 3 is used, No. 3 violently splashes the pulp against the screen, instead of spreading part of it under No. 4. This concentrated splash causes excessive wear on the screen opposite No. 3, enlarging the holes there before the rest of the screen shows wear. It is usually necessary to drop the end stamps ½ in. or 1 in. more than the others, to prevent banking at the ends; but this causes the end

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* Joseph W. Pinder, American Institute of Mining Engineers, February, 1898.

† A. H. Bromly, *Engineering and Mining Journal*, Nov. 12, 1898, p. 575.

‡ F. F. Sharpless, *Engineering and Mining Journal*, Aug. 20, 1898.

§ *Mining and Scientific Press*, Oct. 8, 1898, p. 352.

stamp to strike the cams sooner than the others, and so compels a slower speed than could otherwise be run. A case is reported in which the end stamps were made heavier than the others, but had the same drop. This worked very well, especially when a fast drop was required, and would seem to be a better method of overcoming the banking at the ends.

Freedom of Discharge from Stamp Batteries.[*]—The use of a chuck block in a stamp battery prevents the pulverized ore passing in a direct line from the dies to the screen; and this, by hindering the discharge, reduces the capacity and increases the slimes. More than 20 years ago Sharwood had mortars made for the Keltz mine, in which the inside plates dropped into a recess sloping 60°. This gave a straight path for the pulp from the die to the bottom of the screen.

The D. B. Morrison quick-running stamp mill,[†] which increases the number of blows per minute by the use of a piston and water cushion, has been introduced at the Dorothy Morton gold mine, British Columbia. The weight is 1,000 lb. per stamp, and it makes 135 7-in. drops per minute; stamping 6 to 7 tons per 24 hours. It is expected to make 150 drops.

The Wood steam stamp has a single stamp to a mortar. It is a vertical slide-valve engine with the piston rod serving as stamp stem. A tappet on the stem operates a bell-crank lever which controls the valve, which is a simple piston valve. The stamp itself weighs 550 lb.; the shoe is 8 in. in diameter; 60 to 70 lb. steam pressure will operate the mill, though compressed air may also be used; 200 blows a minute can easily be obtained. The frame carrying the steam cylinder is separate from the mortar. The upper portion of this frame is adjustable on the uprights by shims, as a means of taking up the wear of the shoe and die. The battery discharges on three sides, and is arranged for inside amalgamation.

At the Mammoth mine, Yavapai County, Ariz., 112 tons of hard quartz ore were stamped through a 40-mesh screen in 342 hours—equal to 7·86 tons in 24 hours. On gold ores of ordinary toughness the capacity is claimed to be 8 to 15 tons in 24 hours. The machine is built by E. P. Allis & Co. for the owners, H. A. Newkirk & Co., Chicago.

Comparison of pulverizers in regard to their tendency to make slimes:[‡] Following are some results of sizing the products of different pulverizers, set to grind everything fine enough to pass through a sieve of a certain mesh; the ores were all of the same general character (the pyritic "banket" of South Africa):

Machine.	Meshes Per Square Inch.	Meshes Per Linear Inch.	Per Cent. Passing One Mesh and Remaining on the Next Finer.			
			30×30	40×40	60×60	90×90
Wet stamps	900	30·0	5·60	12·66	17·58	64·16
Wet stamps	700	26·5	11·15	28·53	9·21	51·11
Dry stamps	400	20·0	20·30	9·80	21·80	49·10
Rolls	400	20·0	26·63	33·09	13·06	26·30
Rolls	500	22·4	9·30	41·85	15·38	33·47
Ball mill	500	22·4	20·07	24·38	13·88	41·67
Niagara pulverizer	....	....	20·17	24·30	24·30	31·23

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[*] W. S. Sharwood, *Mining and Scientific Press*, Feb. 26, 1898.

[†] Engineering, 1898, p. 681.

[‡] Franklin White, "Proceedings Chemical and Metallurgical Society of South Africa," Jan. 15, 1898.

These figures illustrate the well-known fact that rolls produce less fines than stamps.

At the Waitekauri Gold Mining Co.'s mill, Coromandel Peninsula, New Zealand, the pulp from a Krupp ball mill with 60-mesh screen contained less slimes than the pulp from stamps crushing through a 40-mesh screen.[*] Both machines were crushing ore that had been dried.

SCREENS.

The Columbian Separator.—This is a dry screen set at an angle of about 40° and shaken by an oscillatory movement perpendicular to the plane of the screen. The claims made for it are that it has large capacity, does not blind up, has long life, and a coarser size can be used for the same result than with a horizontal or rotating screen; *e.g.*, a 50-mesh screen will give 100% through 80-mesh, the amount of the diminution depending upon the slope angle; it is greater with larger angles. The machine is made by the Jeffrey Manufacturing Co., Columbus, Ohio.

FINE CONCENTRATORS.

A Gyrating Vanner.[†]—A new form of gyrating vanner has been installed in the Ottawa Gold Milling and Mining Co.'s mill. It has the usual 4×12-ft. belt. The shaking frame is supported on six toggles (three on each side) resting on the floor. Beneath the frame is a short vertical shaft carrying an unbalanced wheel, the revolution of which causes the machine to gyrate. The diameter of gyration is about ¾ in.; the table makes 225 gyrations per minute. The belt travel is obtained by a friction pulley similar to that on a Hendy feeder. The tail piece of this clutch is attached to the floor, and the rate of travel depends on the position at which the attachment is made. The gyration of the frame operates this friction pulley. Results of tests on an iron pyrites ore from stamps showed 48% of pyrite saved in the concentrates with one Frue vanner per 5-stamp battery and 54% with two, and 61% with one gyrating vanner to a 5-stamp battery. The coarse concentrates went to one side of the belt and the fine to the other side.

Automatic Blanket Table.[‡]—Thomas White has designed a shaking blanket table which can be cleaned up with but little labor. A light frame suspended by four rods carries two tables side by side, each about 9 ft. long and 22 in. wide. The slope can be regulated by means of turnbuckles on the supporting rods. The tables are covered with blankets, and there are three drops (height not stated) in the length of the table, which receives 200 throws of ⅜ in. each per minute, longitudinally. When the blankets are charged with concentrates and gold the feed is switched off by a trough to a spare table and the tables are tilted sidewise and washed off into a trough by a spray pipe. Such a double table weighs only 600 lb., and is claimed to treat 4 tons in 24 hours.

Lord & Banfield's dry concentrator[§] for gold gravel consists of three screens,

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[*] A. H. Bromly, *Engineering and Mining Journal*, Nov. 12, 1898, p. 576.

[†] F. T. Snyder, *Canadian Mining Review*, February, 1898, p. 43; May, 1898, p. 134.

[‡] Australasian Institute of Mining Engineers; *Canadian Mining Review*, 1898, p. 287.

[§] *Oesterreichische* Zeitschrift, 1898, p. 161.

the coarsest uppermost and the finest lowest, the last having wooden crosspieces, making it a riffle box. The whole apparatus is given a rocking motion something like that of a cradle. Beneath the lowest screen is a pressure box, into which a blast of air from a bellows is forced. The air passing up through the screen sorts and concentrates the gold in the fine sand. Two men work the machine.

Wood's dry concentrator is similar to the above except that the air is furnished in intermittent pulsations, and cloth is used under the riffles instead of the finest screen plate. It requires three men. The amount treated is 1 to 4 tons in eight hours. In a test the Lord & Banfield machine extracted 63% of the gold and Wood's 53%.

AUXILIARY MACHINES.

A Spokeless Tailing Wheel.[*]—This consists of a short hollow cylinder of great diameter in which the buckets for lifting sand and water are placed. Its axis is horizontal. It is like the spoked sand wheel except that the spokes and shaft are left out; instead it runs on friction rollers to which the driving power for rotation is applied and it is prevented from falling over by guide rollers halfway up the sides and by flanges on the friction rollers, upon which the T-rail runners, on the outside of the cylinder, roll.

Frenier & Le Blanc Spiral Sand Pump.—This consists of a spiral ribbon of steel plate, in form like a dismounted clock-spring, on each side of which is a disk of steel plate, which is joined to the spiral by a continuous air-tight joint. There results from this a spiral tube of steel which has a rectangular cross section of equal area throughout. The outside diameter is 4 ft. It is mounted on edge with horizontal shaft, makes 20 revolutions per minute, and receives water and sand as the open end descends into the well. The sand and water work their way inward by the spiral path till they reach the hollow shaft which connects with a standpipe carrying the sand and water up 20 ft. At the Congress mine, Arizona, 90 tons of wet stamp pulp is raised 15 ft. every 24 hours by one of these pumps. The makers are Frenier & Le Blanc, Rutland, Vt.

A new form of sampler for the material fed to a gravity stamp mill is described by F. T. Snyder.[†] It is a cast-iron hollow cone with a 90° angle, revolved with its axis horizontal and with the ore which has been through coarse rolls fed inside perpendicular to the lower element of the cone. The cone has a slot, which once in every revolution comes round and lets the ore through to the rear side; at all other times it is sheared off to the front.

MECHANICAL CONCENTRATION OTHER THAN GOLD.

Plants for Small Mines.—F. L. Bartlett says[‡] that while the more extensive mills with systematic treatment by screens, jigs and vanners are suitable for larger and more permanent mines, the small mines must have simple and low-cost machinery with the best efficiency and capacity that can be obtained. The Wilfley type of table seems to offer the solution of the difficulty. The

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[*] *South African Mining Journal*, 1898, p. 1021. [†] *Canadian Mining Review*, February, 1898, p. 43.
[‡] *Engineering and Mining Journal*, May 7, 1898, p. 549.

following mill is an example of what can be done in this line with a capacity of 20 tons in 11 hours, yielding galena, chalcopyrite with some blende, and quartz with the remainder of the blende: One No. 2 Gates breaker, one set 12×30 in. slow-speed rolls, a revolving screen 3×6 ft., a 24-ft. elevator of No. 85 link-belt chain, two shaking tables, a 50 h. p. boiler, a 35 h. p. engine, a feed pump, an injector, and a tank pump, with necessary shafting, pulleys and belts, cost $3,325 f. o. b. cars. Freight and carting to mill came to $975. The building cost $1,000, and is large enough to double the capacity. Setting up the machinery and necessary adjuncts cost $1,000, making a total of about $6,300.

*Concentrating Iron Ore.**—A concentrating plant, which has been in operation for a few months at the Pewabic mine in the Menominee range, was expected to treat the ore wholly by dry methods, but there has been so much clay that jigging by water had to be resorted to. Thus far the yield by concentration has been only about 19%; but it is expected that, as the openings become large enough to permit a better selection of ground, the yield will be about 33%.

At the Grangesberg iron mines, Sweden, the small ore was formerly thrown away, but now it is treated by dumping into a large wooden trough, the bottom of which is a strong grating. The large pieces that will not pass the grating are picked for magnetite and hematite. The undersize goes to a revolving screen, run wet, which sends different products to magnetic separators to save the magnetite. The waste from these is picked for hematite.

Zinc and Lead Ore Dressing in the Joplin District, Missouri.—In the first attempts at concentrating the lead and zinc ores of the Joplin district the ore was dumped over a grizzly to get rid of the coarse rock, and the undersize was washed in sluices, where much fine ore was lost. The process was slow and wasteful. Hand jigs (movable sieves) were introduced to save the fine ore, and are still used by small operators.

The tank being nearly full of water and ore shoveled into the jig box, the operator, by skillful jigging for five to eight minutes, causes a separation into layers, which are removed separately. The fine ore which passes through the screen is periodically shoveled out and treated by itself. To treat 30 tons in nine hours five such jigs are required if the ore is zinc ore only, or six if it is zinc and lead, divided as follows: Four for rough jigging, one to separate the lead and zinc, and one to treat the fines.

In mechanically operated plants, when no crushing is required, the Robinson jig is used. For this there is a tank 12 ft. long, 6 ft. wide, and 6 ft. high. In this tank is a box 10 ft. long, having longitudinal grate bars, with cross bars 4 in. high to hold back the ore. This box is pivoted at one end, while the other is raised and lowered by a crank operated from the line shaft. A bed of 500 lb. of coarse ore is put in, and the ore is then fed at the lever end, the feed being the undersize of a grizzly with 2-in. spaces, the oversize of which goes to the dump. The jig runs at 80 revolutions per minute. The fine concentrates pass through the sieve, the coarse concentrates are discharged

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* Editorial in *American Manufacturer*, Sept. 9, 1898, p. 375.

through side openings with gates and mix with the fines, while the tailings go off at the end into an elevator.

For hard ores, that require crushing, both the German and the Missouri systems are used. A 100-ton mill of the German type has a 15-in. Blake crusher, followed by a set of 24-in. rolls, delivering by elevator to four trommels, which are followed by an hydraulic separator. The oversize of the first trommel (¾-in. holes) is returned to the rolls. The oversizes of the ⅝-in., ½-in., and ¼-in. trommels and the spigot of the classifier go to separate jigs, and the overflow of the classifier to a convex slime table. The jigs have four compartments. The last two sieves of Nos. 1, 2 and 3 make middlings, which are reground by auxiliary rolls that deliver by the elevator to the trommels. There are only three such mills in operation in Joplin. In the improved Missouri system the ore is fed through a 15-in. crusher to a ⅝-in. trommel, the undersize of which goes to a 5-sieve roughing jig, while the oversize goes to a 12-in. crusher and thence to a ⅜-in. trommel. The undersize of the latter goes to a 7-compartment finishing jig, and the oversize to a pair of 24-in. rolls, which deliver to the roughing jig. The object of the second crusher is to prevent sliming the soft ores that would happen if everything larger than ⅝ in. went directly to the rolls.

The roughing jig runs at 130 r. p. m., the first four sieves of the finishing jig at 200, and the last three at 300. The first two sieves of the roughing jig make clean galena, the next two make clean blende, and the last makes middlings, which go to special rolls and are returned to the jig. All the hutchwork goes to the cleaning jig. The first sieve of the latter yields clean galena, and the second, third and fourth yield clean blende; the fifth sometimes yields clean blende; the sixth and seventh yield middlings, which go to the special rolls and are returned to the roughing jig. These jigs are patented by George T. Cooly, of Joplin.

Diamond Washing.[*]—The "blue ground" in which the Kimberley diamonds occur is called "kimberlite." Its matrix, though difficult to determine, is generally conceded to be serpentine (sp. gr. 2·6), and the chief minerals imbedded in it, in addition to the diamond (sp. gr. 3·5), are garnet (sp. gr. 3·1–4·3), olivine (sp. gr. 3·3), mica (sp. gr. 2·8), ilmenite (menaccanite, sp. gr. 4·9), calcite (sp. gr. 2·6), zircon (sp. gr. 4·4), magnetite (sp. gr. 5·0), and occasionally pyrite (sp. gr. 5·0). Sometimes also fragments of black slate are found. On exposure to the air the rock disintegrates, especially if wetted, and on this property is largely based the method of treatment.

When the rock comes to the surface it is spread about 10 in. deep on immense "floors," which consist merely of the bare ground stripped of grass, bush, and loose stones. The first treatment consists in running harrows back and forth between two traction engines placed several hundred yards apart. After a proper interval the hard lumps that will not disintegrate are collected, portable tracks are laid, and these lumps carried to the crushing house. A second harrowing then takes place, and if no rains have occurred in the meanwhile the rock is wetted by hose, a complete system of pipes being laid over

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[*] *Thomas H. Leggett, Cassier's Magazine*, September, 1898, p. 383.

the floors for this purpose. After another period of disintegration the rock is loaded into cars on movable tracks and sent to the washing machines; any remaining pieces of hard "blue" being sent separately to the crushing house. The weathering takes about a year for the De Beers rock, but for the Kimberley only half as much time is necessary. On July 1, 1897, there were 2,061,355 tons of rock on the "floors" of the De Beers Co. (from the De Beers, Kimberley, and Wesselton mines).

At the Kamfersdam mill, which is treating weathered rock ("yellow ground"), the material is dumped over grizzlies with 3-in. spaces. The oversize goes to Comet crushers, and then all the rock goes with water to double sets of rolls. These are 48 in. in diameter, 30 in. corrugated face, running 45 r. p. m. The upper pair are set 1¾ in. apart, the lower ⅞ in. The product is elevated to trommels with ¾-in. holes, the undersize of which goes to the upper row of washing pans, while the oversize goes to the fine rolls together with the tailings from these pans. These rolls are 72 in. in diameter, 30 in. face, and set $\frac{3}{16}$ in. apart. They deliver by bucket elevator to a second row of pans. These washing pans are annular, 14 ft. outer diameter, 4 ft. inner, the outer rim being 12 in. high, the inner rim 8 in. The upper pans have flat bottoms, but the bottoms of the safety pans slope slightly toward the outer rim. They are fitted with stirring arms driven 8 or 9 r. p. m. by a central shaft, the stirrers being set so as to work the material toward the outer rim. The tails flow over the inner rim; the concentrates are periodically discharged through a sliding door extending across the bottom and go to the "pulsator house." One pan treats 300 to 400 loads (240 to 320 tons) in 10 hours. Some of the plants use only one set of pans.

The hard rock which will not disintegrate is treated in a separate mill. In the De Beers plant for this purpose the rock is broken to 2 in. in Comet crushers and goes thence to trommels. The oversize goes to circular sorting tables to guard against large diamonds being crushed, though no diamonds of any consequence have been picked out on these tables. Following the tables is a series of rolls, trommels and jigs, the prevailing jig sizes being ⅝, ½, ⅜, ¼, and ⅛ in. In each case the hutch product goes through rolls to finer jigs. This mill treats about 900 tons of rock a day, and sends 8 or 9 tons to the "pulsator house."

In the "pulsator house" the concentrates from washing pans and hard-rock jigs are sized in trommels having holes ⅜, ¼, $\frac{3}{16}$, ⅛, and $\frac{1}{16}$ in. diameter. Everything larger than ⅜ in. is hand picked; everything smaller than $\frac{1}{16}$ in. is thrown away. The intermediate sizes are jigged through beds of lead bullets. The tails of these jigs are waste and the hutches go to the picking house. About 250 lb. go to the picking house from 100 tons of original ore. The diamonds are washed in sulphuric and nitric acid and then sorted according to color. Recently two employees of the company, Kirsten and Labram, have experimented with a light side-shaking table, covered with grease of a particular composition, to which the diamonds adhere, while all other minerals are washed off. In one test, with material from the final jigs ⅛ to $\frac{1}{16}$ in. size, out of 6,601 carats of diamonds contained in the lot treated, only 111 (1·7%) were lost. In another test, on material from $\frac{3}{8}$ to $\frac{3}{16}$ in. size, out of 19,031 carats of contained diamonds only 40·25 carats (0·2%) were lost.

Some of the smaller companies, to save locking up a large amount of capital on the "floors," send all the rock to be crushed at once, but this method has not found favor with the De Beers experts. It is essential that the pulp in the pans should have a certain consistency, about one-third of its bulk being fine mud. The weathered rock furnishes this, but with direct crushing a large amount of waste has to be returned to the pans.

*Weathering vs. Direct Crushing of Diamond Ore.**—The Robinson Diamond Mining Co. erected a plant to treat its ore by direct crushing and washing instead of by weathering and washing, but found that by this treatment some of the diamonds were broken in the rolls and there was a large loss in the tailings. Consequently it decided to first weather the "blue ground;" the machinery for the direct process, which cost $418,820, was adaptable to the weathering process. E. H. Jones, in discussing the directors' report, said that the New Bulfontein Diamond Co. had had a similar experience, its tailings carrying off 30% or more of the diamonds, but by first weathering the blue, 30% more diamonds were obtained than by direct treatment. At the Wesselton mine also fully 30% of the diamonds were lost by the direct process.

MAGNETIC CONCENTRATION.

Theory and Results of the Wetherill Magnetic Separation Process.—Prof. William A. Anthony gives† outline descriptions of the Edison, Buchanan, Conkling, and Wetherill magnetic separators in order to show the difference in principle between the last and all preceding designs, which he states concisely as follows: "Instead of small magnet cores and large air-gap area, there should be large magnet cores and small air-gap area—large magnet cores to reduce the resistance and permit of the production of a large number of magnetic lines, and a small air-gap area through which all these lines must pass, and in which, therefore, they are crowded together, which is the same thing as saying that the magnetic force is intensified."

The power consumed by the Wetherill machines at Franklin Furnace is small, being only about 100 watts to each magnet.

C. Blömeke‡ states that the Wetherill machine of the magnetic type used at Friederichssegen, near Ems, Germany, does twice the work and occupies much less space than the Wenstrom machine. It does away with the troublesome roasting, and easily separates minerals of high gravity which cannot be separated by water, *e.g.*, monazite, rutile, etc. It also treats the fine dust.

Stefan Farbaky reports the results obtained with Wetherill machines in comparison with wet jigging.§ The experiments were made on ore containing hematite disseminated in quartz. The lot used for magnetic treatment contained 28% Fe. It was reduced by breaker and rolls and screened into the sizes shown in Table No. I. The tailings from the first treatment on the mag-

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* Directors' report, and discussion at meeting, Aug. 1, 1898, *South African Mining Journal*, Aug. 6, 1898, p. 921.

† *Cassier's Magazine*, March, 1898, p. 433.

‡ *Oesterreichische* Zeitschrift, 1898, p. 147.

§ *Oesterreichische Zeitschrift*, March 26, 1898, p. 182.

netic machines were retreated. The lot used for jigging contained 25·96% Fe. The results are shown in Table No. II. The scheme of jig work includes the retreatment of the middlings, but Table No. II. does not include the results of retreatment.

TABLE I.

Size of Grains. mm.	Per Cent. of Each Size.	Per Cent. of Concentrates.	Iron in Concentrates.	Iron in Tailings.	Capacity of Machine. Kg. Per Hour.
	%	%	%	%	
3·5-2·5	20	54·9	35·03	18·3	1,410
2·5-1·5	21	37·8	39·6	21·8	1,393
1·5-0·75	30	44·6	41·4	17·9	1,117
0·75-0	29	37·0	42·6	23·6	747
	100	43·5	39·2		1,125

TABLE II.

Size of Grains. mm.	Concentrates.		Middlings.		Tailings.	
	Ratio to Original Ore.	Iron.	Ratio to Original Ore.	Iron.	Ratio to Original Ore.	Iron.
	%	%	%	%	%	%
11-8	3·55	36·16	11·00	....	0·70	11·08
8-5·6	6·70	36·85	19·05	....	2·80	10·12
5·6-4	6·20	35·12	6·35	....	1·75	11·14
4-2·8	4·90	37·43	13·10	....	2·90	10·46
2·8-2	4·95	35·23	6·75	....	1·30	8·19
2-1·4	6·05	36·16	5·15	....	3·20	9·78
Sand.	10·20	38·94	2·25	23·88	11·10	12·73
Meal.	2·70	36·97	0·90	29·33	5·40	13·76
Slime.	....	....	....	....	22·45	22·53
	45·25	36·83	3·15	25·44	51·60	16·46

These results show that the Wetherill machine saved 43·5% of the crude ore, the product containing 60·6% of all the iron and averaging 39·2% iron; while jigging saved 45·25% of the crude ore, the product containing 64·2% of all the iron and averaging 36·8% iron.

Magnetic Separation at Monteponi, Italy.[*]—The ore, containing calamine, dolomite and limonite, averaging 22% zinc, is roasted, which brings it up to 30% zinc, when it is treated by the magnetic machine, which yields one-third iron ore carrying 10% zinc and two-thirds zinc ore carrying 40% zinc. This is roasted and treated on magnetic machines, and zinc ore is then jigged out, during which operation the burnt dolomite slacks and crumbles, yielding concentrates containing 48% zinc. The result of this is 50% of rich zinc ore which contains 84% of all the zinc of the original ore. This is good concentration. Jigging without roasting was tried and gave poor results. Roasting followed by jigging was tried without the magnet, but even that gave a poorer concentrate and a larger loss of zinc than the above plan.

GENERAL MILLING CONSIDERATIONS.

Adaptation of the Process to the Ore.[†]—The more important of the earlier unsuccessful mills designed to concentrate the ores of the Broken Hill district,

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[*] *Oesterreichische Zeitschrift*, 1898, p. 347; see also THE MINERAL INDUSTRY, Vol. VI.

[†] T. J. Greenway, *Engineering and Mining Journal*, Aug. 27, 1898, p. 249.

New South Wales, were designed by an American who gained much of his experience in the Lake Superior district, the general arrangements being largely copied from that district. The plant adopted consisted of steam stamps, hydraulic classifiers, Collom jigs, and circular convex slime tables. Heberle mills were used for recrushing, but these proved costly to operate, and the results were very poor. In the newer and more successful mills the crushing plant consists of Blake breakers and heavy Cornish rolls. The ore is generally crushed to pass a 0·1 in. round hole and goes to the jigging plant without classification.

COAL WASHING.

Oberegger's jig[*] has a thick piston pulsating back and forth horizontally between two hutches. It therefore gives pulsion to one hutch and suction to the other by a single movement of the plunger; the return movement gives the reverse action in the two. The pyrite and slate pass out at the level of the sieve, under a gate, and are raised above the level of the water by a screw conveyor. The good coal overflows with the water at the tailboard. The earlier form used two sieves to a plunger for the coarse coal and four for the fine. The later uses four sieves for coarse and fine. The oscillation is horizontal and longitudinal in a compartment lying between the two sieves on the one side and the two sieves on the other. The forward motion of the plunger gives pulsion to the two head sieves, for example, and the return gives pulsion to the two tail sieves. The vertical movement of the excentric is transformed by a bell crank to give horizontal movement to the plunger. This plan places the driving mechanism directly over the jig and gives it as large a jigging area for the space occupied as any design. The capacity, water and power are given in the following table:

	50-30 mm. Grains.	30-14 mm. Grains.	14-5 mm Grains.		50-30 mm. Grains.	30-14 mm. Grains.	14-5 mm. Grains.
Capacity—lb. per hour.........	165-176	198-220	121-128	Gallons of water per minute..	127-131	127-131	95-105
Double strokes per minute....	60	72	160-170	Power, including that to raise the water..................	3·5 h. p.	3·7 h. p.	2·2 h. p.

The Reden Coal Jig.[†]—At the Reden mine a coal jig, which was formerly run with a discharge of slate beneath a gate which was regulated by hand, requiring care, and at times causing loss of coal in the slate or contamination of coal by slate, has been changed over. The gate is now connected to the plunger by a lever, which causes it to rise and fall with the plunger. A noticeable lessening of the ash in the coal has occurred as a result.

The Elliot trough washer has been installed at the Hultschiner mine in the Ratibor district, Prussia, and washes 200 centners of coal per hour.[†] The coal fed has 13 to 20% ash and is brought down to 7%.

A new breaker at Cranberry coal mine, Hazleton, Pa., is described by W. S. Ayres. It was designed to yield 2,000 tons of merchantable coal in 10 hours,

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[*] *Oesterreichische Zeitschrift*, 1898, p. 638; *Colliery Guardian*, 1898, p. 798. [†] Preuss. Zeitschrift, 1898, p. 184.

and 3,000 tons capacity is now assured. The arrangement and design of all the machinery were planned with special reference to avoiding the use of elevators; to the production of as little of the small sizes and as much of the large sizes of coal as possible; to the introduction of easy grades and turns in all chutes, so as to avoid chipping and waste; and to reduce the amount of labor needed. The crushing rolls and the jigs are of novel design.[*]

The rolls were increased both in diameter and length over the ordinary. The form of the section of a tooth is that of the area included between the segments of two overlapping circles; the tooth therefore has rounded sides and a cutting edge. The tooth also tapers to a point which hooks forward in the direction of its motion. The large diameter rolls produce less of the smaller sizes of coal than those of small diameter, because the teeth act so as to approximately split a lump of coal. With small diameter the opening between the rolls widens upward so rapidly that the coal rides on the teeth and is abraded by them before being carried through.

The jig, so called, consists of a box in which is an inclined plate conveyor extending from near the bottom of the box upward to the discharge lip. Each plate of the conveyor is in the form of a horizontal step. The frame of the conveyor is hinged at its upper end and receives an oscillating motion near the lower end. The unwashed coal is fed on this conveyor at a point 12 to 16 in. below where it emerges from the water. The clean coal is floated away from the steps by each oscillation, while the slate, owing to its greater specific gravity and flat shape, remains on the steps and is carried up and delivered at the discharge lip. The coal falls off the lower end of the conveyor into a bucket elevator.

Practice at the Ruby Coal Breaker, Gunnison County, Colo.[†]—Twelve miles west of Crested Butte the Colorado Fuel and Iron Co. prepares an anthracite coal in the following manner: The coal is dumped from the mine cars on the grizzly.

(1) A grizzly sloping 22° 40′. The first 12 ft. is of diamond bars with ¾-in. spaces; the last 16 ft. is of round bars with 2-in. spaces. Over 2 in. goes to picking table, 2; through 2 in. to No. 2 mud screen, 6; through ¾ in. to No. 1 mud screen, 5.

(2) Picking table: Good coal to rolls, 3, or to lump coal bin; slate, to 12.

(3) Breaking rolls: Coal to main screen, 4.

(4) Main (revolving) screen; over 2¾ in. square holes to fine rolls, 10, or to shipping bin, 13; through 2¾ in. on 2 in., ditto; through 2-in. square holes to fine screens, 11.

(5) From 1. No. 1 mud screen: Over 1½ in. square holes to automatic slate pickers, 8; through 1½ in. on ¾ in., ditto; through ¾ in. on ½ in., ditto; through ½ in., waste (because very dirty), to 12.

(6) From 1. No. 2 mud screen: Over 2¾ in. square holes to picking chute, 7; through 2¾ in. on 2 in., ditto; through 2 in. on 1½ in., to automatic slate pickers, 8; through 1½ in. on ¾ in., ditto; through ¾ in. on ½ in., ditto; through ½ in., waste, to 12.

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[*] American Institute of Mining Engineers, February Meeting, 1898.

[†] R. M. Hosea, *Mines and Minerals*, August, 1898.

(7) Two picking chutes for No. 1 and No. 2 coal, respectively; good coal to fine rolls, 10, or to shipping bin, 13; slate, waste, to 12.

(8) From 5 and 6. Three automatic slate pickers for No. 3, No. 5, and No. 6 coal; good coal to picking chutes, 9; slate, waste, to 12.

(9) Three picking chutes for No. 3, No. 5, and No. 6 coal; good coal to shipping bins, 13; slate, waste, to 12.

(10) From 4 and 7. Fine rolls. Coal to screens, 11.

(11) From 4 and 10. Two revolving screens: Over 1½ in. to shipping bin, 13; through 1½ in. on ¾ in., ditto; through ¾ in. on ½ in., ditto; through ½ in. on ¼ in., ditto; through ¼ in., waste, to 12.

(12) From 2, 7, 8, 9 and 11. Culm and slate bin. Waste by car to dump.

(13) From 4, 7, 9 and 11. Shipping bins. The different sizes are: No. 1, over 2¾ in. square hole; No. 2, through 2¾ in. on 2 in.; No. 3, 2 in. on 1½ in.; No. 5, 1½ in. on ¾ in.; No. 6, ¾ in. on ½ in.; buckwheat, ½ in. on ¼ in.

The automatic slate pickers are of the pattern in which the slate is retarded by passing over a stone slab, and falls through a slot, while the coal passes over the slot. The form consisting of longitudinal V troughs, one arm of which has a tapering slot, through which the flat slate will pass, while the cubical coal passes over, is not successful at this plant, because the coal breaks into somewhat flat pieces. The hand picking is done in Coxe picking chutes, as described in *Trans. Am. Inst. Min. Eng.*, Vol. XIV., p. 418. In these each portion of coal is examined by only one picker.

The slope of the conveying chutes (which are made of No. 10 steel and are half-round with 18 in. diameter) is 3½ in. per foot for No. 1, No. 2, No. 3, and No. 5 coal, and 4 in. per foot for No. 6. These slopes are somewhat less than are used for corresponding sizes in Pennsylvania breakers, but still are found too steep for this coal. It is likely that the slopes will be made as little as 2½ to 3 in. per foot.

*Mechanical Preparation of Coal at Saint Eloy Collieries.**—This consists of screening the coal by a grating and by Coxe screens, hand picking the slate from the coarser sizes upon rope bands made of old winding ropes, and washing the finer sizes upon piston washers. The grating has bars 150 mm. (6 in.) apart. The first Coxe screen has round holes 70, 60 or 50 mm. in diameter. The oversizes of this and the grating screens are picked together upon a band. The second Coxe screen has 30 mm. ($1\frac{1}{16}$ in.) round holes. The oversize is picked on bands or washed in piston washers. The third Coxe screen has 18 mm. or 14 mm. holes, and is treated like the last, but separately from it. The undersize of the last screen is either picked on bands, washed in piston washers, or sold without treatment. The Coxe screens are 1·5 meters long by 1 meter wide, and are shaken 155 times per minute. Each frame carrying the above three screens handles 25 tons of coal per hour. The varying sizes of screens are to suit the varying demands of the market. By shifting the streams of coal and by running or stopping the bands, the coal may be delivered picked or unpicked and washed or unwashed, and may be mixed in the bins in any manner desired.

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* *Colliery Guardian*, 1898, p. 465.

Coal Cleaning at Treharris, South Wales.[*]—In a coal-cleaning plant at this mine coal is dumped over a grizzly with 1½-in. spaces. The oversize is conveyed by an oscillating trough to picking belts, which deliver their cleaned coal by anti-breakage loading jibs to cars. The undersize of the grizzly falls into "billy boxes," in which it is automatically weighed, and then delivered by a scraper conveyor and a bucket elevator to a double trommel. All three sizes go to shipping bins, the two smaller being picked on the way. All the machinery is arranged with special care to prevent breakage of the coal.

The mechanical preparation of coal at the Ferdinand mine,[†] at Kattowitz, Upper Silesia, consists of screening the coal and picking it upon bands. The coarsest size is broken and either sold or sent to the washers. All the other sizes are picked on bands and then go to the loading bins. The first screen is a Karop grating. The second and third sizes are made in a Schwidtal shaking frame. The three finest screens are in a Karlik suspended gyrating frame.

Coal Washing in Alabama.[‡]—Prof. William B. Phillips gives a tabular statement showing the amount of coal washed for coke making in Alabama. In 1891 only 0·4% of the State's product was washed, but this figure has continuously increased since then, so that in 1895 49·6% of the production was washed. In 1893 and 1894 a small proportion of the coal larger than slack was washed, but at present only the slack (through 1¾-in. hole) is washed. The best results obtained by the Robinson washer (which treats unsized coal) gave a reduction of ash from 11·9 to 6·7% on the average, showing the removal of 43·5% of the ash. Phillips quotes some results of work with the Stein coal jigs (which are fed with sized coal), in which the ash was reduced from 17·69 to 6·72% on the average, showing the removal of an average of 60·93%.

Effect of Crushing on the Removal of Sulphur from Coal by Washing.[§]—A sample of ordinary Pittsburg slack coal, and a sample of the same after being crushed to pass through a $\frac{3}{16}$-in. hole, were washed separately in a trough washer with the following results: Unwashed slack, 10·51% ash, 2·876% S; washed slack, 7·97% ash, 2·23% S; washed after grinding to $\frac{3}{16}$ in., 4·56% ash, 1·188% S. The last would make coke with about 1% S (the furnace limit). Another sample of Pittsburg coal, containing 11·95% ash and 2·121% S, was crushed to about 20-mesh size and washed as before, the result showing 4·86% ash and 1·046% S. Coke made from this sample, in a crucible, had 0·836% S. In connection with this work the predominance of sulphur in the finer sizes was noticed. A sample of finely crushed coal showed: Larger than 20-mesh, 1·11% S; finer than 20-mesh, 1·49% S. In a sample of coal dust subjected to suction from a centrifugal blower the coarse contained 1·14% S and the fine contained 1·36%. This fact is important where the water used in washing is drained off and re-used, for some of the float pyrite will adhere to the coal, and it will also accumulate and finally go off with the coal. In the above washing experiments pyrite could readily be hand-skimmed from the surface of the escaping water.

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[*] *Colliery Guardian*, Oct. 7, 1898, p. 648.

[†] *Pruess. Zeitschrift*, 1899, p. 133.

[‡] Proceedings of the Alabama Industrial and Scientific Society, Vol. VII., Part II., 1897, p. 47.

[§] American Institute of Mining Engineers, October Meeting, 1898.

THE PROGRESS OF METALLOGRAPHY IN 1898.

BY ALBERT SAUVEUR.

MUCH has been done during the past year to encourage the students and friends of metallographic methods. The various investigators engaged in this field have shown great activity, and several excellent papers have been contributed. Among these contributions those of Messrs. Stead and Charpy stand pre-eminent.* Those distinguished metallurgists have added abundantly to our knowledge of the structure and properties of steel on the one hand, and of metallic alloys, especially of bearing-alloys, on the other. We are permanently the richer for their work, and our gain is a substantial one of much industrial value.

Considered collectively the results of the metallographic researches of 1898 constitute a link permanently added to our chain of acquired facts.

It must also be a source of gratification to witness the steadily growing interest taken in the subject by the industrial world as well as by educational institutions, for the microscope has found its way during the past year into many an industrial laboratory and metallurgical department of technical schools.

In the following pages the writer has endeavored to outline, necessarily very concisely, the most important recent contributions of metallography to our knowledge of metals and alloys.

IRON AND STEEL.

Cubic Structure of Iron.—Mr. Stead argues with great force that, as Mr. Osmond had already contended, the polygonal granules of ferrite found in iron and low-carbon steel should not be called crystals, but grains; for while their mass is truly crystalline, they themselves are not crystals. The lines of demarcation between these polygonal bodies are junction lines, not lines of cleavage. Fig. 1 illustrates this granular structure of wrought iron and of low-carbon steel. The junction lines between the grains of ferrite are plainly seen. Mr. Stead has beautifully developed the crystalline structure of the individual

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* "The Crystalline Structure of Iron and Steel," John E. Stead, *The Metallographist*, October, 1898; "Study of White Alloys called *Antifriction*," *G. Charpy*, *The Metallographist*, January, 1899.

grains of ferrite, and his investigations have led to results which constitute a triumph for metallographic methods, and testify of the skill and intelligence with which his researches were conducted. He has shown us conclusively that these grains of ferrite or pure iron are made up of cubic crystals of iron, which are always oriented in the same direction in the same grains, but differently in different grains. This explains why on slightly etching a polished section of iron or of low-carbon steel it is found to be made up of polygonal areas, some of which appear white and brilliant, while others are dark (see Fig. 1). It is due to the different orientations of the crystals of which the grains are composed; if their phase is such, *i.e.*, if the crystals of a certain grain dip at such an angle that the reflected light enters the objective, the grain will appear brilliant, while, on the contrary, if they reflect the light outside the objective that grain will appear dark. By causing the specimens to revolve some of the bright grains will become dark, while others that were dark become brilliant as soon as their crystals make the proper angle with the incident light. Mr. Osmond compared this phenomenon to the effect produced by the flashing, revolving light of a lighthouse.

Mr. Stead finds that phosphorus, aluminum, and especially silicon (in large quantity), when associated with carbonless iron give rise to a very marked development of the cubic crystals composing the grains of ferrite, and he has illustrated his observations with some remarkable photomicrographs, one of which is reproduced in Fig. 2. I believe, with Mr. Stead, that this is the most perfect crystalline structure ever developed by metallographic methods. In order to develop the crystalline structure of the grains it is necessary to etch the polished surface deeply, sometimes for several hours, in an acid bath. A slight etching, such as is universally used, brings out only the joints between the grains, leaving their internal structure unrevealed (compare Figs. 1 and 2).

Coarse Crystallization and Brittleness in Soft Steel.—Mr. Stead has shown that in practically carbonless iron and in very low-carbon steel, a prolonged heating at a temperature between 600° and 750° C. (dull red heat) develops "granular masses of exceeding coarseness," which upon being heated to 900° C. (full red heat) or thereabout assume again the normal fine-grained structure. Fig. 3 shows one of these large granules completely surrounded by grains of smaller diameter. "This example," Mr. Stead says, "is most instructive, as it indicated that reorganizations and crystallizations start from isolated centers."

It is found, moreover, that no such growth takes place, under similar conditions, in steel containing over 0·20% of carbon, so that there is a radical difference, in this respect, between the behavior of carbonless or nearly carbonless iron and that of more highly carburetted compounds, when subjected to thermal treatment.

This extraordinary growth of the grains of ferrite under the influence of a prolonged heating at a dull red heat is explained by Mr. Stead as follows: ". . . the axes at many different angles of several contiguous grains alter their position until they become of the same angle in all, with a coincident disappearance of the granular junctions, the result being a larger grain containing the same mass as the sum of all the masses of the smaller grains added

FIG. 1.—SOFT STEEL, 0.11% CARBON, ANNEALED.
Magnified 130 diameters. Etched 1 minute in 0.1 nitric acid.

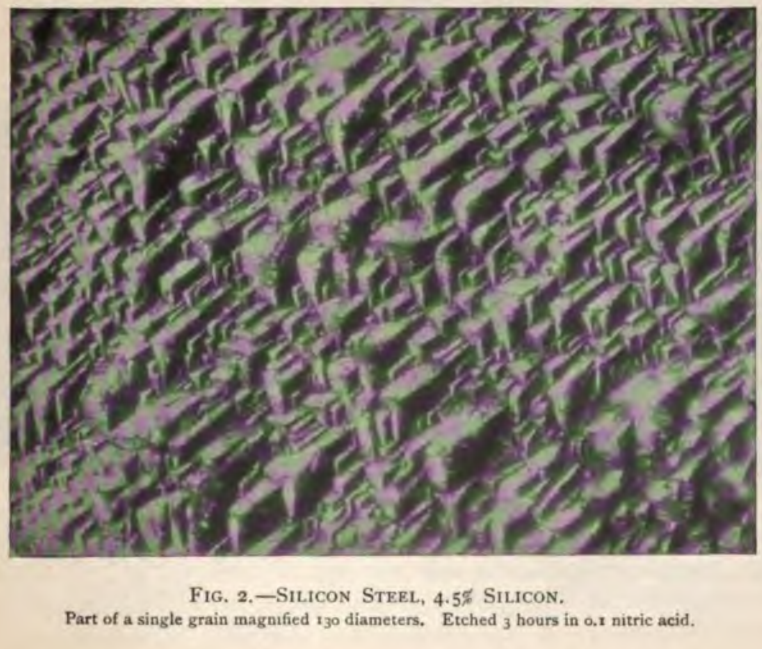


FIG. 2.—SILICON STEEL, 4.5% SILICON.
Part of a single grain magnified 130 diameters. Etched 3 hours in 0.1 nitric acid.

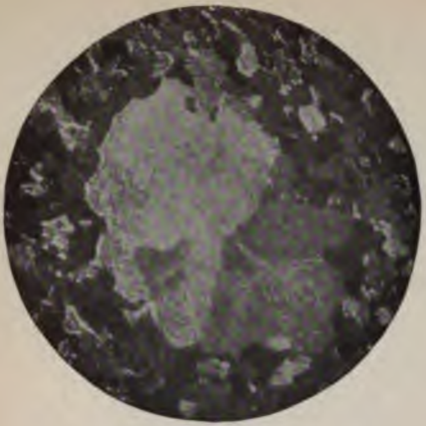


FIG. 3.—DECARBONIZED BAR.

Heated for 3 hours to 700° or more, showing the commencement of coarse granulations.
Magnified 50 diameters.

FIG. 4.—BRITTLE PLATE BROKEN AT RI
ANGLES, THE ROUGH CLEAVAGES
OF CRYSTALS.

Natural size.

FIG. 5.—BLISTER STEEL.

The line of fracture follows the membranes of cementite surrounding the grains.

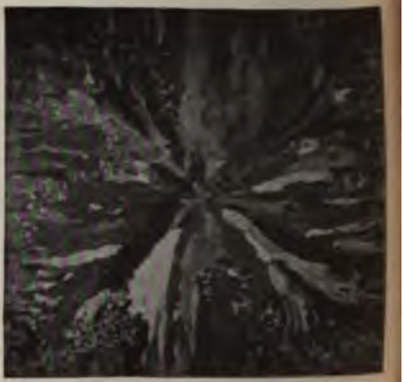


FIG. 6.—STEEL ANNEALED IN LIME AT AB
700° C., ALL THE CARBON (0.215) REMOVED
AND COLUMNAR STRUCTURE PRODUCED.

Magnified 3½ diameters.

together. Technically expressed, the iron becomes more or less coarsely crystalline."

The revelation is certainly startling and of much interest to the practical man, for low-carbon steel containing less than 0·1% of carbon is frequently used, and makers and consumers, therefore, should carefully avoid any prolonged heating of such material at a dull red heat, an operation which previous to Mr. Stead's investigations had been considered quite harmless. It probably explains the singular brittleness and coarse fracture sometimes exhibited by very soft steel, and for which chemical analysis had never been able to account; for Mr. Stead proceeds then to show that this coarsely granular iron, while not necessarily so, is very liable to be brittle. If the orientation of the crystals throughout the mass assumes the same or nearly the same direction, which they have a tendency to do under the influence of prolonged heating at the above temperature, then the metal will be brittle; it will possess three lines of weakness in the three directions of the little cubes composing the mass—and, therefore, at right angles to each other. If the finished product be a plate, it will be easily fractured by a blow, along two directions at right angles to each other (see Fig. 4). The lines of fracture do not follow the junction lines of the grains, but cut through the grains following the planes of cleavage of the crystals, which being all in the same phase offer no resistance to the extension of the fracture. If, on the other hand, the orientation of the crystals of contiguous grains, instead of following the same general direction, is at various angles, the metal, although coarsely crystalline, will not be brittle—and thus it appears that a coarsely crystalline piece of iron may be less brittle than a finer-grained one. It remains true, however, that under the same condition of orientation of the cubic crystals, the finer-grained structure will be the toughest, for the junction lines are as many obstacles placed in the path of the fracture.

Planes of Weakness; Intercrystalline Brittleness.—A very remarkable fact connected with this brittleness of soft steel is that the fracture invariably takes place at an angle of 45% to the direction of the rolling, and at right angles to the surface of the steel, and the microscope shows that such is also the orientation of the crystals in metal made brittle by annealing. "Such material," Mr. Stead says, "can be bent and hammered close together when the bending is done longitudinally or at right angles to that direction, but it breaks off readily when bending is attempted at angles of 45° to the direction in which the plates were rolled." And further, "We are led from this to conclude that, just as light impresses a latent image on a bromide photographic plate which cannot be seen, but is developed and made manifest by the action of certain chemical agencies, so the rolling appears to impress a latent disposition in the steel to crystallize in certain fixed positions, and annealing develops it afterward."

Mr. Stead properly calls the peculiar brittleness described above "intercrystalline" brittleness, to indicate that the fracture follows the planes of cleavage of the crystals of the grains of ferrite, thus distinguishing it from "intergranular" brittleness, in which the line of fracture follows the lines of junction between the grains.

Intergranular brittleness follows no particular direction, and is typically found in cast iron. According to Mr. Stead, intergranular brittleness is due to a hard membrane or to some foreign matter enveloping the grains, or else it results from an imperfect cohesion between the grains brought about by some causes which at present are not well understood.

In order to ascertain the paths followed by the fracture, Mr. Stead places the thin section of metal, polished side down, over a V-space cut out of a solid piece of steel, and applies pressure at the back so as to bend the specimen and start a fracture. Figs. 4 and 5 show the appearance of two samples which have been thus treated. The brittleness of the metal of Fig. 4 is intercrystalline, for the lines of fracture cut through the grains, following the planes of cleavage of the crystals. Fig. 5, which represents the structure of blister steel, illustrates an instance of intergranular brittleness. The fracture follows the joints between the grains of ferrite, which are here surrounded by a membrane of cementite (the white portions in the illustration).

Although thousands of samples have been tested, none has been discovered with the intercrystalline weakness in sheets that had not been annealed after it left the rolls. The coarse structure developed by prolonged annealing at a dull red heat is invariably destroyed by heating to 900° C., the metal becoming then exceedingly tough.

Columnar Structure.—Mr. Stead has shown that the peculiar structure called "columnar," *i.e.*, a structure made up of columnar forms of iron which radiate from the exterior of a bar inward, can apparently only be developed in steel, when the carbon is removed from the surface by oxidizing agents, at a temperature below 850° C., and in no other way.

Fig. 6 exhibits the columnar structure assumed by a steel bar annealed in lime at about 700° C. for three weeks, by which treatment the totality of the carbon (0·21%) has been removed.

Whenever such a columnar structure is detected, therefore, it is apparently safe to infer that the steel has been exposed for a long time to an oxidizing atmosphere at a temperature not exceeding 850° C., an information which may, in certain instances, prove very valuable, and the microscope alone can furnish it.

On reheating to 900° C. and cooling quickly, the columnar granules assumed the normal equi-axial form, and no heat treatment could be discovered that would then restore the columnar structure.

Mr. Stead attributes the decarburization of steel annealed in ore or in any other substance capable of readily parting with its oxygen, not to the oxidizing action of the packing material itself, but to that of carbonic acid, which parts with some of its oxygen to oxidize the carbon of the steel. He believes that a mixture of carbonic acid and carbonic oxide, inert toward the iron itself (probably about two parts of CO to one of CO_2), penetrates into the steel, the action of the packing material being merely to provide oxygen for the formation of CO and CO_2.

Coarse, Brittle Envelope Caused by Close Annealing.—On this subject the following is quoted from Mr. Stead:

"On close annealing for 48 hours pieces of tin-plate bar, in most cases the external envelopes were coarse and the interiors most finely granular. On sudden shock some of them broke off short, and others after bending in one direction broke on straightening. When the outer coarsely granular envelopes were planed off, the interior fine-structured steel was tough and in every way perfectly safe material. It is clear, then, that commercially supplied soft steel may be rendered unsafe by the presence of an envelope of coarse structure. In certain processes to which wire manufacturers subject their steel this peculiar envelope is produced, and it is not an unusual thing to come across brittle wire rods in which the microscope clearly proves the presence of a ring of carbonless iron, the granules of which are arranged in columnar forms. All such samples, however, became tough and good on heating for a minute or two to 900° C. It is scarcely necessary to point out to practical men the danger there must be of using material enveloped with a coarsely granular ring, and this ring is often produced in annealing all classes of steel. It will also be equally clear that the safest way to anneal dead soft steel, if of coarse structure, is to heat it to 900° for a short time, and cool rapidly to 630°, and then cool slowly, but on no account to 'stew' it at between 750° and 650°; if, however, the structure is fine to commence with, long heat at 630° should be sufficient to remove all mechanical strains."

Structure of Burnt Steel.—Mr. Stead has confirmed Mr. Osmond's statement that burnt steel was made up of large grains, with intergranular spaces between them, and frequently covered with a film of oxide. The structure of burnt steel is well illustrated in Fig. 7.

Upon reheating burnt steel to 900° C., although the huge grains are broken into finer grains, the original oxidized envelopes retain their form. It is then easy to tell by the microscope whether or not steel has been burnt, and also if, after being burnt beyond recovery, it has again been reheated.

The Structure of Cemented Steel.—Prof. Arnold has carefully examined the microstructure of a series of Swedish iron bars which had been submitted to the cementation process for different lengths of time (the firing period varied from 8 to 11 days, the temperature being about 1,000° C., and the bars were allowed to cool for about 14 days), and he has illustrated his remarks with some very beautiful and instructive drawings. It was found that in the softer brands (resulting from a short firing period) the bars were only very slightly carbonized at the center (0·10% carbon, see Fig. 9), while the outside contained nearly 1% of carbon (Fig. 11), the carbon content increasing gradually from center to outside (see Fig. 10). After a longer firing the difference of carbonization between the outside and the center of the bars decreased rapidly, and after a sufficient length of time the carbon was found fairly uniformly distributed throughout the bar, amounting to about 1·6 to 1·8%.

Fig. 8 shows the structure of a cross section of the nearly carbonless Swedish iron untreated. It consists of irregular crystalline grains of ferrite or iron, through which are distributed the end views of the slag "fibers," which in the longitudinal section would appear as elongated streaks.

Figs. 9, 10 and 11 illustrate the structure of a soft-grade cemented bar (after a period of firing lasting eight days), respectively at the center, between the center and the outside, and at the outside. Fig. 9 consists of ferrite (the dark background), slag dots, and a single patch of perlite surrounded by thick, irregular walls of white cementite. Fig. 10, showing the structure of the median layer, is composed of large areas of ferrite and of perlite in nearly equal proportions. Fig. 11, taken near the outside of the bar, shows that in this region the iron is saturated; it is a mass of perlite.

Fig. 12 shows the structure of a bar fully converted (after a firing of 11 days). The bar contains about 1·6% of carbon and the structure is uniform throughout. It consists of large polygonal grains of perlite, surrounded by thick membranes of cementite.

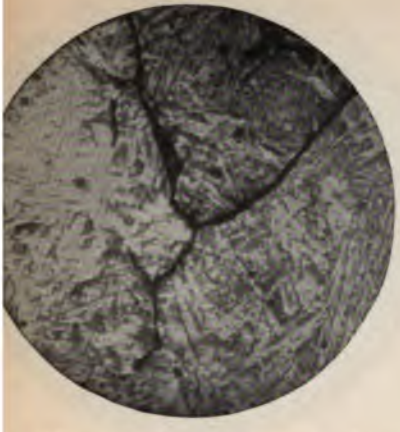
The author finds that it is not possible to introduce much over 2% of carbon into cemented iron, even after 10 conversions, and that after such treatment a large proportion of that element is found in the graphitic state. Let us recall in this connection that Mr. Saniter, on heating some pure iron wire in contact with an excess of carbon, found it possible to make it absorb nearly 3%, of which 0·53% was graphitic.

FIG. 8.—TRANSVERSE SECTION OF SWEDISH BAR IRON.
Carbon 0·0%. Magnified 230 diameters.

Prof. Arnold concludes his remarks as follows: "The above results seem to indicate that it would be possible to convert iron up to the saturation point (0·9% carbon) at a temperature of say 800° C., but that it is not possible to produce supersaturated high number bars unless the temperature of the chest be kept (as is the case in practice) at about 950° C."

Steel: An Alloy of Iron and Fe_3C.—The analogy existing between the constitution of carbon steel and that of certain metallic alloys has been pointed out by several investigators, and recently, more at length, by myself.

If steels containing various amounts of carbon be allowed to cool slowly from a high temperature, and if the temperature at which some evolutions of heat take place (as indicated by a retardation, an arrest, or an actual reversal in



. 7.—Burnt Bulb Angle showing Intergranular Spaces.

Magnified 30 diameters.

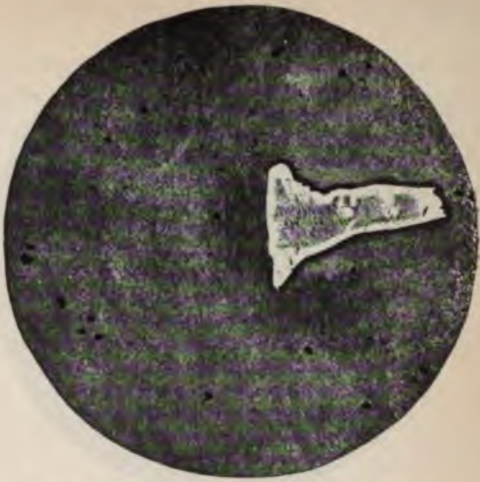


Fig. 9.—Soft-Grade Cemented Bar—Center of Bar—Carbon 0.1%.

Magnified 125 diameters.

10.—Soft-Grade Cemented Bar—Layer etween Center and Outside of Bar—Carbon 0.5%.

Magnified 125 diameters.

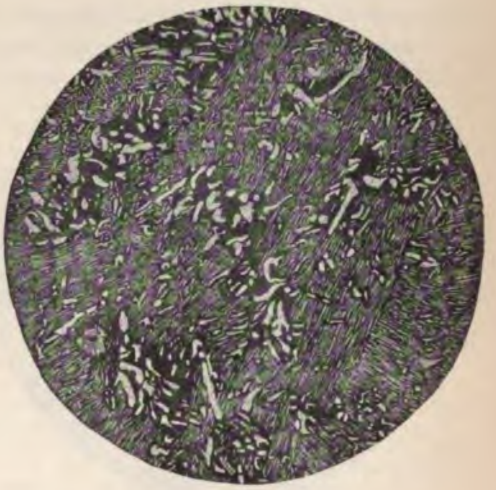


Fig. 11.—Soft-Grade Cemented Bar—Outside of Bar—Carbon 0.9%.

Magnified 125 diameters.

FIG. 12.—CEMENTED BAR, FULLY CONVERTED—
STRUCTURE UNIFORM THROUGHOUT—
CARBON 1.6%.

Magnified 33 diameters.

FIG. 15.—ALLOY OF SILVER AND COPPER—
CU, 65% ; AG, 35%.

Magnified 600 diameters.

FIG. 16.—ALLOY OF SILVER AND COPPER—
CU, 15% ; AG, 85%.

Magnified 600 diameters.

FIG. 17.—ALLOY OF SILVER AND COPPER—
ENTECTIC—CU, 28% ; SILVER, 72%.

Magnified 600 diameters.

the rate of cooling) be plotted as ordinates and the carbon-contents as abscisses, a curve is obtained very similar to the fusibility curves of alloys forming neither isomorphous mixtures nor definite compounds. This is well illustrated in Figs. 13 and 14.*

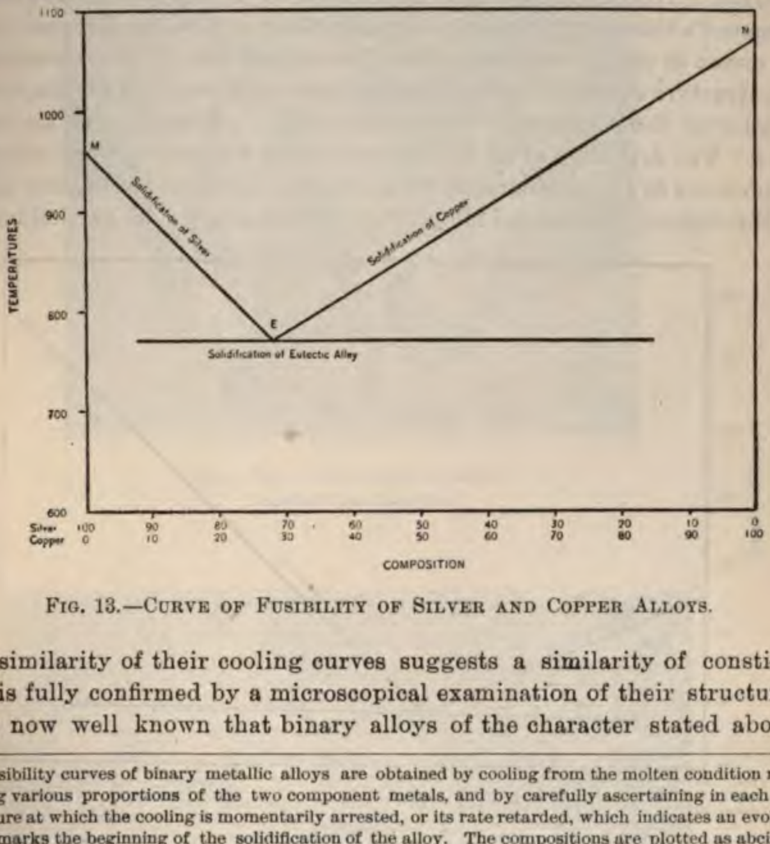


FIG. 13.—CURVE OF FUSIBILITY OF SILVER AND COPPER ALLOYS.

The similarity of their cooling curves suggests a similarity of constitution which is fully confirmed by a microscopical examination of their structure.

It is now well known that binary alloys of the character stated above are

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*The fusibility curves of binary metallic alloys are obtained by cooling from the molten condition mixtures containing various proportions of the two component metals, and by carefully ascertaining in each case the temperature at which the cooling is momentarily arrested, or its rate retarded, which indicates an evolution of heat and marks the beginning of the solidification of the alloy. The compositions are plotted as abcisses and the temperatures corresponding to the evolutions of heat as ordinates. In the case of alloys forming neither isomorphous mixtures nor definite compounds, a second evolution of heat occurs at a lower temperature, which is always the same, whatever the compositions of the alloy; when plotted it yields therefore a horizontal line. It indicates the solidification of the eutectic alloy, or alloy of lowest freezing point.

The temperatures at which retardations occur during the cooling of steel do not, of course, when plotted, give absolutely straight lines; experimental errors alone would preclude such possibility. Moreover, slight amounts of impurities have a notable influence upon the position of the retardations, and it is quite impossible to obtain various grades of steel absolutely free from impurities or containing exactly the same amount. It is sufficient for our reasoning that the curve should be made up, as it is, of two branches which meet in the region indicated. The two branches appear not to deviate very much from straight lines, and might possibly coincide exactly with them if experimental errors and other disturbing factors could be eliminated. It should also be stated that these evolutions of heat do not begin and end at the same temperature, but, on the contrary, cover a noticeable range—sometimes as much as 50° C. or more—so that the first and second evolutions run together long before a carbon-content of 0·8% has been reached, with the result that steel containing over 0·5% carbon appears to have only one retardation extending over a considerable range of temperature; the end of the first retardation merges into the beginning of the second.

Finally, when the metal contains less than about 0·25% carbon, a third retardation is detected, located between the first evolution and the point of recalescence. It has been purposely left out of this diagram because it would have rendered the resemblance which the latter bears to the curves of solubility or of fusibility just examined, less marked, and to little purpose, for the existence of a third evolution of heat confined to these narrow limits does not sensibly affect the strength of the deductions drawn from the striking analogy of the curves.

made up of particles of one of the constituents (the one present in excess) embedded in a matrix of the eutectic alloy (*i.e.*, the alloy having the lowest possible melting point; such an alloy is made up by the juxtaposition of particles or thin plates alternately of one and the other component). As an illustration I reproduce in Figs. 15, 16 and 17 some photomicrographs of Mr. Osmond's showing (1) the structure of an alloy of silver and copper containing an excess of copper with regard to the composition of the eutectic alloy, (2) the structure of an alloy containing an excess of silver, and (3) the structure of an alloy of the same series having exactly the composition of the eutectic mixture. The first alloy (Fig. 15) is composed of a matrix of pure copper (the white portions in the photograph) with particles of the eutectic alloy disseminated throughout; the second alloy (Fig. 16) is made up of particles of pure silver (the black portions in the photograph) surrounded by the eutectic alloy; finally the third alloy (Fig. 17) exhibits the characteristic structure of the eutectic mixtures; it is composed of small particles or thin plates alternately of silver and copper.

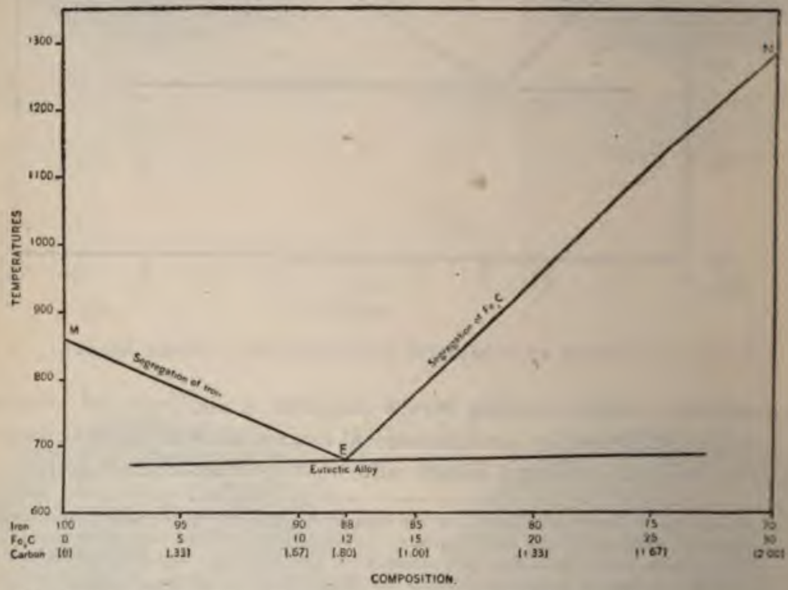

FIG. 14.—CURVE INDICATING EVOLUTIONS OF HEAT (ACCOMPANIED BY STRUCTURAL CHANGES) OCCURRING DURING THE COOLING OF STEEL.

The constitution of steel as revealed by the microscope is in every way similar to that of such metallic alloys, for steel is composed, according to its degree of carburization, of particles of iron (ferrite) or of the carbide Fe_3C (cementite) embedded in a matrix of perlite; *i.e.*, of a constituent made up by the juxtaposition of thin plates alternately of ferrite and cementite, and similar, therefore, to the eutectic mixture of metallic alloys. The beautiful photomicrographs, reproduced in Figs. 18, 19 and 20, were taken by Mr. Osmond. Fig. 18 exhibits the structure of a steel containing an excess of ferrite with

FIG. 18.—STEEL—0.45% CARBON.
Magnified 1,000 diameters.

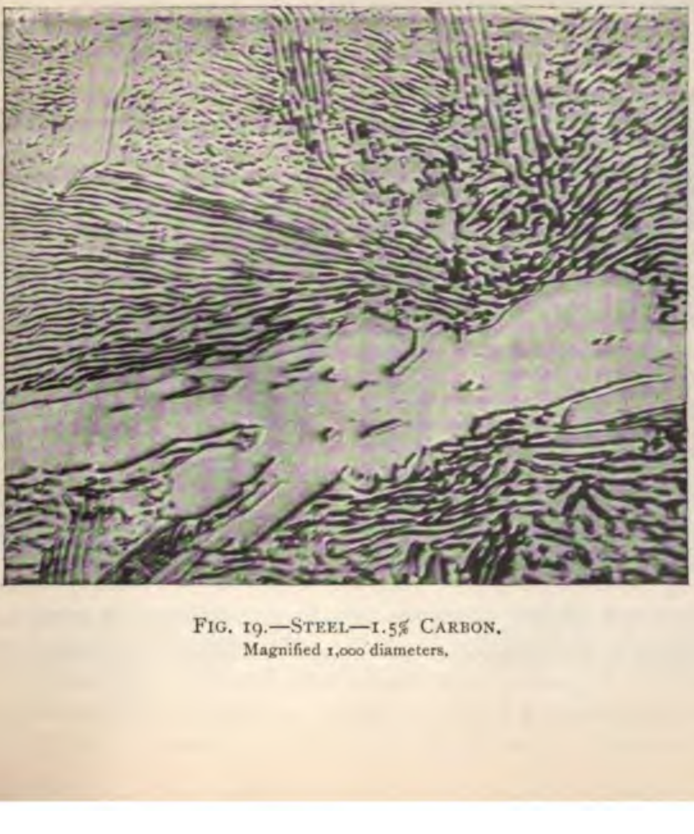

FIG. 19.—STEEL—1.5% CARBON.
Magnified 1,000 diameters.

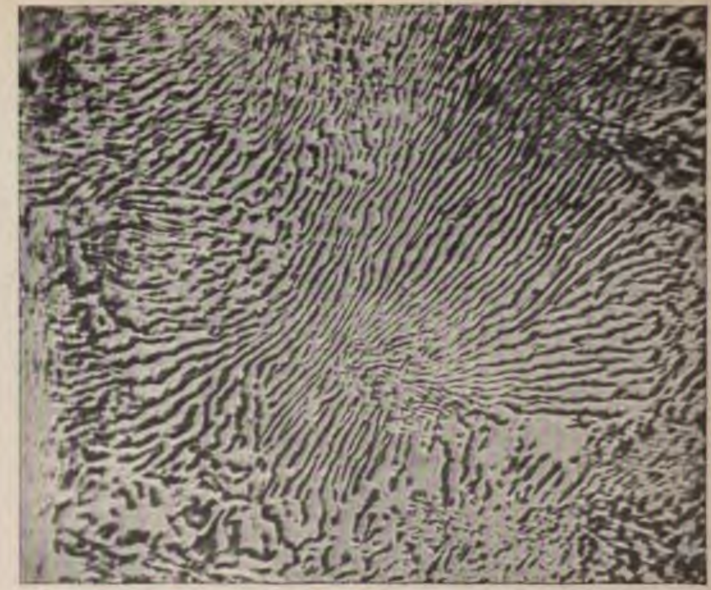

FIG. 20.—STEEL.—1% CARBON.
Magnified 1,000 diameters.

FIG. 22.—SURFACE OF FUSIBILITY OF ALLOYS OF LEAD, TIN, AND BISMUTH.
Reproduction of a model in relief.

FIG. 23.—ALLOY CONTAINING 21% PB, 5.9% AND 74% BI, SHOWING THE THREE SUCCESSI STAGES OF THE SOLIDIFICATION.

regard to the composition of the eutectic alloy (*i.e.*, of perlite); the white portions in the photograph correspond to the ferrite, the composite constituent to the perlite. Fig. 19 shows the structure of a steel containing an excess of cementite (the white areas, standing in relief). The steel of Fig. 20 has the composition (at least in the regions shown) of the eutectic alloy; it is a mass of perlite.

The analogy, then, between the constitution of steel and that of binary metallic alloys forming neither isomorphous mixtures nor definite compounds is very striking, and leads to the conclusion that steel is a true alloy of iron and the carbide Fe_3C, perlite corresponding to the eutectic alloy of the series.

Allotropic Theory.—The controversy between the supporters and the opponents of the allotropic theory was continued during 1898, and it would appear that the "allotropists" held more than their own.

It does not seem reasonable to doubt any longer the existence of allotropic iron. The fact that practically carbonless iron exhibits two retardations which cannot be attributed to chemical causes, and must therefore be the manifestations of changes of internal energy (without any accompanying change of state), is in itself a proof of allotropy, for it is the very definition of allotropy. The sharp change of magnetic properties which is so closely identified with the second retardation is another very strong indication of allotropic changes. Until the anti-allotropists have explained these two phenomena on some other reasonable ground, the belief in the existence of allotropic iron is most natural and indeed inevitable. The only question which appears to be still debatable is whether allotropic iron is the cause of hardening by sudden cooling.

Mr. Howe has shown, by numerous and carefully conducted experiments, that in cooling very suddenly some practically carbonless iron wires, the specific effects of hardening (increase of tenacity, of elastic limit, of hardness, decrease of ducility) are induced to such an extent that they cannot reasonably be ascribed to the extremely small amount of carbon present (0·009% in one instance), nor to any other cause outside of allotropy. It would seem then that allotropic iron is hard, and that its retention by sudden cooling does at least contribute to the final hardness of the metal. Whether allotropic iron is the sole primary cause of hardening, however, the carbon acting merely as a brake in preventing the passage of iron to its soft condition during sudden cooling, does not appear to have been shown conclusively. The proof of the proposition involves very great difficulties.

OTHER ALLOYS THAN STEEL.

The microscope is evidently destined to play an important part in the solution of many problems connected with metallic alloys.

The modern theory of alloys so brilliantly worked out from the appearance of their curves of fusibility, remained somewhat in the tentative stage until the microscope, by revealing the structural composition of alloys, fully confirmed the conclusions reached, thus dispelling any remaining doubt concerning the signification of the evolutions of heat observed during their solidification. *The constitution of binary alloys* which form neither isomorphous mixtures per definite compounds, has been briefly described in the preceding pages and is illustrated in Figs. 15, 16 and 17.

Ternary Alloys.—Until the recent important investigations of Mr. Charpy, the microstructure of binary alloys only had been studied systematically jointly with their physical properties and their fusibility curves. The latter investigator, however, has taken hold of a number of ternary alloys used in the arts as anti-friction metals and has submitted them to a careful examination, determining their compressive strength, hardness, surface of fusibility and microstructure.

The results of Mr. Charpy's researches are of great interest to the industrial world, in view of the industrial importance of the material investigated, and of the obscurity which still surrounds a proper selection of the components of bearing metal.

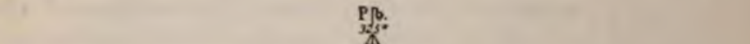

FIG. 21.—REPRESENTATION OF FUSIBILITY OF ALLOYS OF LEAD, TIN AND BISMUTH.

Mr. Charpy applied to the study of these ternary alloys the methods which had been used with so much success in the case of binary alloys. Using triaxial diagrams he constructed the surfaces of fusibility of his alloys exactly as the curves of fusibility of binary alloys had been constructed; *i.e.*, by allowing a number of alloys containing various proportions of the three component metals to cool slowly from the molten state and noting in each case the temperature at which an evolution of heat took place. If a perpendicular be then raised from each point to the plane of the triangle, and a distance measured upon that perpendicular proportional to the temperature recorded in the case of the alloy represented by the foot of the perpendicular, the loci of all these constitute the surface of fusibility of ternary alloys, which corresponds to the curve of fusibility of binary alloys. Fig. 21 shows the surface of fusibility of alloys of lead, tin and bismuth. Fig. 22 exhibits the same surface, reproduced from a model in relief.

By attributing to the evolutions of heat observed during the solidification of his alloys, meanings identical to those known to be true in the case of binary mixtures, he was able to foretell the constitution of the ternary alloys examined, and his conclusions were fully confirmed by the appearance of their structure under the microscope. He found that when ternary alloys forming neither isomorphous mixtures nor definite compounds are allowed to cool from a molten condition, when a certain temperature is reached (which varies with the composition of the alloy), one of the component metals begins to solidify and continues to do so until a certain lower temperature is reached, when a second constituent, this time a binary one, separates out from the portion remaining liquid; this binary constituent is made up of small particles or plates of two of the component metals, and is therefore similar in structure to the eutectic alloys of binary alloys. Finally, when a still lower temperature is attained, the portion remaining liquid solidifies as a whole and at a constant temperature; it is the eutectic alloy, or alloy of lowest melting point of ternary mixtures, and it possesses the same characteristic structure as the binary eutectic alloys, being made up by the close juxtaposition of particles or thin plates of the three component metals. The three successive stages of the solidification is well illustrated in Fig. 23, which shows the structure of an alloy containing 21% of lead, 5·5% of tin, and 74% of bismuth, and which corresponds therefore to the point A in Fig. 21. The following structural constituents are easily distinguished: (1) Large areas of metallic bismuth shown in white in Fig. 23, and corresponding to the grains of bismuth which solidified between 175° and 125° C.; (2) around each of these grains of bismuth a zone in which particles of bismuth alternate with particles of tin; the latter have been dug out by the acid and appear black in the photograph; this binary constituent was formed between 125° and 96° C., while the point A was being displaced along the line $E'' e$ (Fig. 21); (3) finally, as a matrix, the ternary eutectic mixture, which appears homogeneous under a low magnification, but in which may be distinguished, under high magnification, some bright grains of bismuth; some grains of tin dug out by the acid, and some grains of lead covered, through the action of the etching acid, with a layer of white chloride of lead.

Mr. Charpy further found that the constitution of ternary alloys forming some binary definite compounds may be inferred from their curves of fusibility in a similar way, provided these definite compounds are treated as components of the alloy instead of the metals themselves. In the case of alloys of copper, antimony and tin, for instance, which contain the binary compounds $SbCu_2$ and $SnCu_3$, the alloys rich in copper should be considered as being composed not of copper, tin and antimony, but of copper, $SbCu_2$, and $SnCu_3$.

Mr. Charpy has shown very conclusively that bearing metals should be made up of hard grains embedded in a plastic matrix, and that a micrographic test is required to ascertain whether the alloy possesses such a constitution.

THE MINING STOCK EXCHANGES IN 1898.

In the following pages we have summarized the fluctuations and transactions in the mining stocks traded in on the various exchanges in the United States and London. In the United States there is very little speculation in mining stocks at the present time, outside of Boston, where the shares of copper mining companies in Arizona, Michigan and Montana are dealt in extensively. In Denver and Colorado Springs there is considerable speculation in the shares of various Colorado mining companies, while Salt Lake City has a market which sometimes shows considerable activity, but transactions at these centers is chiefly confined to stocks of local interest. The speculation

FLUCTUATIONS OF MINING STOCKS AT BOSTON DURING 1898.

Name of Company.	Par Value.	January.		February.		March.		April.		May.		June.	
		H.	L.	H.	L.	H.	L.	H.	L.	H.	L.	H.	L.
Copper:													
Adventure Con. *b*	$25	......	......	......	......	......	......	......	......	......	......	......	......
Allouez *b*	25	1·00	·50	3·00	1·13	2·75	1·38	2·00	1·75	3·50	1·88	5·75	3·00
Arcadian	25	......	......	......	......	......	......	......	......	......	......	......	......
Arnold *b*	25	9·00	2·00	10·00	7·50	12·00	8·38	12·00	10·00	16·50	11·50	17·00	13·88
Ash Bed *b*	25	......	......	3·88	2·25	3·00	2·13	2·75	2·25	3·38	2·63	4·25	3·00
Atlantic *b*	25	28·75	23·38	32·00	28·50	34·00	23·25	28·50	24·00	27·50	25·00	32·00	27·00
Baltic *b*	25	......	......	......	......	......	......	13·00	11·50	......	......	24·25	16·75
Boston & Montana *a*	25	160·00	149·00	190·00	163·75	191·75	158·00	177·75	167·00	202·00	178·50	211·50	195·25
Butte & Boston *a*	25	26·63	23·25	28·50	22·75	27·00	18·50	25·38	22·88	26·50	23·75	26·00	20·50
Calumet & Hecla *b*	25	500·00	489·00	530·00	507·00	535·00	510·00	520·00	515·00	555·00	520·00	575·00	550·00
Centennial *b*	25	11·75	8·00	12·50	9·00	14·75	10·50	13·50	11·00	15·88	12·00	17·00	13·75
Franklin *b*	25	17·00	10·38	17·00	14·00	15·50	12·00	12·00	10·50	12·00	10·38	15·00	11·50
Humboldt *b*	25	2·00	·80	2·50	1·50	2·00	1·50	1·75	1·50	4·50	1·88	6·25	4·13
Isle Royal *b*	25	......	......	......	......	......	......	......	......	......	......	......	......
Old Dominion *c*	25	25·25	23·50	31·00	24·50	32·13	23·88	27·25	24·25	28·50	25·88	28·88	22·00
Osceola *b*	25	43·00	39·00	44·00	39·50	44·88	35·00	41·00	38·50	49·50	41·50	55·25	46·38
Parrot *a*	10	......	......	24·50	21·75	25·75	21·00	24·00	22·00	23·00	18·50	24·00	22·00
Quincy *b*	25	118·00	108·50	117·00	108·50	114·00	105·00	110·00	106·00	116·50	107·00	118·50	113·00
Tamarack *b*	25	145·00	131·00	160·00	145·00	159·50	139·00	154·00	142·00	167·00	154·00	167·00	154·00
Tecumseh *b*	25	3·50	1·00	3·38	2·50	3·50	2·00	3·00	1·50	3·00	1·75	3·38	2·50
Washington *b*		......	......	......	......	......	......	......	......	......	......	......	......
Winona *b*	25	......	......	......	......	......	......	......	......	......	......	......	......
Wolverine *b*	25	19·50	18·00	23·00	19·50	23·00	17·50	20·50	18·63	22·50	20·00	28·00	21·00
Gold:													
Boston & C. C. *d*	1	......	......	......	......	......	......	......	......	......	......	·20	·15
Gold Coin *d*	1	2·50	1·80	2·00	1·00	1·13	1·00	......	......	1·00	·88	1·00	·90
Merced *e*	15	5·75	4·50	5·75	4·75	6·00	4·00	4·75	4·63	5·00	4·50	......	......
N. A. Gold Dreg. *a*	10	......	......	......	......	......	......	......	......	......	......	......	......
Pioneer *e*	10	6·88	5·50	......	......	5·75	5·00	5·50	5·00	6·00	5·00	6·25	5·38
Santa Ysabel *e*	5	7·50	5·25	8·00	5·50	7·13	5·50	6·00	5·00	5·88	4·75	5·25	4·00
Victor *d*	5	......	......	......	......	......	......	......	......	......	......	7·25	6·50
Miscellaneous:													
Ætna *e*	5	......	......	3·88	3·50	......	......	......	......	4·00	2·25	4·00	......
Bonanza	10	·35	......	......	......	......	......	......	......	......	......	......	......
Breece *d*	25	·35	·30	·35	......	......	......	......	......	......	......	·45	......
Dominion Coal *f*	100	23·75	20·50	22·75	13·50	20·75	15·00	19·50	17·50	24·50	17·50	24·50	22·00
Dominion Coal *g*	100	107·50	106·00	108·00	107·00	108·00	99·00	101·50	91·83	109·25	100·00	118·00	109·50
Dunkin *d*	25	......	......	......	......	......	......	......	......	......	......	......	......
Illinois Steel *j*	100	55·75	48·00	55·50	46·00	50·00	44·00	......	......	......	......	......	......
Lake Superior Iron *b*	25	......	......	......	......	......	......	......	......	......	......	......	......
Napa Con. Q. *e*	7	8·50	7·00	8·25	7·00	8·00	6·00	6·75	6·00	7·00	6·00	......	......
New Idria e	5	......	......	......	......	......	......	......	......	6·63	6·00	......	......

which was formerly so active in San Francisco has now been dead for a long time, and the transactions in mining stocks at the New York Exchange are insignificant. There is, however, a good deal of trading in New York and Philadelphia in the shares of coal and iron mining companies and many industrial companies which are engaged in the mining industry either as miners, smelters or consumers of the first mineral products. A large proportion of these companies are recognized to be strong and ably managed concerns, from which regular dividends may be expected, and the transactions in their shares are largely in the nature of buying for investment. At the present time when the bonds of first-class railway companies are selling regularly at prices which permit of a realization of hardly 3½% on the investment, and the shares of the same companies at prices which do not yield 4½%, it is becoming necessary for capital to turn to the securities of the so-called industrial companies as investments from which a better return for the money can be obtained.

The value of such securities is governed by the earning power of the companies which they represent, and when this has been demonstrated by the

FLUCTUATIONS OF MINING STOCKS AT BOSTON DURING 1898.—*Continued.*

Name of Company.	July.		August.		September.		October.		November.		December.		Sales.
	H.	L.	H.	L.	H.	L.	H.	L.	H.	L.	H.	L.	
Copper.													
Adventure Con. *b.*	......	......	......	......	......	......	......	......	......	......	10·00	9·63	5,950
Allouez *b*	5·00	3·25	4·75	4·00	4·25	3·00	5·88	3·25	6·00	4·75	7·00	4·93	143,834
Arcadian	24·00	20·00	23·25	20·00	24·00	21·50	29·00	23·25	40·00	29·00	73·00	36·50	148,211
Arnold *b*	20·25	14·25	19·00	16·00	17·13	15·50	18·50	16·00	18·50	13·75	15·00	8·00	204,352
Ash Bed *b*	3·75	2·88	3·25	2·75	2·75	2·00	3·00	2·00	3·00	2·00	2·25	1·50	55,776
Atlantic *b*	36·00	26·50	29·13	26·00	27·00	25·00	29·00	25·00	35·50	28·75	32·50	30·00	45,648
Baltic *b*	20·00	17·75	20·00	17·50	18·00	15·00	19·25	16·00	27·88	18·00	32·25	25·00	165,654
Boston & Montana *a*	210·00	203·00	231·00	206·00	228·00	219·75	247·50	224·50	243·00	226·50	288·00	228·00	422,446
Butte & Boston *a*	25·88	23·50	26·50	24·25	24·63	21·50	41·50	23·00	59·25	48·00	94·25	57·25	875,779
Calumet & Hecla *b*	575·00	560·00	600·00	575·00	590·00	275·00	590·00	580·00	650·00	595·00	649·00	605·00	5,202
Centennial *b*	15·50	13·75	21·00	14·50	19·50	16·88	20·00	17·00	27·50	20·75	40·00	24·50	473,148
Franklin *b*	13·00	11·00	15·00	12·00	14·75	13·00	15·63	13·00	21·00	14·88	24·00	15·13	107,657
Humboldt *b*	6·13	4·88	6·25	4·63	5·06	4·50	5·88	4·00	5·50	3·75	4·00	3·00	81,699
Isle Royal *b*	......	......	......	......	27·00	22·50	25·00	23·00	33·00	24·50	50·00	27·00	67,510
Old Dominion *c*	25·00	22·25	29·75	25·13	29·13	27·00	30·00	27·00	38·00	28·25	40·00	31·25	544,118
Osceola *b*	48·00	45·25	58·50	47·38	57·25	52·75	65·00	56·00	77·00	64·45	87·00	66·75	294,754
Parrot *a*	23·38	22·50	23·50	22·50	24·00	22·50	24·25	23·50	30·00	24·00	40·00	28·00	38,029
Quincy *b*	116·00	110·50	120·00	110·00	119·50	115·50	127·00	118·00	144·50	124·00	149·00	126·00	26,597
Tamarack *b*	157·00	154·00	175·00	153·00	179·00	171·00	182·00	171·00	199·00	173·00	190·00	168·00	31,112
Tecumseh *b*	2·50	2·00	4·75	2·00	3·50	2·50	3·50	2·88	6·13	3·00	6·00	4·50	26,283
Washington *b*	·80	·75	2·38	1·75	1·50	1·00	1·75	1·00	2·00	1·50	1·50	1·00	64,978
Winona *b*	......	......	......	......	......	......	......	......	......	......	15·50	14·25	9,489
Wolverine *b*	24·50	22·00	27·88	24·25	28·00	22·25	28·50	26·00	34·25	27·50	35·75	29·50	120,355
Gold:													
Boston & C. C. *d*	·30	·15	......	......	·25	......	·30	·25	·40	·25	·45	·35	14,355
Gold Coin, *d*	1·00	·90	·93	·75	·75	·30	·90	·25	·90	·50	1·00	·40	26,877
Merced *e*	......	......	8·00	6·00	6·50	4·25	6·00	5·00	5·50	5·25	7·50	5·00	18,105
N. A. Gold Dreg. *a*	......	......	......	......	......	......	......	......	......	......	19·25	14·38	21,775
Pioneer *e*	6·00	5·00	5·75	5·13	5·50	4·88	5·25	4·75	5·25	4·88	6·00	4·88	64,071
Santa Ysabel *e*	......	......	5·75	4·88	5·50	4·50	7·75	4·25	7·50	6·50	7·00	6·00	18,485
Victor *d*	......	......	7·00	5·75	8·00	6·63	......	......	7·25	5·75	5·50	2·88	45,907
Miscellaneous:													
Ætna *e*	4·00	......	......	......	......	......	......	......	......	......	4·00	3·50	1,725
Bonanza	·45	......	......	......	......	......	1·05	·73	1·15	·75	1·80	1·20	49,375
Breece *d*	·50	......	·95	·65	·73	·45	......	......	......	......	......	......	8,350
Dominion Coal *f*	23·50	21·00	23·88	22·50	26·50	21·00	30·00	25·00	36·00	29·00	35·00	30·13	283,712
Dominion Coal *g*	109·00	108·00	112·50	111·00	113·00	111·50	115·00	111·50	117·00	114·00	118·00	116·50	4,357
Dunkin *d*	......	......	......	......	·27	·10	·23	·20	·25	......	·30	·20	6,000
Illinois Steel *j*	60·13	57·00	73·88	59·00	74·50	64·50	90·00	65·00	......	......	......	......	44,244
Lake Superior Iron *b*	......	......	......	......	......	......	......	......	......	......	......	......	100
Napa Con. Q. *e*	7·00	......	7·00	......	......	......	......	......	......	......	7·75	5·25	1,400
New Idria *e*	7·00	......	......	......	......	......	......	......	......	......	7·63	6·50	890
Total	......	......	......	......	......	......	......	......	......	......	......	......	[illegible],584,398

(*a*) Montana. (*b*) *Michigan.* (*c*) *Arizona.* (*d*) Colorado. (*e*) California. (*f*) Nova Scotia. (*g*) Preferred. (*j*) Illinois.

FLUCTUATIONS OF MINING STOCKS AT COLORADO SPRINGS DURING 1898.

Name of Company.	Par Value.	January.		February.		March.		April.		May.		June.	
		H.	L.	H.	L.	H.	L.	H.	L.	H.	L.	H.	L.
Alamo	$1·00	·03	......	·03	·02	·03	·02	.02	......	·02	......	·03	·02
Anaconda Gold	5·00	·41	·38	·43	·40	·38	·30	.39	·29	·53	·40	·53	·45
Argentum-Juniata	2·00	·20	·18	·21	·19	·21	·18	.26	·18	·27	·23	·27	·25
Battle Mountain	1·00	......	......	......	......	......	......	......	......	......	......	......	......
Columbine	1·00	......	......	......	......	......	......	......	......	......	......	......	......
Creede & Cripple Creek	1·00	·04	·03	·04	·03	·05	·03	......	......	......	......	·04	·03
Cripple Creek Cons	1·00	·10	·09	·11	·10	·11	·08	.09	·08	·09	·08	·09½	·09
Currency	1·00	......	......	......	......	......	......	......	......	......	......	......	......
Des Moines	1·00	......	......	......	......	......	......	......	......	......	......	......	......
Elkton Cons	1·00	·90	......	·88	·75	·84	·69	.76	·51	·95	·70	·94	·88
El Paso	1·00	·20	·16	·17	·08	·09	·05	.07	·06	·07	·06	·09	·07
Fanny Rawlings	1·00	·17	·15	·21	·15	·24	·17	.20	·16	·17	·15	·20	·17
Favorite	1·00	·04	·03	·04	·02	·03	·02	.03	......	·03	......	·03	......
Findley	1·00	......	......	·02	·01	......	......	.02	·01	·04	·02	·03	......
Franklin	1·00	......	......	......	......	......	......	......	......	......	......	·05	......
Gold Crater	1·00	......	......	1·00	......	1·00	......	......	......	·80	......	......	......
Golden Fleece	1·00	·62	·52	·52	·44	·45	.35	·41	·24	......	......	......	......
Gold & Globe	1·00	·04	......	·03	·02	......	......	·05	......	......	......	·03	·02
Independence	1·00	......	......	......	......	......	......	......	......	......	......	......	......
Ingham Cons	·50	·14	·09	·16	·07	·09	.05	·07	·06	·07	·06	·07	·05
Isabella	1·00	·36	·31	·32	·26	·26	.26	·22	·20	·24	·21	·28	·24
Jack Pot	1·00	·05	·04	·05	·04	·04	.03	......	......	·04	·03	·07	·03
Lexington	1·00	·07	......	·07	......	......	......	......	......	·07	......	......	......
Lillie	5·00	·70	·65	·76	·67	·85	.74	·82	·75	·92	·84	·90	·85
Magnet Rock	1·00	......	......	......	......	......	......	......	......	......	......	......	......
Matoa	1·00	·17	·16	·16	·14	·15	.12	·14	·13	·15	·14	·24	·21
Mobile	1·00	......	......	......	......	......	......	......	......	......	......	......	......
Mollie Gibson	5·00	·29	·19	·22	·20	22	.20	·21·	·20	·25	·21	......	......
Moon-Anchor	1·00	·99	·87	1·00	·95	·99	.89	·97	·62	·76	·67	·94	·75
Mt. Rosa	1·00	·17	·11	·14	......	·15	.10	·13	·10	·13	·12	·12	......
New Haven	1·00	......	......	·01	......	......	......	......	......	......	......	·01	......
Oriole	1·00	·01	......	......	......	......	......	......	......	......	......	......	......
Pharmacist	1·00	·08	·06	·07	·06	·06	.05	·07	·05	·06	·05	·06	·05
Portland	1·00	·81	·74	·82	·75	·79	.74	·90	·74	·00	·88	1·17	1·06
Prince Albert	1·00	......	......	......	......	......	......	·03	·02	......	......	......	......
Specimen	1·00	·04	·03	·04	·03	·03	.02	·06	·02	·04	·03	·04	·03
Theresa	1·00	·08	·07	·13	·07	·07	.05	·07	·06	·07	·06	·07	·06
Trachyte	1·00	·02	·01	·02	·01	·01	......	......	......	·01	......	·02	·01
Union	1·00	·15	·10	·14	·12	·13	.09	·11	·09	·12	·10	·17	·11
Work	1·00	·04	·01	·04	·03	·04	.02	·03	......	·03	·02	·04	·03

payment of regular dividends for a series of years, as in the case of the American Sugar Refining Co., the National Lead Co. and certain others, the shares rank high as worthy investments. Newer industrial companies, such as the Federal Steel Co., the General Chemical Co., the American Smelting and Refining Co., the American Steel and Wire Co., etc., have still to demonstrate their value before they can reach the position occupied by the group of older industrials. There is no doubt that most of these industrial companies, both the older and the newer, were largely over-capitalized at the time of organization, and it is also unfortunate that many of them adopt the policy of making incomplete reports of their earnings and resources to their shareholders, wherefore investors are liable to be left in the dark as to the actual results of operations and the value of their holdings. A franker policy should be adopted in this respect.

With regard to mining stocks, many of which, especially those of copper mining companies, are rated with the industrials in New York and Boston, it is necessary to remember that a mine does not pay its dividends out of regular earnings from manufacturing and commercial transactions, but out of the wealth deposited by nature. Consequently a mine in paying dividends is draw- on its resources, which are being diminished not only by the amount of the

FLUCTUATIONS OF MINING STOCKS AT COLORADO SPRINGS DURING 1898.—*Continued.*

Name of Company.	July.		August.		September.		October.		November.		December.		Sales.
	H.	L.	H.	L.	H.	L.	H.	L.	H.	L.	H.	L.	
Alamo	·03		·04	·03	·05	·04	·04	·03	·05	·04	·05		421,900
Anaconda Gold	·43	·39	·38	·34	·70	·40	·63	·49	·64	·40	·85	·40	187,176
Argentum-Juniata	·25	·22	·22	·20	·25	·19	·24	·22	·40	·22	·46	·35	502,239
Battle Mountain					·39	·33	·39	·30	·45	·28	·43	·39	100,140
Columbine					·19	·15	·16	·09	·15	·13	·16	·14	185,600
Creede & Cripple Creek	·04		·05	·03	·08	·05	·06	·04	·06	·01	·05		82,200
Cripple Creek Cons	·09	·08	·09	·06	·11	·08	·09	·08	·09	·06	·09	·08	713,317
Currency			·04	·03	·05	·02	·04	·02	·05	·03	·04	·03	92,800
Des Moines			·04	·01	·02	·01	·02		·02		·02		141,667
Elkton Cons	·95	·85	1·17	·92	1·27	1·16	1·19	1·00	1.06	·93	·96	·70	2,505,271
El Paso	·09	·08	·12	·08	·17	·12	·11		·14	·12	·13	·10	827,632
Fanny Rawlings	·20	·18	·40	·20	·59	·38	·52	·44	·48	·39	·47	·35	642,295
Favorite	·03		·04	·03	·05	·04	·03	·02	·03	·02	·04	·03	199,333
Findley	·06	·03	·10	·05	·10	·07	·10	·08	·19	·12	·17	·15	2,889,040
Franklin			·03	·02									273,000
Gold Crater													100
Golden Fleece			·24	·21	·40	·25	·26	·25	·30	·25	·30	·29	13,125
Gold & Globe	·03		·02		·05	·02	·05	·04	·07	·05	·06	·05	208,650
Independence					·70	·63	·70	·50	·61	·53	·62	·52	414,500
Ingham Cons	·06		·08	·07	·10	·07			·09	·07	·08	·07	584,930
Isabella	·26	·20	·29	·22	·28	·24	·27	·24	·29	·25	·32	·26	1,839,729
Jack Pot	·07	·06	·13	·08	·15	·09	·17	·11	·26	·21	·37	·24	2,436,139
Lexington					·08	·07	·08	·07	·08	·07	·07		41,300
Lillie	·90	·87	1·25	·90	1·22	1·00	1·10	1·07	5·50	1·30			214,527
Magnet Rock					·02		·02	·01	·02	·01	·02	·01	422,090
Matoa	·23	·20	·27	·21	·41	·30	·34	·29	·36	·28	·34	·27	688,400
Mobile	·01				·03	·01	·03	·02	·04	·03	·04	·03	1,667,600
Mollie Gibson	·21	·20	·22	·19	·23	·20	·22	·20	·28	·22	·29	·23	154,569
Moon-Anchor	·98	·87	1·19	·92	1·23	1·15	1·19	1·11	1·25	1·11	1·28	1·17	710,779
Mt. Rosa	·12		·16	·12	·20	·18	·20	·17	·25	·18	·20	·18	445,090
New Haven	·01		·03	·01	·03	·02	·02		·02		·02		775,025
Oriole					·02	·01	·02	·01	·02	·01	·02	·01	256,360
Pharmacist	·06	·04	·04	·02	·10	·03	·07	·03	·05	·04	·04		2,963,850
Portland	1·17	1·03	1·44	1·13	1·51	1·41	1·46	1·38	1·48	1·41	1·49	1·42	927,375
Prince Albert	·02		·03	·01	·03	·02	·02		·03	·02	·02		383,600
Specimen	·04		·04		·06	·04	·05	·03	·04		·06	·04	458,000
Theresa	·07	·06	·07	·06	·10	·07	·08	·07	·09	·07	·08	·07	496,400
Trachyte	·02	·01	·03	·01	·03	·02	·03		·05	·03	·05	·04	11,988,454
Union	·17	·13	·19	·13	·27	·16	·21	·19	·29	·19	·20	·18	4,526,470
Work	·05	·04	·10	·04	·11	·08	·09	·08	·15	·09	·16	·10	4,417,805
Total shares sold													46,148,477

dividends paid, but also by the cost of producing the mineral which yields the dividends, and a mining stock ought properly to pay not only a fair interest on its cost, but also an extra amount to provide for the eventual redemption of the principal. In stocks which are actively traded in the stock exchanges this policy is apt to be overlooked, so that the shares of some companies, such as the Lake Superior copper mines, which have proved themselves to be regular dividend payers for a long series of years, often sell at prices which permit a return of only 6 or 7% on the investment, leaving the loss of principal by the eventual exhaustion of the mine to fall upon the final holders of the shares. The evil result of such inflation of values may be escaped by speculators and many temporary investors, but in the case of mining companies, for the shares of which no active market is to be anticipated, it is quite unsafe to purchase at prices which will not enable the dividends to cover redemption of principal as well as afford a proper interest upon it in view of the extra hazard which is taken.

THE BOSTON MINING STOCK MARKET IN 1898.

The Boston mining stock market opened the year in comparative quiet,

FLUCTUATIONS OF MINING STOCKS AT DENVER DURING 1898.

Name of Company.	Par Value.	January.		February.		March.		April.		May.		June.	
		B.	A.	B.	A.	B.	A.	B.	A.	B.	A.	B.	A.
	$												
Anaconda	5·00	·28	·46	·38	·45	·30	·42	·25	·39	·35	·55	·39	·55
Anchoria-Leland	1·00	·93	·104	·85	1·10	·68	·98	·42	·81	·58	1·00	......	......
Arcadia Cons	1·00	·02	·03	·01	·03	·006	·02	·005	·02	......	......	·011	·02
Argentum-Juniata	1·00	·18	·21	·19	·21	·18	·21	·15	·20	·22	·28	·24	·28
Bankers	1·00	·04	·07	·06	·08¼	·04	·08	·04	·07	·04	·07	·05	·08
Banner	1·00	......	......	......	......	......	......	......	......	·06	......	·06	·08
Ben Hur	1·00	......	......	·02	·04	·01	·04	·01	·03	·02	·03	......	......
Big Six	1·00	·08	·21	·09	·20	·01	·15	·03	·16	·03	·15	·03	·15
Creede & Cripple Creek	1·00	·03	·05	·03	·13	·02	·05	......	......	......	......	......	......
Cripple Creek Cons	1·00	·10	·11	·09	·12	·08	·11	·06	·13	·07	·11	·06	·10
Della S.	1·00	·04	·20	·05	·12	·03	·20	·01	·03	......	......	......	......
Elkton Cons	1·00	·81	·89	·75	·88	·68	·85	·53	·74	·70	·95	·88	1·00
El Paso	1·00	14	·20	·08	·18	·05	·10	·05	·08	·05	·08	·06	·10
Enterprise	1·00	·03	·06	·03	·06	·02	·05	·01	·05	·03	......	·03	·05
Fanny Rawlings	1·00	·14	·18	·14	·25	·14	·25	·13	·24	......	..	·17	·22
Findley	1·00	·01	·02	......	......	......	......	......	......	......	......	......	......
Garfield Cons	1·00	·05	·07	·06	·08	·04	·07	·05	·07	......	......	·06	·15
Gilpin & C. C.	1·00	......	......	·16	·17	·16	·22	·23	·27	·26	·30	·20	·22
Gold Coin	1·00	·75	1·00	·80	1·02	·60	1·03	·80	1·03	·85	·95	·88	1·08
Gold Crater	1·00	·50	1·40	·50	1·50	......	......	......	......	......	......	......	......
Gold & Globe	1·00	·15	·05	......	......	·01	·04	·01	·04	......	......	......	......
Gold Eagle	1·00	·15	·30	·10	·50	·15	·38	·21	·40	·30	·40	·25	·50
Gold Fleece	1·00	·50	·60	·45	·54	·28	·51	·21	·50	·22	·39	·18	·22
Independence	1·00	......	......	......	......	......	......	......	......	......	......	......	......
Iowa	1·00	·20	·48	·15	·50	·10	·36	......	......	......	......	......	......
Ironclad	1·00	·03	·04	02	·04	·01	·03	·01	·02	·01	·03	·02	·04
Isabella	1·00	·31	·35	·26	·36	·19	·26	·19	·23	·20	·24	·23	·27
Jack Pot	1·00	·04	·06	·04	·06	·02	·04	·02	·04	·03	·05	·03	·07
Jefferson	1·00	·03	·07	·04	·06	·03	·07	·02	·06	·03	·05	·04	·10
Keystone	1·00	·04	·06	·04	·09	·02	·05	·03	·05	·04	·08	·04	·05
Lillie	5·00	·62	·72	·50	·80	·50	1·00	·64	·88	·60	·96	·75	1·00
Matoa	1·00	·15	·17	·13	·16	·11	·15	·13	·16	·13	·16	·15	·23
Mineral Rock	1·00	·02	·07	·02	·10	·05	·09	·04	·10	......	......	......	......
Mollie Gibson	5·00	·18	·31	·17	·23	·18	·23	·16	·23	·20	·28	·22	·25
Moon-Anchor	1·00	·86	·98	·94	1·03	·93	1·00	·64	·97	·50	·78	·75	·96
Mt. Rosa	1·00	·09	·16	·12	·17	·10	·14	·09	·14	·11	·15	·12	·15
New Haven	1·00	·01	·02	·008	·02	·006	·02	......	......	......	......	·01	......
New Zealand	1·00	·04	·07	·03	·10	·03	·07	·05	·07	·06	·09	......	......
Nugget	1·00	·02	·10	·03	·08	·01	·12	·02	·07	......	......	......	......
Ophir	1·00	·04	·15	·03	·13	·05	·15	·06	·12	·04	·10	·09	·12
Pharmacist	1·00	·06	·08	·06	·07	·04	·07	·04	·07	·04	·06	·05	·06
Portland	1·00	·76	·84	·72	·84	·73	·79	·73	·88	·60	1.01	·99	1·16
Prince Albert	1·00	·01	·03	·01	·03	·005	·04	·006	·03	·01	·05	·01	·03
Sacramento	1·00	·02	·03	·02	·04	·02	·03	·01	·03	·02	·03	·02	·04
Republic	1·00	·02	·06	·02	·04	·01	·04	·02	·03	......	......	·02	·04
Smuggler	1·00	·20	·50	·20	·50	·09	·60	·20	1·00	......	......	......	......
Specimen	1·00	......	......	......	......	·01	·04	·02	·04	·03	·04	·03	·05
Trachyte	1·00	·02	·03	·01	·03	·005	·02	......	......	......	......	·01	·02
Union Gold	1·00	·10	·15	·12	·15	·09	·14	·09	·12	·10	·13	·10	·19
Union Leasing	1·00	·01	·06	·01	·03	·01	·10	·02	·10	......	......	......	......
Virginia Mountain	1·00	·02	·03	·02	·04	·02	·03	·01	·03	·01	·05	·04	·05
Work	1·00	·04	·05	·03	·06	·01	·04	·02	·04	·02	·03	·02	·06

although the gradual advance in demand and prices had given a firmer tone to the copper stocks for some time previous, and the regular dividend payers were at a higher level than most of them had shown since 1892, while the purely speculative stocks generally commanded very moderate prices. These conditions prevailed during the first eight months of the year, prices fluctuating between comparatively narrow limits. Early in September a speculative fever developed and an upward movement began, slowly at first, but acquiring increased momentum. It was most apparent at first in the dividend-paying stocks, but these halted after reaching a high level, and opportunity was given for the purely speculative shares and the wildcats, which naturally soon put into appearance. Shrewd speculators were quick to manipulate the market, which had been given strength by investment buying, and promoters took advantage of the situation to organize new companies on favorable terms, and

FLUCTUATIONS OF MINING STOCKS AT DENVER DURING 1898.—*Continued.*

Name of Company.	July.		August.		September.		October.		November.		December.		Sales.
	B.	A.	B.	A.	B.	A.	B.	A.	B.	A.	B.	A.	
Anaconda	·35	·48	·33	·42	·38	·82	·48	·66	·50	·65	·52	·61	$552,400
Anchoria-Leland	·70	·80	·60	·90	·70	·86	·72	·82	·70	·83	·90	......	39,500
Arcadia Cons	·01	·02	·01	·02	·02	·03	·02	·03	·02	·04	·03	......	69,050
Argentum-Juniata	·20	·25	·19	·23	·18	·26	·22	·25	·21	·39	·34	·45	84,900
Bankers	·04	·08	·04	·07	·06	·08	·06	·07	·05	·07	·06	......	607,000
Banner	......	......	......	......	......	......	......	......	......	......	......	......	18,000
Ben Hur	·02	·03	·01	·04	·03	·07	·03	·04	·04	......	·03	·04	130,100
Big Six	·05	·15	·04	·12	......	......	·05	·06	·04	·06	·04	·07	31,200
Creede & Cripple Creek	......	......	......	......	......	......	......	......	......	......	......	......	14,000
Cripple Creek Cons	·08	·10	·07	·10	·07	·12	·08	·09	·06	·09	·08	·09	308,400
Della S	......	......	......	......	......	......	......	......	·04	·06	......	......	1,000
Elkton Cons	·85	·96	·91	1·20	1·14	1·28	1·00	1·19	·92	1·06	·70	·96	274,691
El Paso	·07	·09	·07	·12	·10	·19	·10	·14	·11	·14	·10	·13	292,480
Enterprise	·03	·05	·04	·07	·04	·07	·04	·05	·04	·05	·03	·06	113,100
Fanny Rawlings	·16	·23	·19	·50	·30	·65	·44	·55	·40	·49	·37	·50	122,300
Findley	·03	·06	·05	·11	·09	·10	·08	·10	·12	·19	·14	·18	112,000
Garfield Cons	·10	·13	......	......	·12	·14	·10	·13	·12	·14	·12	·16	134,325
Gilpin & C. C	·12	·40	......	......	·39	·45	·40	·45	......	......	·43	·46	195,550
Gold Coin	·80	1·00	......	......	1·39	1·51	1·23	1·50	1·35	1·53	1·46	1·64	42,450
Gold Crater	......	......	......	......	......	......			......	......	......	......	......
Gold & Globe	·02	......	·02	·03	·04	·05			·04	·13	·05	·14	7,700
Gold Eagle	·18	·40	·18	·40	......	......			......	......	......	......	16,100
Gold Fleece	·16	·30	·19	·30	·20	·76			......	......	......	......	116,974
Independence	......	......	·62	·78	·60	·76	·49	·72	·52	·61	·58	·62	151,400
Iowa	......	......	......	......	......	......	......	......	......	......	......	......	......
Ironclad	·02	·03	·02	·06	·02	·04	·02	03	·03	·04	·02	·04	201,250
Isabella	·20	·26	·22	·29	·24	·30	·23	27	·25	·30	·26	·32	631,100
Jack Pot	·06	·08	·07	·14	·10	·16	12	·20	·20	·25	·23	·36	401,505
Jefferson	·06	·13	......	......	·04	·08	·03	06	·05	·06	·04	·06	29,200
Keystone	·04	·05	·04	·06	·05	·07	·04	·06	·05	·06	·05	·06	107,100
Lillie	·85	·95	·88	1·25	1·02	1·25	1·00	1·16	......	......	......	......	123,200
Matoa	·21	·33	·19	·23	·28	·43	·24	·32	·28	·36	·28	.33	218,100
Mineral Rock	......	......	......	......	......	......	......		......	......	......	......	2,650
Mollie Gibson	·11	·22	·18	·25	·19	·25	·20	·24	·20	25	·17	·35	54,709
Moon-Anchor	·87	·98	·91	1·20	1·12	1·20	1·10	1·20	1·13	1·25	1·15	1·30	121,950
Mt. Rosa	·11	·15	·10	·18	·16	·20	·16	·20	·17	·25	·17	·23	146,000
New Haven	·01	·02	·02	......	·02	·03	·02		·02	·03	·02	·03	100,000
New Zealand	·04	·08	......	......	·08	·18	·10	·15	·10	·14	·11	·16	75,448
Nugget	·02	·08	......	......	......	......	·05	·08	·01	·05	......	......	30,000
Ophir	·08	·13	......	......	·08	·11	·06	14	·05	·10	·10	......	89,800
Pharmacist	·04	·06	·03	·05	·03	·10	·03	·06	·03	·05	·03	·04	194,600
Portland	1·03	1·18	1·12	1·45	1·40	1·51	1·39	150	1·40	1·48	1·40	1·48	87,780
Prince Albert	·01	·03	·01	·03	·02	·03	·01	·03	·02	·03	·01	·02	74,700
Sacramento	·02	·04	·02	·05	·04	·06	·03	·04	·03	·05	·04	·05	181,600
Republic	......	......	......	......	......	......	......		......	......	......	......	18,000
Smuggler	......	......	......	......	......	......	·20	·35	·25	·28	......	......	2,700
Specimen	·03	·04	·03	·06	·03	·07	·03	·05	·04	·05	·03	·05	96,200
Trachyte	·008	·02	......	......	......	......	......	......	......	......	......	......	1,000
Union Gold	·12	·17	·12	·19	·16	·25	·19	·23	·19	·22	·17	21	320,330
Union Leasing	......	......	......	......	......	......	......	......	......	......	......	......	5,000
Virginia Mountain	·03	·04	·04	·06	·04	·05	·03	·04	·03	·04	·03	·04	136,500
Work	·04	·05	·05	·12	·09	·11	·08	10	·10	·14	·11	·16	209,250
Total sales	......	......	......	......	......	......	......		......	......	......	......	6,968,607

soon found them selling at prices far beyond their wildest expectations. The new companies brought out at first were chiefly to take over properties in the Lake Superior region, but all of any likely, or even possible, merit in that region being soon pre-empted, attention was directed to Arizona and elsewhere, although the latter movement hardly developed itself until early in 1899. Referring to the accompanying table, which includes only the companies that were well organized in 1898, it appears that of the copper companies only the Atlantic, Boston & Montana, Calumet & Hecla, Osceola, Parrot, Quincy, Tamarack and Wolverine have been regular dividend payers in late years. The Arcadian and Baltic are thought to have good prospects, while most of the others are either brand new companies or old companies which have been heretofore unsuccessful.

FLUCTUATIONS IN THE PRICES OF STOCKS AT NEW YORK DURING 1898.

Name and Location of Company.	Par Value.	January.		February.		March.		April.		May.		June.	
		H.	L.	H.	L.	H.	L.	H.	L.	H.	L.	H.	L.
Adams Cons., Colo	$10	......	......	......	......	......	......	......	......	......	......	......	......
Alamo (g.), Colo	1	·04	·03	·04	·02	·03	·02	·03	·02	·02	......	·03	·02
Alice (g.), Mont	25	......	......	1·10	......	·85	......	1·05	·50	·90	......	·80	......
American Flag, Colo	1	......	......	·22	·15	·20	·15	·08	·06	·10	.04	·10	·04
Anaconda (g.), Colo	5	·42	·39	·48	·35	·41	·26	·38	·20	·53	.33	·54	·40
Anchoria-Leland (g.), Colo	1	......	......	·96	·88	·94	·68	·80	·62	·77	.61	·75	·71
Argentum-Juniata (g. s.l.), Nev	2	·22	·18	·21	·19	·22	·16	......	......	·23	.22	·29	......
Belcher (s. g.), Nev	3	·63	·47	·40	......	·48	·25	·30	·13	·23	.15	......	......
Best & Belcher (g. s.), Nev	3	1·15	......	·35	......	·42	·28	·37	·25	·34	.31	·25	·18
Boston & Montana, Mont	25	......	......	......	......	......	......	177·00	17·300	198·00	182·00	203·00	196·00
Breece (s. i.), Colo	25	·30	......	·32	......	·48	·34	·47	·35	......	......	......	......
Brunswick Cons. (g.), Cal	1	·14	·12	·10	......	·15	·10	·15	·11	.22	·13	·16	·12
Bullion-Beck & Champ. (g.s.l.c), Ut	10	·15	·10	·12	......	......	......	......	......	.20	·08	·03	......
Caledonia, S. Dak		......	......	......	......	......	......	......	......	......	......	......	......
Calumet & Hecla (c.), Mich	25	......	......	......	......	......	......	517·00	......	555.00	525·00	......	......
Catalpa (s. l.), Colo	10	·09	......	·10	......	·07	......	·07	......	.11	......	......	......
Chollar (s. g.), Nev	3	·30	·23	......	......	......	......	·34	·30	.15	......	·07	......
Chrysolite (s. l.), Colo	50	......	......	......	......	......	......	......	......	.13	·11	......	......
Comstock Tunnel (s. g.), Nev	100	·04	·03	·04	......	·04	·02	·04	·02	.03	......	......	......
Comstock Tunnel Bonds	100	......	......	·04	......	·03	......	·04	·03	.04	......	·03	......
Comstock Tunnel Scrip	100	......	......	......	......	......	......	......	......	......	......	......	......
Cons. Cal. & Va. (s. g.), Nev	2½	1·45	1·05	1·50	·88	·91	·70	1·05	·64	.73	·49	·55	·30
Cons. Imperial (g. s.), Nev	1	·02	......	·03	·01	·02	......	·02	......	.02	......	·02	......
Creede & Cripple Creek (g.), Colo	1	·04	·02	·04	·03	·04	·03	·09	·03	.08	·03	·11	·04
Crescent, Colo	10	......	......	·05	......	......	......	·04	......	.06	......	·05	......
Cripple Creek Cons. (g.), Colo	1	·11	·09	·12	·10	·11	·08	·10	·07	.10	·08	·10	·09
Crown Point (g. s.), Nev	3	·32	·25	......	......	·25	·10	·24	·10	......	......	·08	·04
Deadwood-Terra (g.), S. Dak	25	1·25	1·00	1·00	·90	1·10	·70	1·10	·65	......	......	......	......
Elkton Cons., Colo	1	·89	·83	·89	·75	·84	·70	·71	·53	.96	·69	·95	·90
Enterprise, Colo	1	......	......	......	......	......	......	......	......	......	......	......	......
Fanny Rawlings, Colo	1	·06	·05	·02	......	......	......	......	......	......	......	......	......
Father de Smet (g.), Dak	100	·20	......	......	......	·23	·16	·23	·14	......	......	......	......
Garfield Cons., Colo	1	·08	·06	·08	·06	·07	·05	·06	·05	.15	·14	·16	·13
Golden Fleece (g. s.), Colo	1	·59	·48	·54	·43	·47	·35	·41	·21	.40	·20	·85	·22
Gould & Curry (g. s.), Nev	3	·30	......	......	......	·25	·15	......	......	.30	·20	·16	·07
Hale & Norcross (s. g.), Nev	3	1·50	1·25	......	......	1·30	1·10	1·10	·55	1.00	·90	·75	·70
Havana	1	......	......	......	......	·76	·75	·76	·75	.76	·75	·76	·75
Homestake (g.), S. Dak	100	46·50	40·00	46·00	41·75	50·	42·	48·00	42·00	50.00	40·00	50·00	47·00
Horn Silver (s. l.), Utah	25	1·20	1·10	1·35	......	2·50	1·30	1·50	1·20	1.40	1·30	1·50	·55
Iron Silver (s. i.), Colo	20	·42	·25	·50	·43	·50	·35	·50	·45	.60	·43	·70	·60
Isabella (g.), Colo	1	·35	·30	·32	·21	·28	·20	·22	·17	.24	·21	·28	·23
Japan (g.), Colo	10	......	......	......	......	......	......	......	......	4.25	......	5·00	2·50
Jefferson (g.), Colo	1	·06	·03½	·05	·04	·06	·04	·06	·02	.05	·03	·06	·04
Justine (g.), Colo	1	·02	......	·03	·01	·07	·05	·05	·01	.05	·03	·04	·02
Kingston & Pembroke (i.), Ont	10	·22	·21	·21	·20	·22	·11	·22	·10	.22	·20	·29	·22
Lacrosse (g), Colo	10	·17	·16	·16	......	·16	·12	·17	·10	......	......	......	......
Lagonda, Colo	1	......	......	......	......	......	......	......	......	.18	·10	·24	·17
Leadville Cons. (g.), Colo	10	......	......	......	......	·09	·08	·13	·07	.12	·09	·12	·10
Lillie, Colo	5	·79	·65	·76	·69	·81	·74	·85	·75	1.00	·81	1·12	·89
Little Chief (s. l.), Colo	50	......	......	·18	·16	·18	·12	·15	·11	.25	·15	·25	......
Mexican (g.s.), Nev	3	·25	·20	·45	·20	·40	·25	·50	·15	.20	·12	·20	·19
Miami, Colo	1	·40	·30	·39	·38	·40	·39	·40	·39	.50	·20	·65	·45
Mollie Gibson (s.), Colo	5	·26	·13	·23	·12	·24	·19	·22	·20	.28	·20	·33	·21
Mt. Rose (g.), Colo	1	·17	·12	·16	·13	·14	·10	·12	·09	.14	·11	·14	·11
Occidental Cons. (g. s.), Nev	3	1·05	......	......	......	......	......	......	......	......	......	·70	......
Ontario (s. l.), Utah	100	4·50	2·50	6·00	3·50	5·00	3·50	5·80	3·00	6.00	2·50	6·00	3·00
Ophir (g. s.), Nev	3	·75	·55	·56	......	·70	·40	·75	·40	.45	·35	·25	......
Osceola, Mich	25	......	......	......	......	......	......	40·00	38·50	47.00	41·00	52·00	46·50
Pharmacist (g.), Colo	1	·09	·05	·08	·07	·10	·06	·07	·06	.07	·05	·07	·06
Phœnix Cons. (g.), Ariz	1	·10	·09	·09	......	·09	·05	·09	·05	......	......	·25	·10
Plymouth (g.), Cal	10	·12	......	......	......	·14	·07	·09	·06	......	......	......	......
Portland (g.), Colo	1	·82	·75	·81	·73	·81	·74	·85	·75	1.05	·85	1·20	·95
Potosé (g. s.), Nev	3	·46	·32	·30	·25	·50	·35	·45	·26	.61	·22	·22	·07
Quicksilver, Cal., com	100	3·50	1·00	3·00	1·25	2·00	1·00	3·50	1·25	3.50	1·00	2·50	1·25
Quicksilver, Cal., pref	100	8·00	3·00	8·00	3·00	7·00	3·00	7·00	2·25	11.00	2·25	8·00	5·00
Quincy (c.), Mich	25	......	......	......	......	......	......	107·00	......	115,00	108·00	115·00	113·00
Rocky Mount. (g.), Colo	1	·17	·14	·17	·08	·15	·10	·09	·08	.17	·05	·14	·02
Savage (g. s.), Nev	2½	·40	·20	......	......	·25	·21	·30	·10	.16	·10	......	......
Sierra Nevada (g. s.), Nev	3	·90	·50	1·10	·74	2·00	·85	2·25	·69	.65	·35	·70	·41
Small Hopes (s. l.), Colo	20	......	......	·75	......	·90	......	1·05	·75	1.05	·80	1·10	......
Standard Cons. (g. s.), Cal	100	1·60	......	1·75	1·60	1·85	1·65	1·80	1·60	1.65	1·55	1·70	1·35
St. Francisco Caríboo (g.), B. C.	5	......	......	2·75	2·63	2·75	2·63	2·75	2·25	2.75	2·25	3·00	2·50
Syndicate (g.), Cal	100	·05	......	......	......	·05	·04	·05	·04	.05	......	......	......
Tamarack (c.), Mich	25	......	......	......	......	......	......	150·00	145·00	160.00	149·00	160·10	157·00
Union (g.), Colo	1	·15	·11	·14	·13	·14	·09	·15	·09	.14	·10	·12	·11
Union Cons. (g. s.), Nev	2½	·46	·25	·40	......	·40	·25	·45	·20	.30	......	·15	·10
Utah Cons. (s.), Nev	3	·10	......	·15	·11	·15	·10	·13	·05	.10	·07	·07	·04
Waldorf (g.), Colo	1	·13	·10	·15	·05	·10	·08	·10	·03	.05	·02	·11	·02
Work (g.), Colo	1	·05	·04	·05	·03	·04	·02	·03	·02	.04	·02	·04	·02
Yellow Jacket (g.s.), Nev	3	·37	......	·37	·25	·33	·24	·30	·10	.27	·20	·26	·17
Yukon, Can	1	·14	·10	·12	......	·14	·10	·18	·10	.14	·11	·14	·10

FLUCTUATIONS IN THE PRICES OF STOCKS AT NEW YORK DURING 1898.—*Continued.*

Name and Location of Company.	July.		August.		Sept.		October.		November		December		Sales.
	H.	L.	H.	L.	H.	L.	H.	L.	H.	L.	H.	L.	
Adams Cons., Colo	.....	.....	.....	.....	·05	.....	·05	.....	.08	·05	·10	·05	$8,800
Alamo (g.), Colo	.....	.....	·04	·03	·06	·03	·05	·03	.06	·04	·06	·05	1,350,600
Alice (g.), Mont	.....	.....	·77	·65	·85	·70	·90	·75	1.20	·85	·90	·75	9,700
American Flag, Colo	.....	.....	·06	.....	·10	·05	·10	·04	.08	·03	.....	.....	118,350
Anaconda (g.), Colo	·43	·38	·41	·32	·80	·40	·68	·45	.65	·48	·64	·50	183,000
Anchoria Leland (g.), Colo	·80	·70	87	·68	·87	·73	·76	·70	.75	·72	.....	.....	85,000
Argentum-Juniata (g. s. l.), Nev	.....	.....	·19	·18	·23	.....	·20	.....	.32	·20	·33	·15	3,950
Belcher (s. g.), Nev	.....	.....	·15	·08	·18	·10	·20	.03	.20	·08	·24	·14	2,900
Best & Belcher (g. s.), Nev	·15	.....	·19	·10	·31	·15	·31	.10	.....	.....	·40	·30	12,100
Boston & Montana, Mont	209·00	203·00	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	65
Breece (s. i.), Colo	.....	.....	·90	·55	1·26	·70	1·10	.80	1·00	·85	·90	·80	4,775
Brunswick Cons. (g.), Cal	·12	·10	·12	·08	·11	·08	·15	.08	·15	·10	·15	·10	24,500
Bullion-Beck & Champ. (g.s.l.c), Ut	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	2,500
Caledonia, S. Dak	.....	.....	.....	.....	·50	.....	·60	.50	·50	.....	·60	·50	....
Calumet & Hecla (c.), Mich	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	630
Catalpa (s. l.), Colo	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	2,110
Chollar (s. g.), Nev	·20	·15	·20	·10	·20	·10	·17	.02	·38	·08	·24	·12	4,600
Chrysolite (s. l.), Colo	.....	.....	.....	.....	·16	·10	·15	.10	·14	·09	·15	·12	2,100
Comstock Tunnel (s. g.), Nev	.....	.....	·02	.....	.....	.....	·04	.03	·04	·03	·04	.....	22,400
Comstock Tunnel Bonds	.....	.....	·02	.....	·04	·02	·04	.03	·04	·03	·04	·03	8,150
Comstock Tunnel Scrip	.....	.....	.....	.....	.....	.....	·05	.03	·05	·03	·04	.....	....
Cons. Cal. & Va. (s. g.), Nev	·42	·25	·44	·28	·92	·32	·83	.60	1·50	·92	1·50	1·00	25,835
Con. Imperial (g. s.), Nev	.....	.....	.....	.....	·02	.....	·02	.....	·03	·02	·03	.....	25,100
Creede & Cripple Creek (g.), Colo.	·04	·03	·04	·02	·08	·04	·09	.08	·08	·04	·06	·04	54,600
Crescent, Colo	.....	.....	·10	.....	·03	.....	.....	.....	·05	.....	·10	·05	4,900
Cripple Creek Cons. (g.), Colo	·10	.....	·10	·07	·12	·06	·09	.06	·10	·04	·10	·04	126,300
Crown Point (g. s.), Nev	·04	.....	·15	·05	·18	·10	·15	.08	·20	·05	·17	·12	5,400
Deadwood-Terra (g.), S. Dak	.....	.....	·35	·25	·40	·25	·45	.29	·45	·35	·50	·38	6,050
Elkton Cons., Colo	.....	.....	1·20	·97	1·26	1·15	1·21	1.00	1·06	·92	1·00	·72	70,400
Enterprise, Colo	·35	.....	.....	.....	.....	.....	·38	.35	·40	·25	·38	·35	900
Fanny Rawlins, Colo	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	14,800
Father de Smet (g.), Dak	.....	.....	.....	.....	·17	·15	·17	.10	·16	·10	·18	·10	650
Garfield Cons., Colo	.....	.....	·15	·11	·13	·12	·13	.11	·15	·12	·17	·12	761,700
Golden Fleece (g. s.), Colo	·26	·17	.....	.....	·40	·25	.....	.....	·35	·24	·27	·22	145,200
Gould & Curry (g. s.), Nev	.....	.....	·17	·07	·30	·10	·20	.10	·25	·07	·25	·15	20,100
Hale & Norcross (s. g.), Nev	·45	.....	·80	·55	·90	·45	1·10	.65	1·20	·75	1·75	·90	4,600
Havana	·76	·75	·80	·75	·76	·75	·76	.75	·76	·75	.....	.....	....
Homestake (g.) S. Dak	53·50	50·00	52·00	45·00	57·00	45·00	60·00	45.00	61·00	60·00	50·00	.....	1,345
Horn Silver (s. l.), Utah	1·35	1·30	1·50	1·15	1·40	1·20	1·20	1.10	1·20	1·10	1·30	1·10	3,975
Iron Silver (s. i.), Colo	·80	·75	·78	·60	·70	·55	·70	.55	·75	·40	·81	·70	23,775
Isabella (g.), Colo	.....	.....	·28	·21	·28	·20	·28	.21	·30	·23	·36	·20	149,400
Japan (g.), Colo	5·00	2·50	5·00	2·50	5·00	2·50	5·00	2.50	5·00	2·50	5·00	2·50	....
Jefferson (g.), Colo	.....	.....	·11	·07	·09	·07	·09	.05	·09	·05	·10	·05	1,045,700
Justine (g.), Colo	·03	·02	·04	·01	·05	·02	·04	.02	·09	·02	·04	·02	2,808,600
Kingston & Pembroke (i.), Ont	·20	·16	·17	·14	·16	·14	·15	.10	·16	·10	·18	·12	5,500
Lacrosse (g.), Colo	.....	.....	.....	.....	·15	·08	·15	.08	·14	·07	·14	·07	15,400
Lagonda, Colo	·25	·20	·45	·20	·35	·32	·50	.30	.....	.....	·40	·36	56,400
Leadville Cons. (g.), Colo	·11	·10	·10	·03	·08	·07	·08	.07	·09	·07	·11	·07	21,800
Lillie, Colo	·90	.....	1·10	·95	1·25	1·06	1·20	1.06	1·25	1·10	.....	.....	111,700
Little Chief (s. l.), Colo	.....	.....	·26	·18	·24	·18	·21	.16	·20	·17	·21	·12	15,634
Mexican (g. s.), Nev	·13	·10	·22	·07	·30	·10	·23	.07	·30	·15	·40	·17	19,300
Miami, Colo	.....	.....	·54	·40	·54	·40	·41	.35	.....	.....	.....	.....	182,700
Mollie Gibson, (s.), Colo	·25	·18	·24	·14	·24	·19	·25	.19	·24	·18	·30	·20	29,300
Mt. Rose (g.), Colo	·13	·10	·25	·10	·21	·13	·21	.16	·20	·16	·25	·17	454,500
Occidental Cons. (g. s.), Nev	.....	.....	·45	·40	·80	·45	·80	.20	·60	·38	1·00	·50	1,000
Ontario (s. l.), Utah	5·00	3·00	4·00	2·50	4·00	3·50	4·25	3.50	5·00	3·50	6·25	5·00	1,189
Ophir (g. s.), Nev	·20	·15	·30	·12	·55	·13	·55	.37	·83	·50	·70	·40	12,750
Osceola, Mich	48·00	46·00	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	120
Pharmacist (g.), Colo	·06	·05	·05	·03	·10	·07	·07	.03	·20	·03	·08	·03	122,400
Phœnix Cons. (g.), Ariz	·14	·10	·20	·13	·18	·11	·19	.08	·17	·11	.....	.....	119,320
Plymouth (g.), Cal	.....	.....	·12	·08	·14	·07	·13	.07	·12	·07	·12	·07	9,000
Portland (g.), Colo	1·18	.....	1·42	1·10	1·50	1·40	1·45	1.35	1·50	1·25	1·50	1·35	146,200
Potosè (g. s.), Nev	·15	·06	·22	·10	·25	·05	·20	.05	·20	·10	·21	·07	22,600
Quicksilver, Cal., com	2·00	1·50	2·00	1·25	2·13	1·25	1·75	1.25	2·00	1·25	3·00	1·25	1,910
Quicksilver, Cal., pref	8·00	5·00	7·00	3·00	6·00	3·50	6·00	3.00	6·50	3·50	11·50	5·00	3,858
Quincy (c.), Mich.	118·00	114·00	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	90
Rocky Mount. (g.), Colo	·14	·06	·16	·10	·20	·12	·18	.03	·11	·02	·05	·01	940,000
Savage (g. s.), Nev	.....	.....	·12	·05	·17	·08	·14	.03	·23	·08	·20	·01	3,500
Sierra Nevada (g. s.), Nev	·52	·02	·60	·45	·96	·55	·77	.55	1·25	·60	1·20	·60	42,500
Small Hopes (s. l.), Colo	1·15	1·05	1·00	·90	1·10	1·00	1·30	1.00	1·30	1·00	1·30	1·00	9,700
Standard Cons. (g. s), Cal	1·60	1·40	1·55	1·40	1·75	1·50	1·85	1.60	1·90	1·75	1·20	2·75	10,958
St. Francisco Cariboo (g.), B. C.	3·00	2·50	3·00	2·50	2·75	2·50	3·00	2.50	3·00	2·00	.....	.....	23,970
Syndicate (g.), Cal	.....	.....	·05	·04	·05	.....	.....	.....	.....	.....	.....	.....	66,800
Tamarack (c.), Mich	158·00	155·00	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	100
Union (g.), Colo	·18	·15	·17	·11	·26	·14	·24	.16	·23	·17	·18	·12	990,200
Union Cons. (g. s.), Nev	.....	.....	·24	·12	.....	.....	·20	.10	·30	·15	·23	·14	15,800
Utah Cons. (s.), Nev	·08	·04	·09	·05	·15	·05	·08	.05	·17	·05	·20	·04	47,800
Waldorf (g.), Colo	·08	·05	·07	·05	·07	·03	·04	.02	·04	·02	·01	.....	509,100
Work (g.), Colo	·14	·08	·12	·14	·12	·07	·10	.06	·15	·07	·15	·09	536,400
Yellow Jacket (g. s.), Nev	·10	.....	·30	·10	·28	·10	·24	.10	·25	·15	·33	·15	120,100
Yukon, Can	·14	·10	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	88,300
Total sales	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	11,902,859

PRICES OF INDUSTRIAL AND COAL STOCKS AT NEW YORK AND PHILADELPHIA DURING 1898.

Name of Company.	Par Value.	January.		February.		March.		April.		May.		June.	
		H.	L.	H.	L.	H.	L.	H.	L.	H.	L.	H.	L.
American Coal	$25	140·00	120·00	140·00	120·00	140·00	100·00	140·00	100·00	140·00	100·00	140·00	100·00
Bethlehem Iron	100	......	......	......	......	......	......	......	......	......	......	......	......
Cambria Iron	50	40·75	38·50	38·50	37·00	39·50	35·75	38·00	36·00	43·88	37·38	45·50	43·00
Cambria Scrip	50	......	......	......	......	......	......	......	......	......	......	......	......
Cambria Steel	50	......	......	......	......	......	......	......	......	......	......	......	......
Central of New Jersey	100	......	......	97·50	89·00	95·00	86·50	94·00	88·00	96·00	93·00	96·00	94·00
Choctaw & Gulf Certificates	50	32·00	28·50	33·00	28·00	31·00	28·50	31·88	29·50	38·50	30·38	39·00	34·13
Choctaw Transfer Certificates	50	......	......	......	......	......	......	......	......	......	......	......	......
Colorado Coal & Iron Dev	100	1·25	1·00	1·00	·25	1·00	·25	1·00	·13	1·00	·13	1·00	·38
Colorado Fuel & Iron	100	26·63	23·00	25·00	20·13	21·00	17·00	19·75	15·00	23·00	20·00	23·75	21·00
Col. & H. C. & Iron	100	7·50	6·00	7·25	5·00	6·50	5·00	6·00	5·00	6·50	5·00	7·00	6·00
Consolidated Coal	100	41·00	38·00	40·00	39·00	41·00	40·00	45·00	36·00	45·00	38·00	46·00	40·00
Delaware & Hudson	100	154·00	110·75	114·50	108·75	110·50	106·00	108·50	104·00	111·88	108·00	112·00	108·00
Federal Steel	100	......	......	......	......	......	......	......	......	......	......	......	......
Federal Steel, preferred	100	......	......	......	......	......	......	......	......	......	......	......	......
General Electric	100	41·00	31·00	39·75	30·50	34·25	29·25	33·25	23·00	37·38	32·25	39·38	36·25
H. & Broad Top	50	17·00	......	15·00	......	15·00	......	......	......	10·00	......	15·00	13·00
H. & Broad Top, preferred	50	43·00	40·50	45·00	42·00	42·75	41·50	41·50	40·50	41·00	23·00	40·50	40·25
Illinois Steel	100	55·75	46·00	55·00	40·00	50·00	44·00	45·50	44·50	59·88	49·75	60·00	54·88
Lehigh Navigation	50	......	......	42·00	24·50	40·38	39·50	40·00	39·00	39·50	38·25	40·00	37·50
Lehigh Valley R.R.	50	......	......	25·00	21·00	23·00	20·50	22·75	20·25	24·00	21·63	23·88	21·00
Maryland Coal, preferred	100	58·00	43·00	55·00	40·00	55·00	44·00	55·00	40·00	55·00	40·00	54·50	47·00
Minnesota Iron	100	60·00	55·00	65·00	59·50	64·00	60·00	63·50	59·00	73·50	63·50	76·50	72·25
National Lead	100	37·75	34·00	36·38	30·00	33·25	26·50	31·00	27·50	33·75	30·00	36·00	33·50
New Central Coal	100	8·13	6·88	8·50	7·00	8·00	6·00	8·00	6·00	8·00	6·25	9·00	7·00
Pennsylvania Coal	100	370·00	330·00	370·00	300·00	340·00	325·00	325·00	300·00	330·00	300·00	340·00	310·00
Pennsylvania R.R.	50	58·50	50·75	60·13	50·13	58·63	55·50	57·50	55·50	58·75	56·00	58·75	57·25
Pennsylvania Salt	50	103·00	......	107·00	105·00	106·50	101·00	101·50	101·00	103·00	101·50	103·00	......
Pennsylvania Steel	50	15·75	12·00	14·00	11·00	10·00	9·00	10·00	9·00	16·25	12·50	18·50	14·50
Pennsylvania Steel, preferred	50	24·50	21·00	23·00	20·00	22·00	18·50	......	......	27·00	20·00	29·00	26·00
Philadelphia & Reading	100	23·00	21·25	23·25	18·00	19·50	15·75	17·00	15·38	19·25	17·00	20·63	18·88
Phila. & Reading 1st pref	100	52·63	48·88	54·13	41·25	46·00	20·50	46·50	37·00	46·00	40·13	49·00	45·13
Standard Oil	100	394·00	294·00	433·00	361·00	388·00	348·00	376·00	346·00	440·00	374·00	443·00	436·00
Tennessee Coal & Iron	100	28·38	23·63	25·88	19·00	22·00	17·00	21·50	18·38	26·50	20·50	26·50	23·00
United Gas Imp	50	106·00	97·75	114·25	104·00	116·25	98·50	106·88	102·50	114·75	99·38	110·88	103·25
Westmoreland Coal	50	50·00	49·50	50·25	50·00	50·50	48·00	50·50	48·00	50·00	49·50	50.00	49.50

FLUCTUATIONS OF MINING STOCKS AT SAN FRANCISCO DURING 1898.

Name and Location of Company.	Par Value.	January.		February.		March.		April.		May.		June.	
		H.	L.	H.	L.	H.	L.	H.	L.	H.	L.	H.	L.
Alpha Cons., Nev	1·00	·08	·05	·11	·02	·15	·03	·09	·01	·03	·02	·02	·01
Alta, Nev	2·00	·15	·10	·19	·04	·17	·11	·26	·16	·18	·09	·10	·02
Andes, Nev	3·00	·16	·10	·15	·10	·13	·08	·17	·06	·09	·07	·08	·02
Belcher, Nev	3·00	·48	·33	·34	·23	·26	·19	·25	·08	·16	·06	·10	·02
Best & Belcher, Nev	3·00	·58	·43	·46	·36	·37	·30	·50	·26	·33	·25	·27	·09
Bullion, Nev	1·00	·10	·03	·10	·06	·12	·07	·08	·01	·07	·05	·05	·01
Caledonia, Nev	3·00	·24	·15	·40	·16	·31	·24	·27	·15	·25	·12	·26	·11
Challenge Cons., Nev	3·00	·38	·15	·38	·24	·30	·25	·32	·10	·24	·07	·23	·12
Chollar, Nev	3·00	·40	·20	·40	·20	·54	·34	·65	·16	·24	·14	·15	·05
Confidence, Nev	3·00	·97	·72	·78	·65	·73	·52	·78	·43	·45	·33	·45	·21
Con. California & Virginia, Nev.	2·50	1·35	·81	1·00	·83	·86	·76	1·00	·57	·66	·45	·48	·14
Cons. Imperial, Nev	1·00	·01	......	·02	·01	.01	......	·01	......	·01	......	·01	......
Cons. New York, Nev	1·00	·02	·01	·93	·01	·02	·01	·02	......	·01	......	·02	......
Crown Point, Nev	3·00	·33	·21	·33	·11	·25	·11	·27	·09	·12	·06	·15	·02
Exchequer, Nev	1·00	......	......	·06	·01	·06	·03	·04	·01	·01	......	·02	·03
Gould & Curry, Nev	3·00	·44	·30	·32	·18	·25	·16	·33	·11	·27	·18	·19	·04
Hale & Norcross, Nev	3·00	1·65	1·20	1·65	1·40	1·50	1·25	1·35	·47	1·00	·45	·95	·41
Julia Cons., Nev	1·00	·01	......	·02	·01	·02	·01	·03	·01	·01	......	......	......
Justice, Nev	2·00	·41	·28	·46	·30	·30	·23	·24	·09	·09	·06	·08	·01
Kentuck Cons., Nev	1·00	·06	·03	·04	·01	·03	·02	·04	·01	·11	·07	·06	·05
Lady Washington, Nev	1·00	·01	......	·03	......	·03	·02	·09	......	·01	......	......	......
Mexican, Nev	3·00	·32	·20	·36	·21	·44	·28	·43	·15	·21	·07	·20	·08
Occidental Cons., Nev	3·00	1·60	1·20	1·55	1·35	2·75	1·55	2·95	2·65	1·10	·35	·68	·55
Ophir, Nev	3·00	·65	·36	·57	·47	·85	·42	·83	·52	·43	·32	·37	·14
Overman, Nev	2·00	·14	·06	·12	·05	·12	·08	·11	·04	·08	·02	·08	·03
Potosi, Nev	3·00	·43	·27	·27	·18	·46	·13	·41	·29	·24	·17	·19	·05
Savage, Nev	2·50	·25	·13	·28	·13	·27	·22	·26	·11	·18	·08	·10	·04
Scorpion, Nev	1·00	·20	·04	·06	·05	·06	·05	·05	·02	·01	......	·01	......
Sierra Nevada, Nev	3·00	1·05	·53	1·05	·70	1·90	·94	2·05	·73	·67	·40	·68	·28
Silver Hill, Nev	1·00	·05	·03	·04	·03	·04	·01	·02	......	·06	·05	·03	·02
Standard Cons., Cal	1·00	1·70	1·30	1·75	1·50	1·75	1·55	1·65	1·45	1·60	1·50	1·65	1·50
Union Cons., Nev	2·50	·47	·21	·45	·34	·45	·32	·48	·22	·19	·07	·19	·12
Utah Cons., Nev	1·00	·12	·07	·12	·09	·13	·08	·13	·03	·07	·04	·06	·02
Yellow Jacket, Nev	3·00	·42	·28	·45	·22	·31	·20	·28	·11	·27	·05	·25	·08

PRICES OF INDUSTRIAL AND COAL STOCKS AT NEW YORK AND PHILADELPHIA DURING 1898.

Name of Company.	July.		August.		September.		October.		November.		December.		Sales.
	H.	L.	H.	L.	H.	L.	H.	L.	H.	L.	H.	L.	
American Coal	140·00	120·00	140·00	110·00	130·00	110·00	125·00	110·00	140·00	120·00	150·00	120·00	....
Bethlehem Iron	......	......	......	......	......	......	......	......	......	......	70·00	69·00	113
Cambria Iron	43·50	41·25	59·00	41·00	48·00	52·25	56·25	51·50	57·00	39·13	42·88	42·00	113,932
Cambria Scrip	......	......	......	......	......	......	......	......	......	......	85·00	84·00	20,650
Cambria Steel	......	......	......	......	......	......	......	......	......	......	13·00	10·88	82,302
Central of New Jersey	94·50	87·00	96·00	89·75	93·50	90·00	91·75	85·25	95·00	85·00	98·50	92·00	257,816
Choctaw & Gulf Ctfs	39·25	38·00	42·50	38·75	42·00	39·75	41·50	39.75	42·00	39·50	43·75	40·63	160,441
Choctaw Transfer Ctfs	......	......	......	......	......	......	......	......	......	......	29·00	27·75	6,731
Colorado C. & I. Dev	·50	·13	·63	·25	·75	·25	·75	·25	1·75	·38	1·25	·50	4,700
Colorado Fuel & Iron	23·00	20·00	26·63	19·50	23·50	20·00	23·00	20·00	28·00	22·00	32·88	22·50	48,456
Col. & H. C. & Iron	5·25	4·00	6·88	4·00	6·50	4·00	5·50	4·00	5·50	4·00	9·75	4·50	21,202
Consolidated Coal	45·00	40·00	41·00	40·00	46·00	......	50·00	46·00	50·00	47·00	50·00	47·00	175
Delaware & Hudson	108·50	103·00	110·00	106·75	108·75	105·75	109·50	......	101·00	93·00	108·00	99·50	147,441
Federal Steel	......	......	......	......	......	......	......	......	33·00	30·00	52·00	31·00	188,838
Federal Steel, preferred	......	......	......	......	......	......	......	......	82·75	71·50	94·00	76·00	764,534
General Electric	40·00	38·63	42·25	39·63	50·13	42·25	48·00	47·00	......	......	......	......	563,989
H. & Broad Top	13·00	......	10·00	.....	10·00	......	......	......	10·13	......	13·00	11·00	845
H. & Broad Top. pf	40·25	.. ...	39·00	38·50	39·00	37·00	38·00	37·50	41·00	......	43·00	41·00	1,411
Illinois Steel	60·00	39·75	73·63	57·50	75·25	64·50	90·75	67·00	78·75	76·25	......	......	131,553
Lehigh Navigation	38·00	37·00	38·00	37·00	39·00	37·50	85·50	37·50	37·50	35·00	38·25	37·00	7,988
Lehigh Valley R.R.	21·75	20·00	21·88	20·00	21·63	20·50	20·75	19·25	20·00	17·50	22·88	20·00	91,985
Maryland Coal, pref	53·00	45·00	53·00	42·00	53·00	45·00	52·00	45·00	52·00	45·00	56·00	47·00	492
Minnesota Iron	74·50	70·75	99·75	72·63	99·75	88·50	137·00	90·00	142·50	135·00	......	......	213,089
National Lead	36·00	33·50	39·63	36·50	37·63	33·50	34·25	31·63	36·00	32·00	39·38	35·25	331,664
New Central Coal	......	......	46·00	35·00	45·00	35·00	44·00	33·00	45·00	35·00	42·00	38·00	4,270
Pennsylvania Coal	340·00	320·00	356·00	325·00	355·00	325·00	340·00	325·00	340·00	320·00	350·00	320·00	24
Pennsylvania R. R	58·88	57·63	60·00	58·63	59·75	58·13	59·00	53·25	59·75	57·25	61·25	59·38	159,599
Pennsylvania Salt	104·25	103·00	104·00	......	104·00	102·00	104·25	102·25	112·00	104·00	......	......	81
Pennsylvania Steel	18·50	15·00	20·00	17·00	20·25	18·00	18·00	10·00	21·75	16·00	22·00	18·50	32,215
Pennsylvania Steel, pref.	29·00	15·00	41·00	30·25	41·00	38·00	......	......	41·50	40·00	42·00	38·00	2,734
Philadelphia & Reading.	19·25	16·00	20·00	17·50	19·13	17·50	18·13	10·13	19·13	15·50	23·13	18·25	379,448
Phila. & Reading, 1st pf.	47·00	38·25	46·00	41·63	46·63	42·00	43·50	38·13	46·13	37·50	51·00	45·00	795,342
Standard Oil	442·00	354·00	426·00	405·00	416·00	403·50	422·00	414·00	422·00	410·00	432·00	413·50	....
Tennessee C. & I	26·13	24·50	34·50	25·13	32·00	27·00	29·50	26·25	32·00	27·88	38·75	29·75	792,435
United Gas Imp	119·50	107·50	123·00	116·25	125·00	119·50	122·00	120·00	123·00	119·00	128·00	123·00	464,086
Westmoreland Coal	50·13	......	50·00	......	50·00	......	49·00	48·00	50·00	48·00	50·00	49·00	500
Total sales	......	......	......	......	......	......	......	......	......	......	......	......	5,989,364

FLUCTUATIONS OF MINING STOCKS AT SAN FRANCISCO DURING 1898.—*Continued.*

Name and Location of Company.	July.		August.		September.		October.		November.		December.	
	H.	L,	H.	L.	H.	L.	H.	L.	H.	L.	H.	L.
Alpha Cons., Nev	·05	·02	·05	·02	·04	·02	·03	·01	·06	·02	·06	·02
Alta, Nev	·11	·02	·15	·07	·11	·05	·07	·02	·14	·04	·13	·07
Andes, Nev	·07	·03	·07	·02	·15	·06	·10	·07	·12	·06	·08	·03
Belcher, Nev	·19	·03	·19	·11	·22	·07	·15	·05	·26	·11	·34	·16
Best & Belcher, Nev	·19	·08	·19	·11	·37	·13	·28	·13	·36	·24	·45	·32
Bullion, Nev	·07	·02	.04	·01	·07	·02	·11	·01	·07	·05	·05	·02
Caledonia, Nev	·22	·09	·24	·15	·30	·18	·28	·18	·34	·22	·23	·17
Challenge Cons., Nev	·21	·07	·15	·12	·26	·14	·17	·10	·23	·15	·19	·14
Chollar, Nev	·21	·04	·23	·13	·20	·13	·17	·07	·25	·11	·30	·12
Confidence, Nev	·48	·18	·49	·35	·53	·19	·49	·25	·73	·49	·80	·55
Cons. California & Virginia, Nev.	·42	·07	·44	·30	·92	·36	·90	·61	1·50	·96	1·40	·98
Cons. Imperial, Nev	......	......	......	......	·01	......	·01	......	·01	......	·01	......
Cons. New York, Nev	......	......	·03	·01	·02	......	·02	·01	·04	......	·02	......
Crown Point, Nev	·11	·02	·15	·06	·20	·10	·16	·06	·22	·09	·19	·14
Exchequer, Nev	......	......	·03	·02	·02	·01	·02	·01	·02	·01	·01	......
Gould & Curry, Nev	·09	·03	·18	·06	·27	·14	·21	·13	·25	·15	·29	·16
Hale & Norcross, Nev	·60	·35	·80	·60	1·15	·45	1·10	·75	1·40	·90	1·88	1·50
Julia Cons., Nev	......	......	......	......	·07	·01	·01	......	·01	......	·01	......
Justice, Nev	·08	·03	·15	·05	·18	·10	·10	·03	·22	·05	·20	·14
Kentuck Cons., Nev	·04	·01	·05	·04	·09	·02	·03	·01	·07	·02	·11	·05
Lady Washington, Nev	·14	·10	·01	......	......	......	......	......	......	......	......	......
Mexican, Nev	·16	·06	·17	·10	·09	·02	·22	·09	·30	·20	·35	·26
Occidental Cons., Nev	·57	·50	·59	·49	·90	·47	·84	·20	·72	·40	1·15	·52
Ophir, Nev	·27	·09	·33	·15	·76	·17	·63	·40	·77	·64	·71	·46
Overman, Nev	·08	·03	·07	·02	·06	·03	·06	·01	·08	·02	·08	·02
Potosi, Nev	·21	·03	·20	·14	·23	·14	·17	·08	·21	·12	·27	·10
Savage, Nev	·14	·07	·13	·09	·22	·10	·12	·05	·25	·07	·18	·10
Scorpion, Nev	·03	·01	·03	·01	·04	·02	·04	·03	·04	·02	·03	·02
Sierra Nevada, Nev	·55	·36	·62	·46	1·15	·56	·81	·61	1·20	·74	1·30	·98
Silver Hill, Nev	·06	·03	·12	·05	·05	......	......	......	·05	......	......	......
Standard Cons., Cal	1·65	1·40	1·65	1·40	1·80	1·50	1·75	1·60	1·95	1·70	2·50	1·75
Union Cons., Nev	·29	·12	·28	·15	·33	·14	·26	·17	·33	·23	·28	·10
Utah Cons., *Nev*	·07	·01	·06	·05	·26	·06	·07	·04	·26	·06	·17	·07
Yellow Jacket, Nev	·34	·08	·33	·14	·26	·14	·20	·10	·25	·16	·23	·15

FLUCTUATIONS OF UTAH MINING SHARES IN 1898.

Companies.	Par Value.	January.		February.		March.		April.		May.		June.	
		4	15	5	19	5	19	2	16	7	21	4	18
Ajax	$10	$ ·51	$ ·40	$ ·37	$ ·27	$ ·35	$ ·33	$ ·34	$ ·36	$ ·35	$ ·33	$ ·30	$ ·30
Alice	25	......	·50	·83	......	·85	......	......	·80	......	......	·85	......
Anchor	10	·70	·70	·60	·60	·60	·50	·50	·50	·50	·52	·50	·25
Bullion-Beck	10	4·90	5·50	5·00	5·00	5·60	6·00	5·90	5·90	5·60	5·65	6·20	5·50
Buckeye	1	·30	·01	·01	·01	·01	·01	·01	·01	·02	·01	·00	......
Centennial Eureka	50	21·25	21·00	19·00	21·00	21·00	22·50	23·00	24·50	25·50	32·00	31·00	31·00
Chloride Point	1	·56	·55	·56	·63	·66	·65	·65	·91	·90	·90	·91	·91
Daisy	3	......	......	......	......	......	......	......	......	......	......	......	......
Dalton	5	·00	·00	·00	·00	·01	·00	......	·00	·00	·00	·00	·00
Dalton & Lark	1	·03	·02	·03	·03	·02	·01	·01	·02	·02	·05	·05	·06
Daly	20	·75	·75	·60	·50	·60	·60	·60	·70	·65	·68	·60	·62
Daly West	20	3·00	3·75	3·00	3·00	3·50	4·00	4·00	4·25	3·75	3·50	3·75	3·00
Dexter	5	1·25	1·35	1·75	1·50	·95	·97	·75	1·20	1·25	1·06	1·15	1·06
Eagle	1	·01	·01	·01	·02	·03	·02	·03	·06	·05	·04	·04	·04
Eagle & Blue Bell	1	......	......	......	......	......	......	......	......	......	......	......	......
Emerald	1	·02	·02	·02	·03	·02	·03	·06	·02	·05	·05	·03	·02
Four Aces	1	·04	·03	·02	·02	·02	·05	·08	·09	·05	·05	·09	·06
Galena	10	·57	·53	·54	·54	·50	·60	·64	·64	·62	·61	·57	·52
Geyser-Marion	5	·92	·94	·92	·89	·89	1·00	1·00	1·02	·90	·89	·86	·77
Golden Eagle	1	......	·07	......	......	·04	·04	·02	......	......	·03	......	......
Grand Central	1	·70	1·05	1·00	2·00	1·50	1·40	1·65	2·00	4·25	4·60	6·75	7·00
Homestake	1	·00	·01	·00	·00	·00	·00	......	·02	·01	·01	·01	·02
Horn Silver	25	1·25	1·25	1·15	1·25	1·25	1·25	1·30	1·30	1·25	1·25	1·25	1·27
Joe Bowers	1	......	......	......	......	......	......	......	......	......	......	......	......
Little Pittsburg	5	·02	·01	·01	·01	·01	·01	·01	0·1	·02	·02	·02	·02
Lower Mammoth	1	......	......	......	......	......	......	......	......	......	......	......	......
Mammoth	5	2·17	2·17	1·75	2·25	2·12	2·12	1·90	1·96	1·95	1·97	1·75	1·90
Mercur	25	8·45	8·45	8·37	8·35	8·45	8·40	8·35	8 10	7·95	8·02	7·95	[illegible]
Northern Light	5	·39	·23	·12	·12	·22	·21	·26	·29	·15	·16	·07	[illegible]
Omaha	2	·26	·25	·22	·20	·22	·20	·17	·25	·25	·28	·25	·25
Ontario	100	2·85	3·20	2·90	3·12	3·15	3·10	3·75	3·25	3·40	3·75	3·85	3·85
Sacramento	5	·22	·27	·28	·29	·31	·44	·46	·45	·73	·72	·70	·65
Silver King	20	15·50	16·00	16·00	16·50	16·87	16·75	16·75	16·75	16·75	17·75	19·25	19·75
South Swansea	1	1·22	1·20	1·25	1·30	1·22	1·15	1·17	1·22	1·17	1·22	1·25	1·30
Star Consolidated	10	......	......	......	......	......	......	......	......	......	......	......	......
Sunbeam	1	·05	·05	·01	·02	·03	·02	·03	·11	·08	·09	·14	·16
Sunshine	10	·28	·16	·20	·20	·20	·20	·25	·27	·30	·27	·25	·21
Swansea	5	1·95	2·02	·07	2·27	2·12	2·07	2·00	1·95	2·05	2·17	2·22	2·25
Utah	10	·61	·61	·54	·54	·54	·54	·55	·55	·50	·54	·50	·47
Tetro	1	......	......	......	......	......	......	......	......	......	......	......	......
Valeo	1	......	......	......	......	......	......	......	......	......	......	......	......

Copper mining stocks monopolized attention in Boston in 1898. The quicksilver stocks, which are now on a steady dividend-paying basis and are held largely for investment, were generally quiet, with few transactions and small changes in price. The gold mining stocks, which are mostly non-dividend payers and purely speculative, did not attract much attention. Among the industrial stocks Dominion Coal gained strength on favorable reports from the mines and the approaching completion of the great works at Revere of the New England Gas & Coke Co., which were to utilize Dominion coal. Since the end of 1898 these works have been put into successful operation.

On the whole the mining stock market in Boston was more active than ever before in its history, and the boom undoubtedly carried values to unwarranted heights in many cases.

THE CLEVELAND IRON MINING STOCK MARKET IN 1898.

With one or two exceptions the iron mining shares closed the year 1898 at higher prices than at the beginning. There was an active demand for these stocks, which was increased by the improvement in all branches of the iron industry and the payment of extra dividends by several mining companies. Minnesota opened at $58 and advanced in February to $65, falling back to $58

FLUCTUATIONS OF UTAH MINING SHARES IN 1898.—*Continued.*

Companies.	July.		August.		September.		October.		November,		December,	
	2	16	6	20	3	17	1	15	5	19	3	17
Ajax	$ ·30	$ ·31	$ ·32	$ ·34	$ ·37	$ ·33	$ ·44	$ ·45	$ ·65	$ ·64	$ ·60	$1·23
Alice	......	......	......	......	·60	·60	·65	......	·75	......	·75	·75
Anchor	·25	·15	·20	·20	·60	·60	·70	·65	·65	·65	·91	·84
Bullion-Beck	5·65	5·75	6·00	6·05	6·05	5·85	5·50	5·00	5·15	5·25	4·55	5·00
Buckeye	·01	·02	·01	·02	·01	·01	·00	·01	·02	·04	·05	·06
Centennial Eureka	32·00	30·00	26·00	26·00	29·00	26·00	26·50	28·00	31·50	38·00	35·50	36·00
Chloride Point	·91	·96	·97	·98	1·00	·97	·98	1·00	1·12	1·10	1·15	1·37
Daisy	......	......	......	......	......	......	·60	·61	·60	·60	·63	·64
Dalton	......	·00	·00	......	·00	·00	·00	·00	......	·01	·01	·01
Dalton & Lark	·09	·08	·03	·03	·04	·03	·05	·04	·04	·05	·06	·08
Daly	·67	·70	·72	·67	·72	·70	·80	·80	·92	·90	1·02	1·00
Daly West	3·50	3·25	3·50	3·75	4·10	4·50	4·50	4·50	4·00	4·35	4·10	4·15
Dexter	1·35	1·22	1·01	1·30	1·29	1·38	1·62	2·49	2·77	3·00	3·00	3·18
Eagle	·05	·04	·03	·02	·03	·05	·07	·07	·11	·10	·08	·06
Eagle & Blue Bell	......	......	......	......	......	......	......	......	......	1·15	1·75	1·75
Emerald	......	·04	·03	·06	·03	·05	·06	·04	·12	·15	·17	·16
Four Aces	.04	·04	·04	·04	·04	·05	·05	·08	·28	·27	·31	·45
Galena	.51	·51	·41	·49	·37	·20	·16	·29	·39	·59	·61	·60
Geyser-Marion	.75	·77	·74	·65	·63	·69	·64	·66	·63	·63	·82	·89
Golden Eagle	.. ...	·01	.. ...	·02	·02	·04	......	·03	·04	·07	·06	·09
Grand Central	6·95	7·00	6·80	6·35	6·80	6·75	6·80	6·70	7·15	7·25	7·92	8·90
Homestake	......	·00	·00	......	......	·00	......	......	·00	·03	·03	·05
Horn Silver	1·15	1·37	1·35	1·35	1·22	1·20	1·20	1·15	1·15	1·15	1·17	1·15
Joe Bowers	......	......	......	·15	......	·04	·04	·06	·23	·24	·21	·29
Little Pittsburg	......	·02	·01	·02	·02	·02	·02	·03	·03	·04	·10	·15
Lower Mammoth	......	......	......	·46	·37	·30	·30	·35	·70	·70	·63	·65
Mammoth	1·82	1·85	1·85	1·98	1·90	1·75	1·66	2·10	2·15	1·95	2·00	1·97
Mercury	6·90	7·10	7·37	7·30	7·27	7·10	7·12	7·00	6·95	7·05	7·50	7·65
Northern Light	·01	·11	·07	·09	·09	·09	·08	·07	·06	·31	·45	·57
Omaha	·25	·30	·29	·43	·43	·52	·50	·08	·56	·57	·55	·54
Ontario	3·25	3·60	3·50	3·90	4·25	4·50	4·50	5·25	5·35	6·00	6·60	6·65
Sacramento	·60	·60	·56	·52	·55	·55	·56	·57	·57	·62	·56	·53
Silver King	19·50	19·50	19·75	19·25	24·00	21·00	21·00	27·50	27·50	28·50	30·00	31·00
South Swansea	1·12	1·15	1·18	1·20	1·29	1·21	1·22	1·26	1·25	1·30	1·22	1·27
Star Consolidated	.....	......	......	......	.. ...	......	......	......	......	·99	1·00	1·41
Sunbeam	·08	·06	·04	·05	·05	·05	·05	·05	·07	·09	·09	·15
Sunshine	·21	·28	·25	·25	·27	·27	·30	·50	·35	·42	56	·45
Swansea	2·42	2·30	2·52	2·25	2·66	2·75	2·72	2·72	3·00	3·50	3·75	3·65
Utah	·47	·51	·67	·60	·68	·50	·56	·56	·70	·70	·85	·95
Tetro	......	......	......	......	......	......	......	......	·03	·06	·06	·06
Valeo	......	·15	·52	·33	·47	·51	·53	47	1·10	1·45	1·30	1·44

NOTE.—This table gives the closing bid quotation on the first and third Saturday of each month, except where Saturday was a holiday, when the following Monday market is given. As near as may be, this affords the best index of the Salt Lake market and generally is but a little under what the shares can be bought for, particularly for the active stocks.

in March. By July it had advanced to $75 and in August jumped to $100, at which figure it closed the year. Republic was steady throughout the year at $9·50@$10·50, closing at the latter figure. Lake Superior, which was offered at $25 early in the season, strengthened in June, and in September bids of $30 failed to bring out any of the stock. Chandler closed the year at $41, or $1 higher than in January. Cleveland-Cliffs opened at $38 and gradually advanced, touching $47 in October; it closed at $45. Pittsburg & Lake Angeline opened at $85, advanced to $100 in April, and held firmly at the latter figure until August, when it rose to $125, which price was maintained until the end of the year.

THE COLORADO MINING STOCK MARKETS IN 1898.

There was a good deal of activity in Colorado Springs, but transactions on the Denver Stock Exchange showed a great shrinkage. Among the most active stocks were Elkton Consolidated, which fluctuated between 51c. and $1·27; [illegible]lla, which ranged from 20c. to 36c.; Union, Work and Moon-Anchor. Most of the stocks dealt in on these exchanges are of Cripple Creek

FLUCTUATIONS OF MINING STOCKS AT LONDON DURING 1898.

Name of Company.	Location.	Shares Issued.	Par Value.	(Divid'nds Paid in 1898.	January-March.	
					H.	L.
			£ s. d.	£ s. d.	£ s. d.	£ s.d.
Alaska-Mexican, g.	Alaska	180,000	1 0 0	1 7¼	1 7 6	1 2 6
Alaska-Treadwell, g.	Alaska	200,000	5 0 0	6 0	5 5 0	5 0 0
Anaconda, c. s.	Montana	1,200,000	5 6 0	10 2	5 15 0	5 18 9
Cariboo Goldfields	British Col.	70,452	1 0 0	....	1 2 6	17 6
Central Chile Copper	Chile	256,267	1 0 0	....	7 6	1 3
Cons. Goldfields	Mexico	183,685	1 0 0	....	1 10 0	7 6
Cons. Gold Mines	California	250,000	1 0 0	....	14 6	7 6
De Lamar, g. s.	Idaho	400,000	1 0 0	6	4 6	2 6
Doric, g.	Colorado	402,007	5 0	....	3 0	2 0
Golden Gate, g.	California	80,000	1 0 0	....	7 6	2 6
Grand Central, g. s.	Mexico	250,000	1 0 0	6 0	2 0 0	1 8 9
Hall Mines (Ords.)	British Col.	250,000	1 0 0	1 0	1 16 3	1 7 6
Lillooet-Fraser R. & Car	British Col.	250,000	1 0 0	....	12 6	7 6
Montana, g. s.	Montana	657,128	1 19 0	3	5 6	3 6
New Elkhorn Prior	Colorado,	87,500	1 0 0	....	10 0	1 3
Plumas-Eureka, g.	California.	140,625	2 0 0	....	3 9	1 3
Richmond, g. s. l.	Nevada	54,000	5 0 0	....	10 0	5 0
Sierra Buttes, g.	California.	122,500	2 0 0	....	3 9	1 3
Colombian Hydr., g.	Colombia	75,000	1 0 0	1 6	11 3	3 9
Copiapo, c.	Chile	100,000	2 0 0	2 0	2 12 6	1 17 6
Frontino & Bolivia, g.	Colombia	128,662	1 0 0	5 6	2 11 3	2 1 9
St. John del Rey, g.	Brazil	425,482	1 0 0	1 0	1 0 6	17 0
Tolima A., s. g.	Colombia	14,000	5 0 0	....	3 15 0	1 15 0
Tolima B., s.g	Colombia	6,000	5 0 0	....	3 0 0	1 5 0
Utah Con., g.	Utah	300,000	1 0 0	....	2 17 6	1 0 0
Libiola, c.	Italy	50,400	5 0 0	5 0	3 2 6	2 0 0
Linares, l.	Spain	15,000	3 0 0	2 6 6	9 0 0	7 0 0
Mason & Barry, c. sul	Portugal	185,172	3 0 0	4 0	3 17 6	2 17 6
Rio Tinto, c.	Spain	325,000	5 0 0	2 0 0	29 1 3	25 0 0
Rio Tinto, Cum. Pref	Spain	325,000	5 0 0	5 0	6 3 9	6 0 0
Tharsis, c.	Spain	625,000	2 0 0	10 0	7 12 6	6 10 0
Assoc. Gold Mines	W. Australia	450,000	1 0 0	2 0	4 11 3	3 13 9
Broken Hill Propr	N. S. Wales	960,000	8 0	6 0	2 3 9	1 18 9
Great Boulder Propr	W. Australia	1,600,000	2 0	2 0	1 5 6	1 0 0
Hannan's Brownhill, g.	W. Australia	110,000	1 0 0	12 6	8 7 6	6 15 0
Harquahala, g.	W. Australia	290,000	1 0 0	....	1 3	9
Ivanhoe Gold Corp.	W. Australia	200,000	5 0 0	....	6 10 0	5 5 0
Kalgoorlie, g.	W. Australia	110,000	1 0 0	....	6 18 9	4 18 9
Lake View Consols, g.	W. Australia	250,000	1 0 0	10 0	11 5 0	8 15 0
Mt. Lyell M. & R., i. c.	Tasmania	325,000	3 0 0	14 0	15 0 0	12 2 6
Mt. Morgan, g.	Queensland	1,000,000	1 17 6	5 11	*4 13 9	*4 1 3
Waihi, g.	New Zealand	320,000	1 0 0	8 0	*4 15 0	*4 5 0
W. Austr. Joint Stock	W. Australia	400,000	1 0 0	....	2 15 0	2 7 6
Champion Reef, g.	Colar Fields	440,000	10 0	10 6	5 10 0	4 17 6
Coromandel, g.	Colar Fields	134,462	1 0 0	4 0	3 11 3	3 1 3
Mysore Gold,	Colar Fields	500,000	10 0	14 0	*5 11 3	*5 2 6
Nundydroog, g.	Colar Fields	220,000	1 0 0	8 4	4 10 0	3 7 6
Ooregum, g.	Colar Fields	145,000	1 0 0	2 6	3 3 9	2 12 6
Ooregum, pfd.	Colar Fields	108,191	1 0 0	4 6	3 18 9	3 7 6
Angelo, g.	Transvaal	273,700	1 0 0	5 0	6 0 0	4 17 6
Bonanza, g.	Transvaal	200,000	1 0 0	1 0 0	4 0 0	3 12 6
British S. Af. Chartered	South Africa	3,746,707	1 0 0	....	3 7 6	2 3 9
Cape Copper, c.	Cape Colony	300,000	2 0 0	9 6	5 5 0	3 18 9
Cape Copper, pfd.	Cape Colony	75,000	2 0 0	9 6	....	....
City & Suburban, g.	Transvaal	340,000	4 0 0	14 0	*6 6 3	*5 2 6
Cons. Deep Level, g.	Transvaal	187,250	1 0 0	....	6 2 6	4 5 0
Crown Reef, g.	Transvaal	120,060	1 0 0	1 18 0	12 5 0	*11 10 0
De Beers Cons., d.	Cape Colony	790,000	5 0 0	2 0 0	30 13 9	27 3 0
Durban Roodepoort, g.	Transvaal	125,000	1 0 0	17 0	6 12 6	5 7 6
Ferreira, g.	Transvaal	90,000	1 0 0	3 0 0	26 17 6	23 10 0
Geldenhuis Deep, g.	Transvaal	300,000	1 0 0	12 0	7 2 6	6 5 0
Geldenhuis Est., g.	Transvaal	200,000	1 0 0	1 2 0	5 1 3	4 2 6
Henry Nourse, g.	Transvaal	125,000	1 0 0	1 10 0	9 15 0	8 10 0
Heriots (new), g.	Transvaal	111,864	1 0 0	1 0 0	8 5 0	6 17 6
Jagersfontein, d.	Orange Fr. St.	200,000	5 0 0	6 0	8 12 6	7 5 0
Johannesburg Con. Invest	South Africa	1,886,188	1 0 0	....	2 3 9	1 7 6
Jubilee	Transvaal	50,000	1 0 0	1 0 0	*9 7 6	*7 0 0
Jumpers, g.	Transvaal	100,000	1 0 0	13 0	5 18 9	4 12 6
Kleinfontein (new), g.	Transvaal	231,250	1 0 0	....	2 18 9	2 0 0
Langlaagte Est., g.	Transvaal	470,000	1 0 0	6 0	4 5 0	3 0 0
May Con. (new), g.	Transvaal	275,000	1 0 0	....	2 7 6	2 0 0
Meyer & Charlton, g.	Transvaal	85,000	1 0 0	11 0	4 18 9	4 3 9
Namaqua, c.	Cape Colony	94,331	2 0 0	6 0	2 13 9	2 5 0
Primrose (new), g.	Transvaal	300,000	0 0	11 0	4 13 9	3 7 6
Rand Mines, g.	South Africa	332,708	6 0	....	34 17 6	27 15 0
Robinson, g.	Transvaal	550,000	5 0 0	15 0	9 0 0	7 17 6
Sheba, g.	Transvaal	1,075,000	1 0 0	2 3	2 12 6	1 18 9
Simmer & Jack, Propr., g.	Transvaal	940,000	5 0 0	3 6	4 2 6	3 1 3
Simmer & Jack, West., g.	Transvaal	300,000	1 0 0	....	....	....
Simmer & Jack, East., g.	Transvaal	600,000	1 0 0	....	3 3 9	2 2 6
Wemmer, g.	Transvaal	80,000	1 0 0	2 5 0	9 7 6	8 2 6
Wolhuter, g.	Transvaal	215,000	4 0 0	14 0	*6 16 3	*5 8 9
Worcester, g.	Transvaal	95,772	1 0 0	9 0	2 15 0	1 17 6

FLUCTUATIONS IN MINING STOCKS AT LONDON DURING 1898.—*Continued.*

Name of Company.	April-June.		July-September.		October-December.		Year.	
	H.	L.	H.	L.	H.	L.	H.	L.
	£ s. d.	£ s. d.	£ s. d.	£ s. d.	£ s. d.	£ s. d.	£ s. d.	£ s.d.
Alaska-Mexican, g	1 7 6	1 0 0	1 10 0	1 5 0	*1 10 0	*1 2 6	1 10 0	1 2 6
Alaska-Treadwell, g	5 0 0	4 10 0	5 5 0	4 18 9	5 2 6	4 10 0	5 5 0	4 10 0
Anaconda, c. s	5 2 6	4 17 6	4 16 3	4 12 6	5 12 6	5 1 3	5 15 0	4 12 6
Cariboo Goldfields	1 0 0	15 0	1 2 6	17 6	1 15 0	1 2 6	1 15 0	17 6
Central Chile Copper	4 0	3 6	4 0	3 6	5 6	4 0	7 6	1 3
Cons. Goldfields	1 0 0	15 0	17 6	12 6	1 0 0	7 6	1 10 0	7 6
Cons. Gold Mines	3 9	3	9	3	2 0	1 3	14 6	3
De Lamar, g. s	3 6	2 6	4 6	3 6	5 6	3 6	5 6	2 6
Doric, g	2 0	1 6	1 9	1 0	1 3	3	3 0	1 0
Golden Gate, g	3 9	1 3	5 0	2 6	5 0	2 6	7 6	1 3
Grand Central, g. s	1 17 6	1 12 6	1 13 9	1 12 6	1 16 3	1 8 9	2 0 0	1 8 9
Hall Mines (Ords.)	1 15 0	1 2 6	17 6	13 9	17 6	11 3	1 16 3	11 3
Lillooet-Fraser R. & Car	7 6	2 6	10 0	3 9	10 0	5 0	12 6	2 6
Montana, g. s	4 6	3 0	4 6	3 0	8 9	3 6	8 9	3 0
New Elkhorn Prior	3 9	1 3	3 9	1 3	6 3	1 3	10 0	1 3
Plumas-Eureka, g	5 0	2 6	5 0	2 6	3 0	6	5 0	6
Richmond, g. s. l	8 9	6 3	11 3	7 6	11 3	8 9	11 3	5 0
Sierra Buttes, g	3 9	1 3	3 9	1 3	2 6	6	3 9	6
Colombian Hydr., g	11 3	7 6	15 0	10 0	16 3	11 3	16 3	3 9
Copiapa, c	2 12 6	2 7 6	2 7 6	2 2 6	2 16 3	2 5 0	2 16 3	1 17 6
Frontino & Bolivia, g	2 13 9	2 8 9	2 7 6	2 5 0	2 6 3	1 15 0	2 13 9	1 15 0
St. John del Rey, g	1 2 6	19 6	1 5 6	1 2 6	1 7 0	1 4 6	1 7 6	17 0
Tolima A., s. g	2 15 0	1 15 0	2 5 0	1 15 0	1 15 0	1 0 0	3 15 0	1 0 0
Tolima B., s. g	2 10 0	1 10 0	2 0 0	1 5 0	1 10 0	15 0	3 0 0	15 0
Utah Con., g	2 17 6	2 2 6	2 15 0	2 5 0	4 7 6	2 7 6	4 7 6	1 0 0
Libiola, c	2 7 6	2 0 0	2 5 0	1 15 0	2 2 6	1 17 6	3 2 6	1 15 0
Linares, l	8 0 0	7 0 0	8 0 0	7 0 0	8 10 0	6 10 0	9 0 0	6 10 0
Mason & Barry, c. sul	4 0 0	3 15 0	3 17 6	3 12 6	4 0 0	3 0 0	4 0 0	2 17 6
Rio Tinto, c	26 7 6	26 0 0	28 17 6	28 12 0	31 0 0	29 10 0	31 0 0	25 0 0
Rio Tinto, Cum. Pref	6 2 6	5 15 0	6 5 0	6 0 0	6 6 3	5 17 6	6 6 3	5 15 0
Tharsis, c	6 12 6	6 2 6	7 0 0	6 10 0	7 12 6	7 0 0	7 12 6	6 10 0
Assoc. Gold Mines	3 18 9	3 8 9	5 0 0	2 5 0	5 1 9	5 3 9	5 13 9	2 5 0
Broken Hill Propr	2 5 0	2 2 6	4 5 6	2 3 9	2 8 9	2 2 6	4 5 6	1 18 9
Great Boulder Propr	1 0 0	18 6	1 0 0	16 0	1 1 3	18 9	1 5 6	16 0
Hannan's Brownhill, g	8 7 6	7 12 6	9 3 9	6 7 6	*8 7 6	*7 5 0	9 3 9	6 7 6
Harquahala, g	1 3	9	1 3	6	1 0	3	1 3	3
Ivanhoe Gold Corp	6 10 0	5 8 9	8 1 3	4 5 0	7 8 9	6 7 6	8 1 3	4 5 0
Kalgoorlie, g	6 5 0	5 6 3	8 5 0	4 2 6	7 2 6	6 2 6	8 5 0	4 2 6
Lake View Consols, g	9 6 3	8 12 6	11 17 6	7 7 6	11 2 6	9 12 6	11 17 6	7 7 6
Mt. Lyell, M. & R., i. c	12 15 0	9 17 6	6 17 6	5 17 6	7 5 0	6 7 6	15 0 0	5 17 6
Mt. Morgan, g	4 11 3	4 5 0	*5 5 0	*3 17 6	5 3 9	*4 17 6	*5 5 0	3 17 6
Waihi, g	*4 15 0	*4 8 9	*4 6 3	*3 15 0	4 11 3	4 1 3	4 15 0	3 15 0
W. Austr. Joint Stock	16 3	11 3	15 0	6 3	10 0	7 6	2 15 0	6 3
Champion Reef, g	4 18 9	4 16 3	5 0 0	4 15 0	4 16 3	4 12 6	5 10 0	4 12 6
Coromandel, g	3 0 0	2 12 6	2 7 6	2 3 9	2 5 0	1 13 9	2 11 3	1 13 9
Mysore Gold	5 10 0	5 0 0	5 12 6	4 17 6	*5 7 6	5 0 0	5 12 6	4 17 6
Nundydroog g	4 0 0	3 17 6	4 5 0	3 18 9	4 1 3	3 13 9	4 10 0	3 7 6
Ooregum, g	3 8 9	3 3 9	3 16 3	3 5 0	3 10 0	3 6 3	3 16 3	2 12 6
Ooregum, pfd	4 1 3	3 13 9	4 5 0	3 13 9	4 1 3	3 5 0	4 5 0	3 7 6
Angelo, g	5 7 6	5 2 6	6 7 6	4 10 0	6 7 6	6 2 6	6 7 6	4 10 0
Bonanza, g	4 5 0	3 16 3	4 18 9	4 6 3	4 17 6	4 12 6	4 18 9	3 12 6
British S. Af. Chartered	2 11 3	2 6 3	3 8 9	2 6 3	3 1 3	2 15 0	3 8 9	2 3 9
Cape Copper, c	5 2 6	4 10 0	4 16 3	4 10 0	5 18 9	4 15 0	5 18 9	3 18 9
Cape Copper, pfd	5 1 3	4 10 0	5 5 0	4 7 6	5 7 6	4 12 6	5 7 6	4 7 6
City & Suburban, g	5 17 6	5 8 9	6 12 6	4 18 9	5 16 3	5 10 0	6 12 6	4 18 9
Cons. Deep Level, g	3 7 6	3 0 0	3 10 0	2 15 0	3 7 6	2 17 6	6 2 6	2 15 0
Crown Reef, g	*2 15 0	*12 10 0	14 12 6	11 5 0	14 5 0	13 15 0	14 5 0	11 5 0
De Beers Cons., d	27 3 9	26 18 9	27 1 3	23 5 0	26 6 3	24 10 0	30 13 9	23 5 0
Durban Roodepoort, g	6 10 0	6 5 0	6 15 0	5 12 6	6 0 0	5 12 6	6 15 0	5 7 6
Ferreira, g	24 15 0	24 5 0	25 0 0	21 15 0	25 0 0	23 15 0	26 17 6	21 15 0
Geldenhuis Deep, g	7 15 0	7 5 0	9 17 6	6 0 0	10 0 0	9 2 6	10 0 0	6 0 0
Geldenhuis Est., g	5 16 3	5 11 3	7 15 0	5 18 9	7 17 6	7 8 9	7 17 6	4 2 6
Henry Nourse, g	9 5 0	8 17 6	10 17 6	8 7 6	9 17 6	9 7 6	10 17 6	8 7 6
Heriots (new), g	7 15 0	7 2 6	7 17 6	7 5 0	7 12 6	7 0 0	8 5 0	6 17 6
Jagersfontein, d	*7 15 0	*6 15 0	8 5 0	7 5 0	9 12 6	8 10 0	9 12 6	7 5 0
Johannesburg Con. Invest	1 10 0	1 3 9	1 7 6	16 3	1 5 0	1 2 6	2 3 9	16 3
Jubilee	*9 10 0	*9 5 0	11 15 0	6 15 0	11 10 0	10 5 0	11 15 0	6 15 0
Jumpers, g	5 10 0	4 15 0	5 15 0	4 7 6	5 10 0	5 0 0	5 18 9	4 7 6
Kleinfontein (new), g	2 10 0	2 3 9	3 0 0	1 12 6	3 1 3	2 12 6	3 1 3	2 0 0
Langlaagte Est., g	3 7 6	2 17 6	*3 10 0	*2 15 0	3 6 3	3 0 0	4 5 0	2 15 0
May Con. (new), g	2 11 3	2 6 3	3 7 6	2 12 6	3 11 3	3 5 0	3 17 3	2 0 0
Meyer & Charlton, g	4 2 6	3 15 0	5 5 0	3 15 0	5 0 0	4 15 0	5 5 0	4 2 6
Namaqua, c	2 15 0	2 11 3	2 17 6	2 11 3	3 3 9	2 15 0	3 3 9	2 5 0
Primrose (new), g	3 15 0	3 6 3	4 16 3	3 1 3	4 11 3	4 5 0	4 16 3	3 1 3
Rand Mines, g	30 5 0	28 17 6	35 0 0	25 12 6	33 0 0	32 5 0	35 0 0	25 12 6
Robinson, g	8 5 0	7 17 6	8 15 0	7 5 0	9 0 0	8 12 6	9 0 0	7 5 0
Sheba, g	1 17 6	1 13 9	1 13 9	1 10 0	1 11 3	1 7 6	2 12 6	1 7 6
Simmer & Jack, Propr., g	3 11 3	3 6 3	4 10 0	2 1 3	5 6 3	4 11 3	5 6 3	2 11 3
Simmer & Jack, West., g	....	....	....	....	3 11 3	2 16 3	3 11 3	2 16 3
Simmer & Jack, East., g	2 15 0	2 7 6	3 2 6	2 0 0	2 16 3	2 13 9	3 3 9	2 0 0
Wemmer, g	*10 2 6*	*9 10 0*	11 7 6	8 0 0	12 0 0	10 7 6	12 0 0	8 0 0
Wolhuter, g	*5 15 0*	*5 5 0*	6 7 6	5 0 0	5 10 0	4 5 0	6 16 3	4 5 0
Worcester	*2 12 6*	2 7 6	3 7 6	2 12 6	3 2 6	2 12 6	3 7 6	1 17 6

NOTE.—g., gold; s., silver; c., copper; sul., sulphur; i., iron; d., diamonds. * Ex-dividend.

gold mining companies. There were moderate transactions in the silver mining stocks, of which Argentum-Juniata and Mollie Gibson are the most important.

THE NEW YORK MINING STOCK MARKET IN 1898.

Transactions on the Consolidated Stock & Petroleum Exchange were insignificant in 1898, there being little demand for the shares dealt in, which are chiefly those quoted on the San Francisco and Colorado mining stock exchanges, by which their prices are governed. On the New York Stock Exchange there were small transactions in Homestake, Quicksilver, Ontario, Horn Silver and Standard, which rank among the best class of mining stocks. Homestake was especially noteworthy for the advance in its price, $61 per share being quoted at the end of the year, and very little of the stock was obtainable even at that high figure. The anticipated speculation in Klondike gold mining companies failed to amount to anything.

Among the industrial stocks there were large transactions in the shares of the coal-carrying railway companies, many of which are miners as well as carriers, but there were few sales of the shares of coal-mining companies proper, such as the American Coal Co., Pennsylvania Coal Co., Maryland Coal Co., etc. Among the iron and steel companies the Cambria Iron Co. and Pennsylvania Steel Co. showed moderate transactions, as did also the Illinois Steel Co., which was succeeded late in the year by the Federal Steel Co., the last becoming one of the highly speculative stocks on the New York Stock Exchange. In the common shares of the National Lead Co., which is now a miner and smelter of lead ores as well the chief producer of white lead, litharge, etc., in the United States, there were large transactions at prices ranging from $27·50 to $40.

THE SAN FRANCISCO STOCK MARKET IN 1898.

During the first six months of 1898 transactions in mining stocks in San Francisco were nearly at a standstill. The managers of the Comstock mines then united in an effort to revive the waning interest of the public by reopening the lower levels and resuming exploration for ore in parts of the mines which had been under water for many years. The execution of the plan was turned over to a number of officials selected from the directors of the leading companies, who organized as the Comstock Pumping Association and entered into a contract with the Risdon Iron Works, which in consideration of $30,000 guaranteed to lower the level of the water from 1,750 ft. to 2,250 ft., power being supplied at a nominal cost by the local water company at Virginia City. The new plant is to be installed at the C. & C. shaft with the object of first exploring an unprospected zone in the old Consolidated California & Virginia mine. The good effect of the drainage proposition was felt in the stock market even before the end of the year, and although prices closed at a comparatively low level, they showed nevertheless a material advance since the pumping contract was signed.

Outside of the pumping proposition the chief events of the year were the reduction of capital stock by the companies and the settlement of the Hale & Norcross litigation. The former was brought about by the war tax, which bore heavily upon the Comstock companies, all of them having been incorporated originally with capitalizations of $10,000,000 and a stamp of $1 being required upon every 100 shares sold. The Comstock shares now have par values of $1 to $3. The Hale & Norcross litigation ended with the payment of $1·50 per share to the stockholders, the balance of the $320,000 judgment recovered being distributed between the lawyers and claimants for expenses and other services, and the mine being turned over to the Fox board of directors, which has held possession of the company's office and books during the last two years.

THE SALT LAKE MINING STOCK MARKET IN 1898.

The Salt Lake City mining stock market was very active in 1898, with sensational fluctuations in the prices of various stocks. Grand Central opened the year at 70c. and attained a maximum of $9·60, the advance being caused by the declaration of a dividend of 12·5c. per share in June and the regular monthly payment of the same amount subsequently. Silver King doubled in price, while Centennial Eureka advanced on the improved prospects, which were borne out by the resumption of dividends in December. Chloride Point, a new dividend payer, also made a big advance. Among the Park City mines, with the exception of Silver King, there was not much activity, the Daly West being idle and the Daly and Ontario being worked on a reduced scale. It is expected that the Daly West mine will be reincorporated, taking in the half of the property belonging hitherto to the Haggin interests, after which operations will be resumed.

DIVIDENDS PAID BY AMERICAN MINES. ($1=$1000; total, full amount.)

Name and Location of Company.	1891.	1892.	1893.	1894.	1895.	1896.	1897.	1898.	Total Paid.
Adams, S., L., Colo	75	8	......	......	6	......	......	......	$693,500
Ætna Con., Q., Cal	......	......	......	20	20	40	40	40	160,000
Alaska-Mexican, G., Alaska	......	......	......	26	80	70	80	72	321,381
Alaska-Treadwell, G., Alaska (*a*)	450	300	375	375	400	350	300	300	3,625,000
Alice, S., G., Mont	75	......	......	......	......	......	80	20	1,075,000
Alliance, S., L., Utah	......	......	......	......	......	......	5	......	5,000
Am. Dev. & Mg., G., S., C., Mont	......	......	......	52	24	......	......	......	76,422
American Gold, G., S., L., Colo	......	......	......	......	......	......	36	54	362,000
American & Nettie, G., Colo	45	30	......	......	......	......	......	......	225,000
Anaconda, G., Colo	......	13	......	......	......	......	......	......	12,500
Anaconda Copper, Mont	......	......	......	......	......	2250	3000	3000	8,250,000
Anchoria-Leland, G., Colo	......	......	......	......	......	30	72	72	174,000
Apollo, Con., G., Cal	......	......	......	......	......	......	100	......	100,000
Argentum-Juniata, S., Colo	......	......	......	......	156	......	......	......	156,000
Argonaut, G., Cal	......	......	......	......	......	......	......	180	180,000
Aspen, S., L., Colo	100	100	100	40	......	......	......	......	900,000
Associated, G., Colo	......	......	......	......	......	......	......	50	50,000
Atlantic, C., Mich	40	......	......	......	......	......	40	40	780,000
Aurora, I., Mich	200	100	200	......	......	50	......	50	750,000
Bald Butte, G., Mont	30	20	100	200	128	32	8	98	642,148
Bangkok-Cora Belle, S., Colo	......	......	......	......	54	6	......	......	106,000
Belden, F., E., M., N. H	......	45	60	60	48	4	......	......	217,000
Big Seven, Cal	......	......	......	......	......	......	3	3	3,000
Big Six, G. S., Colo	......	......	......	......	.....	3	5	7	15,000
Bimetallic, S., G., Mont	840	200	190	......	......	......	......	......	1,630,000
Boreel, S., Colo	......	......	60	23	......	......	......	......	105,000

DIVIDENDS PAID BY AMERICAN MINES.—*Continued.*

Name and Location of Company.	1891.	1892.	1893.	1894.	1895.	1896.	1897.	1898.	Total Paid.
Boston & Montana, C., S., Mont.	500	......	......	275	1050	1500	1800	1950	9,125,000
Breece, I., Colo.	......	......	......	......	......	......	......	10	30,000
Brotherton, I., Mich.	......	40	80	......	......	......	......	......	120,000
Bullion-Beck & Champion, Utah	......	......	......	425	325	290	170	90	2,228,400
Bunker Hill & Sull., S., L., Ida.	......	......	......	......	......	......	102	228	600,000
Caledonia, G., So. Dak.	......	......	......	......	......	......	......	......	192,000
Calumet & Hecla, C., Mich.	2000	2000	2000	1500	2000	2500	5000	5000	56,850,000
Centennial-Eureka, S.,G.,L.,Utah	330	90	188	195	510	390	98	15	2,025,000
Central, C., Mich.	20	......	......	......	......	......	......	......	1,970,000
Central Lead, L., Mo.	......	......	......	......	......	......	28	54	82,000
Champion, G., Cal.	43	41	41	41	27	......	51	26	296,200
Charleston, P., S. C.	......	......	140	......	......	......	10	30	180,000
Church, G., Cal.	......	......	......	......	5	......	......	......	5,000
Cleopatra, G., S., So. Dak.	......	......	450	......	......	......	......	......	450,000
C. O. D., G., Colo.	......	......	......	......	20	5	......	......	25,000
Colorado Central, S., Colo.	14	55	28	......	......	......	......	......	502,603
Colorado Smelting, Colo.	......	......	......	......	......	......	......	250	1,845,000
Commodore, Colo.	......	......	......	......	......	......	26	20	140,000
Con. Tiger & Poorman, Ida. (*b*)	......	......	......	......	......	......	......	20	20,000
Copper Queen, C., S., Ariz	......	140	300	200	150	......	......	......	1,910,000
Coptic, S., Nev. (*d*)	......	......	......	......	1	......	......	......	77,000
Cortez, S., Nev.	250	95	45	......	......	......	......	......	735,000
Crowned King, G., S., L., Ariz.	......	......	......	......	......	......	......	96	208,000
Dalton & Lark, G., S., L., Utah	......	......	......	......	......	88	......	......	87,500
Daly, S., L., Utah	450	450	188	......	......	37	38	......	2,925,000
Doe Run, L., Mo.	......	......	......	......	......	......	30	30	60,000
Deadwood-Terra, G., So. Dak. (*e*)	50	100	......	......	......	100	80	30	1,350,000
De Lamar, S., G., Idaho	150	200	450	500	450	500	......	48	2,298,000
Della S., Colo.	......	......	......	50	......	......	10	......	60,000
Derbec, G., Cal.	20	......	......	......	......	......	......	......	280,000
Dutch, Cal.	......	......	......	......	......	......	8	17	29,000
Elkhorn, S., L., Mont.	300	303	225	142	50	......	......	......	1,265,000
Elkton Cons., G., Colo.	......	......	......	60	......	90	260	220	656,960
El Paso, Colo.	......	......	......	......	......	......	5	7	12,898
Empire State, Idaho	......	......	......	......	......	......	......	62	61,902
Enterprise, S., G., Colo. (*k*)	25	450	125	......	......	......	......	50	900,000
Florence, S., Mont.	......	......	......	......	35	54	54	25	162,500
Forepaugh, Colo.	......	......	......	......	16	......	......	......	16,000
Franklin, C., Mich.	80	160	120	80	......	......	......	......	1,940,000
Galena, S., L., G., Utah	......	......	......	......	6	49	5	......	71,000
Garfield Grouse, G., Colo.	......	......	......	......	......	24	12	......	36,000
Geyser-Marion, G., Utah	......	......	......	......	......	......	63	33	96,000
Gold Coin, G., Colo.	......	......	......	......	15	85	45	......	150,000
Gold Coin of Victor, Colo.	......	......	......	......	......	......	20	120	140,000
Golden Cycle, Colo.	......	......	......	......	......	......	60	60	158,800
Golden Eagle, G., Colo.	......	......	......	......	......	10	......	......	10,000
Golden Fleece, G., Colo.	......	......	......	......	192	162	6	......	569,179
Golden Reward, G., So. Dak.	......	60	60	......	......	......	......	30	155,000
Gold & Globe, G., Colo.	......	......	......	......	......	11	25	......	36,000
Grand Central, G., Utah	......	......	......	......	......	......	......	219	218,750
Granite Mountain, S., G., Mont.	1400	520	......	......	......	......	......	......	12,120,000
Gwin, Cal.	......	......	......	......	......	......	12	......	12,000
Harquahala, G., Ari.	......	......	......	72	......	......	......	......	126,000
Hecla Con., S., L., Mont.	180	90	60	120	90	45	30	......	2,175,000
Helena & Frisco, L., S., Idaho	190	20	......	15	10	50	......	......	475,000
Highland, G., So. Dak.	......	......	......	......	50	240	240	240	3,784,778
Holy Terror, G., So. Dak.	......	......	......	......	......	......	36	81	117,000
Homestake, G., S. Dak.	150	150	150	256	344	375	375	625	7,181,250
Hope of St. Louis, S., Mont.	......	100	175	75	10	60	100	20	762,222
Horn Silver, S., L., Utah	200	200	230	150	50	50	......	80	5,280,000
Idaho, G., Cal.	93	42	105	......	......	......	......	......	5,480,000
Iowa, G., S., L., Colo.	......	......	......	......	......	50	25	10	95,000
Iron Mountain, S., Mont.	25	......	30	50	105	82	5	20	507,500
Isabella, G., Colo.	......	......	......	......	23	180	68	......	270,500
Jackson, G., S., Nev.	5	......	......	......	2	......	......	......	86,000
Jay Hawk, S., G., Mont. (*h*)	......	33	......	......	......	......	......	......	83,375
Kennedy, G., Cal.	360	500	480	540	184	......	......	......	1,796,000
Keystone, G., Cal	......	......	......	......	......	10	......	......	10,000
Lake Superior, I., Mich.	......	400	......	......	84	......	......	252	2,048,000
Last Chance, S., Colo.	......	650	......	......	......	......	......	......	650,000
Leadville, S., L., Colo.	......	12	12	......	......	......	......	......	316,000
Lexington, G., Colo.	......	36	......	......	......	......	......	......	36,000
Lillie, G., Colo. (*s*)	......	......	......	......	......	......	......	50	50,000
Maid of Erin, S., L., C., Colo. (*j*)	......	......	......	......	......	......	......	6	740,000
Mammoth, G., S., C., Utah	320	......	......	......	......	60	......	200	1,350,000
Matoa, G., Colo.	......	......	......	......	......	......	......	25	25,000
Maxfield, S., L., Utah	36	18	......	......	......	......	......	......	117,000
May-Mazeppa, S., L., Colo.	110	......	......	......	......	......	......	......	180,000
Mead, G., Cal.	......	......	......	......	......	......	......	80	80,000
Mercur, G., Utah	......	......	60	175	200	200	250	855	1,241,000
Merrimac, G., Cal.	......	......	......	......	......	......	9	......	9,400
Minnesota, I., Minn.	840	840	495	......	......	495	......	1425	4,735,000
Modoc, G., Colo.	......	......	......	......	......	......	10	90	100,000

DIVIDENDS PAID BY AMERICAN MINES.—*Continued.*

Name and Location of Company.	1891.	1892.	1893.	1894.	1895.	1896.	1897.	1898.	Total Paid.
Mollie Gibson, S., Colo. (*g*)	1000	1700	1280	100	50	......	......	......	$4,080,000
Montana, Ltd., G., S., Mont	83	......	......	......	205	......	......	36	2,925,640
Montana Ore Purchasing, Mont	......	......	......	......	160	320	160	160	800,000
Moon-Anchor, G., Colo	......	......	......	......	......	24	54	150	261,000
Moose, G., Colo	......	......	......	108	72	6	......	......	186,000
Morning Star, S., L., Colo	50	......	......	......	......	......	......	......	1,025,000
Morning Star Drift, G., Cal	23	83	72	106	154	132	142	82	878,600
Moulton, S., Mont	......	30	......	20	80	......	......	......	460,000
Mount Rosa, G., Colo	......	......	......	5	5	20	10	20	60,000
Mountain Copper, Cal	......	......	......	......	......	......	31	62	93,750
Napa Cons., Q., Cal	40	70	70	50	80	70	80	80	970,000
New Elkhorn, G., Colo	......	......	......	......	......	72	......	......	72,000
New Guston, S., Colo	440	124	......	......	......	......	......	......	1,198,120
New Idria, Q., Cal	......	......	......	......	......	......	10	70	100,000
N.Y. & Hond. Rosario, S., G., C.A	......	......	......	......	......	......	165	165	975,000
North Banner Con., G., Cal	20	......	......	......	......	......	......	......	20,000
North Star, G., Cal	50	50	100	......	......	......	......	50	500,000
Nugget, G., Colo	......	......	......	5	5	......	......	20	30,000
Omaha, G., Cal	......	7	43	43	......	......	......	......	106,100
Ontario, S., Utah	900	750	......	......	......	180	203	......	31,557,500
Osceola, C., Mich	150	150	100	......	100	125	150	300	2,522,500
Pamlico, G., Nev	......	12	......	......	......	......	......	......	189,080
Pandora, G., Mont	3	3	......	......	......	......	......	......	6,000
Parrot, C. S., Mont	360	216	138	67	......	......	......	483	2,207,898
Pennsylvania Con., G., Cal. (*i*)	......	......	......	......	......	......	18	28	51,350
Petro, S., Utah	18	......	......	......	......	......	......	......	17,500
Pharmacist, G., Colo	......	36	44	......	......	......	......	......	80,000
Pioneer, G., Cal	......	......	......	......	......	......	......	50	50,000
Pleasant Valley, C., Utah	......	......	20	......	......	......	......	......	593,056
Plumas Eureka, G., Cal	70	25	53	......	......	......	......	......	2,696,294
Portland, G., Colo	......	......	......	67	556	240	330	570	2,227,080
Princess, G., Colo	......	......	......	......	......	......	45	......	45,000
Quicksilver, Q., Cal. (*l*)	118	......	......	......	......	......	......	......	2,475,082
Quincy, C., Mich	400	350	300	400	600	1000	800	650	10,120,000
Raven, G., Colo	......	......	......	......	......	......	......	15	15,000
Red Cloud, S. L., Idaho	20	80	70	10	......	......	......	......	180,000
Republic, G., Wash	......	......	......	......	......	......	......	120	120,000
Rescue, G., N. M	......	12	......	......	......	......	......	......	12,000
Rialto, G., Colo	32	18	......	......	......	......	......	......	50,250
Richmond Con., S., L., Nev	34	......	14	......	......	......	......	......	4,386,780
Rico-Aspen, Colo	......	......	50	300	......	......	......	......	350,000
Running Lode, G., Colo	15	6	1	......	......	......	......	......	27,000
Sacramento, G., S., L., Utah	......	......	......	......	......	5	15	40	57,000
Saint Joseph, L., Mo	150	150	150	150	150	......	144	113	2,784,500
Santa Rosalia, G., Cal	......	......	......	......	......	......	20	......	20,000
Sheridan, S., G., Colo	75	......	......	......	......	......	......	......	300,000
Sierra Buttes, G., Cal	25	36	31	......	......	......	......	......	1,584,933
Silver King, G., S., L., Utah (*p*)	......	......	......	......	263	375	450	450	1,800,000
Small Hopes, S., Colo	......	38	25	......	......	25	......	25	3,300,000
Smuggler, S., L., Z., Colo	......	......	......	......	......	......	......	120	1,095,000
Smuggler Union, G., S., Colo. (*m*)	......	......	......	......	......	150	......	......	150,000
South Eureka, G., Cal	......	......	......	......	......	......	......	12	12,000
South Swansea, S., L., Utah	......	......	......	......	......	......	68	38	112,460
Standard Con., G., Cal. (*n*)	10	40	20	40	20	......	40	27	5,674,940
Strong, G., Colo	......	......	......	......	......	......	145	300	445,000
Swansea, G., S., L., Colo	......	......	......	2	......	......	......	......	39,000
Swansea, Utah	......	......	......	......	1	20	50	65	136,500
Tamarack, C., Mich	600	600	600	400	400	300	360	440	5,570,000
Temonj, G., Colo	......	......	......	......	10	......	......	......	10,000
Tomboy, G., Colo	......	......	......	......	300	300	......	50	660,000
Trinity River, G., Colo	......	......	15	......	......	......	......	......	15,000
Union, G., Colo	......	......	......	27	13	23	......	......	73,900
United Verde, C., S., Ariz	......	20	435	......	......	......	......	......	562,500
Utah, Utah	......	15	5	......	17	22	4	......	117,000
Victor, G., Colo	......	......	120	105	240	240	90	350	1,155,000
Victor, L. & M., Colo	......	......	......	......	24	12	......	......	42,000
Vindicator, G., Colo	......	......	......	......	......	......	......	138	137,875
Wolverine, C., Mich. (*q*)	......	......	......	......	......	......	......	60	60,000
W. Y. O. D., G., Cal. (*r*)	......	......	......	24	24	......	......	......	48,000
Yankee Girl, S., Colo. (*o*)	260	......	......	......	......	......	......	......	520,000
Yellow Aster, G., Cal	......	......	......	......	......	......	40	129	168,789
Yosemite, S., Utah	......	......	......	......	......	......	5	......	5,000

(*G*) Gold; (*S*) Silver; (*L*) Lead; (*I*) Iron; (*C*) Copper; (*Q*) Quicksilver; (*Z*) Zinc; (*M*) Mica.

(*a*) Formerly the Alaska Mining and Milling Company, reorganized in 1891 as the Alaska-Treadwell Gold Mining Company; the dividends credited for 1891 and 1892 are the payments of the latter company. The Alaska Mining and Milling Company paid $700,000 previously.

(*b*) Consolidation of Cœur d'Alene and Tiger formed in 1895. The Cœur d'Alene Silver Lead Mining Company paid $375,000 in dividends; and the Tiger about $250,000.

(*d*) Formerly the Young America South Mining Company, reorganized as the Coptis in 1891.

(*e*) Previous to consolidation the Deadwood paid $275,000 and the Terra $75,000.

(*g*) Mollie Gibson levied assessments to 1892 amounting to $20,000.

(*h*) Jay Hawk & Lone Pine Consolidated Mining Company, Limited.

ASSESSMENTS LEVIED BY MINING COMPANIES.

Name and Location of Company.	1892.	1893.	1894.	1895.	1896.	1897.	1898.	Total Levied.
Abbie, Cal	....	....	....	....	....	....	$3,500	$3,500
Ada Cons., Utah	....	$330	....	$3,000	....	....	....	3,330
Alamo, Utah	....	....	....	....	....	$625	313	937
Alhambra, Nev	....	....	....	....	....	....	100,000	100,000
Alliance, Utah	....	....	....	50,000	....	....	....	200,000
Allouez, Mich	....	....	$16,000	....	....	....	80,000	1,520,987
Alpha, Nev	$36,750	10,000	6,000	15,750	$15,750	15,750	8,400	316,050
Alta, Nev	27,000	25,200	50,400	20,160	32,400	25,200	16,200	3,664,910
American Quartz, Cal	....	....	....	....	....	1,000	....	1,000
Anchor, Utah	....	90.000	....	....	....	....	....	560,000
Andes, Nev	25,000	....	25,000	15,000	15,000	20,000	10,000	1,195,000
Arrastraville, Cal	....	....	....	....	....	....	10,000	10,000
Arnold, Mich	....	....	....	....	....	....	180,000	180,000
Baliol, Cal	....	....	....	....	....	25,000	30,000	55,000
Banner, Idaho	....	....	....	....	....	10,000	....	10,000
Beaver Creek, Mont	....	....	....	....	....	4,000	....	4,000
Belcher, Nev	78,000	....	50,200	50,200	50,200	57,200	20,800	3,514,800
Bellefontaine, Cal	....	....	....	....	....	....	10,000	10,000
Belle Isle, Nev	30,000	20,000	....	....	10,000	....	....	240,271
Best & Belcher, Nev	50,000	75,600	50,400	50,400	25,200	75,600	30,240	2,589,643
Bogan Silver, Utah	....	....	....	....	....	8,125	....	26,875
Boston & Cripple Creek, (g.), Col	....	....	....	....	....	....	20	20,000
Boulder, g. Cal	....	....	....	....	....	....	10,000	10,000
Brunswick Cons., Cal	20,000	20.000	10,000	20,000	15,000	300,000	....	160,000
Buchanan, Cal	....	....	....	....	....	....	25,000	25,000
Buckeye, Utah	....	....	....	....	3,750	5,000	6,250	15,000
Bullion, Nev	100,000	25,000	30,000	40,000	20,000	20,000	15,000	3,120,000
Burlington, Cal	....	....	....	....	3,000	....	....	3,000
Butte & Boston, Mont	....	....	....	1,500	1,500	....	....	3,000
Butte-Queen, Cal	4,000	....	....	....	....	....	....	16,000
Butte & Phila., Mont	....	....	....	....	....	1,000	....	1,000
Caledonia Silver, Nev	....	....	50,000	....	5,000	....	....	3,240,000
California, Cal	6,000	....	....	4,500	....	1,000	....	17,500
Cedar Creek, Cal	....	....	....	....	....	....	2,000	2,000
Centennial, Mich	....	....	....	....	....	120,000	300,000	460,000
Central Eureka, Cal	....	....	....	....	....	20.000	22,000	78,000
Central G. & S., Cal	....	....	....	....	....	2,250	....	7,500
Central North Star, Cal	....	10.000	....	....	....	....	....	10,000
Challenge Cons., Nev	45,200	15,000	50,000	5,000	7,500	5,000	10,000	427,500
Channel Bend, Cal	....	....	....	....	....	....	2,000	2,000
Chollar, Nev	112,000	61,600	56,000	56,000	28,000	44,800	50,400	2,048,600
Church, (g.) Cal	....	....	....	....	....	....	3,000	3,000
Cinnabar King, Cal	....	....	....	....	....	....	1,000	1,000
Cleveland, (g.) Utah	....	....	....	....	....	....	2,500	2,500
Columbia, Utah	....	....	....	....	....	....	750	750
Commonwealth, Nev	30,000	....	....	....	....	....	....	190,000
Comstock, Utah	....	....	....	....	25,000	....	....	25,000
Confidence, Nev	49,920	....	6,240	14,976	7,488	14,976	13,728	540,126
Cons. St. Gothard, Cal	....	....	....	....	....	....	35,000	35,000
Cons. Cal. & Va., Nev. (*a*)	108,000	6,220	108,000	54,000	118,800	216,000	108,000	678,000
Cons. Imperial, Nev	26,500	....	....	5,000	500	10,000	10,000	2,246,000
Cons. New York, Nev	20,000	25,000	5,000	10,000	5,000	60,000	30,000	160,500
Crown Point, Nev	100,000	80,000	65,000	25,000	70,000	....	25,000	2,970,000
Dalton, Utah	....	3,750	5,000	5,000	5,000	2,500	7,500	53,750
Derbec Blue Gravel, Cal	10,000	5,000	....	....	....	....	....	15,000
Dexter, Nev. (*b*)	....	....	....	....	....	....	30,000	38,000
Eagle, Cal	....	....	....	....	....	5,000	....	5,000
Eagle, Ore	....	....	....	....	....	3,000	3,000	6,000
East Best & Belcher Nev	20,000	....	20,000	....	....	....	....	85,000
East Golden Gate, Utah	....	....	....	....	....	3,000	....	3,000
East Sierra, Nev	....	....	5,000	....	....	....	....	25,000
Emerald, (g.) Utah	....	....	....	....	....	....	3,000	3,000
Eureka Cons., Nev. (*c*)	....	....	....	12,500	....	....	10,000	585,000
Eureka Cons., Utah	....	....	....	....	5,000	....	....	5,000
Eureka Con. Drift, Cal	....	....	....	....	....	75,000	5,000	172,500
Excelsior, Cal	....	....	....	....	....	....	667	667
Exchequer, Nev	45,000	15.000	10,000	5,000	5,000	5,000	....	730,000
Fall River, Cal	....	....	....	....	....	....	5,000	5,000
Far West G. & S., S. Dak	....	....	....	....	....	750	....	42,125
Forlorn Hope, Cal	....	....	....	....	....	....	20,000	25,000
Four Aces, Utah	....	....	....	....	....	2,500	2,500	5,000
Galata, Cal	....	....	....	....	....	....	10,000	10,000
Galena, (g.) Utah	....	....	....	....	....	....	10,000	10,000
Geyser, (s. l.) Colo	....	....	....	....	....	....	50,000	1,075,000
Gold Belt, Utah	....	1,345	....	....	1,667	....	....	3,012
Gold Flat, Cal	11,000	20,000	....	....	....	....	....	13,000
Gold Ridge, Ore	....	....	....	....	....	4,000	....	4,000
Golden Fleece, Cal	....	....	....	....	....	260	....	56,260
Golden King, Cal	....	....	....	....	....	....	3,500	3,500
Good Title, Cal	....	....	....	....	....	....	20,000	20,000
Goodyear, Mont	2,000	4,183	....	....	....	....	....	17,183
Gould & Curry, Nev	76,400	80.200	32,400	48,600	48,600	59,400	37,800	4,614,350
Gray Eagle, Cal	....	....	....	....	....	10,000	....	10,000

ASSESSMENTS LEVIED BY MINING COMPANIES—*Continued.*

Name and Location of Company.	1892.	1893.	1894.	1895.	1896.	1897.	1898.	Total. Levied.
Great Eastern, Utah							1,500	$1,500
Great Western Quicksilver, Cal. (*d*)							5,000	5,000
Grizzly, Cal							15,000	15,000
Hale & Norcross, Nev. (*e*)	168,000	56,000	56,000	39,200	61,600	39,200	11,200	5,688,280
Hartery Cons., Cal			2,000	2,000	2,000			33,000
Hidden Treasure, Cal		1,000						1,000
Hilda, Cal							2,000	2,000
Himalaya, Utah	1,800							10,000
Home, Cal							3,000	3,000
Homestake, Utah						4,000	4,000	4,000
Honorine, Utah								50,000
Horsefly, Cal							8,000	8,000
Horseshoe Bar Cons., Cal						24,000	6,000	85,800
Hudson Bay, Cal		10,000						10,000
Independence, Nev	5,000							345,000
Jack Rabbit, Cal	15,000	13,000	5,000					118,000
Jackson, Nev		10,000						247,500
Joe Bowers, Utah							4,000	4,000
Julia Cons., Nev		5,000		5,500	5,500	5,500		1,495,500
Jumbo, Utah							500	500
Junction, Cal							10,500	10,500
Jupiter Gravel, Cal						40,000	40,000	80,000
Justice, Nev	42,000	30,000	75,000	20,000	15,750		10,500	3,646,750
Karan, Cal							2,500	2,500
Kate Hayes, Cal							10,000	10,000
Kentuck, (g.) Utah							30,000	30,000
Kentuck Cons., Nev.	31,500	10,500	26,250	10,500	5,250	10,500	5,250	125,300
Lady Washington, Nev					5,400			157,800
Laird, Cal							10,000	10,000
Larkin, Cal							2,000	2,000
La Suerte, Cal							7,500	7,500
Leo, Mont					10,000			52,500
Leon, Cal							1,500	1,500
Little Pittsburg, Utah				4,000	6,000	4,000	4,000	18,000
Live Oak Con., Cal							20,000	20,000
Live Oak & Minett, Cal							1,500	1,500
Live Yankee, Cal							5,000	5,000
Lone Hill, Cal						1,125		1,125
Lone Star Cons., Cal,	5,000							12,500
Lower Mammoth, Utah							15,000	15,000
Lucky Bill, Utah					7,200	6,600	1,800	56,400
Marguerite, Cal						20,000	15,000	70,000
Martin White, Nev	50,000	25,000	25,000					1,350,000
Maxfield, Utah							6,000	6,000
May Day, Cal							5,000	5,000
May Day, Utah							2,000	2,000
Mayflower Gravel, Cal. (*k*)							6,000	6,000
Merrimac, Cal							20,000	20,000
Meteor, Utah							3,734	3,734
Mexican, Nev	75,600	76,500	75,600	75,600	40,320	20,000	40,320	2,258,720
Mineral Hill, Cal						2,500		2,500
Modoc Chief, Idaho	4,375							4,375
Montecito, Cal							10,000	10,000
Montreal, Utah	750			375		375	375	5,625
Morgan, Cal							10,000	10,000
Mount Diablo, Nev. (*g*)					7,500			145,000
Mountaineer, Cal							15,000	15,000
Nancy Hanks, Cal							5,000	5,000
National Con., Cal							15,000	15,000
Nashville, (g.) Cal							2,000	2,000
Navajo, Nev	20,000	20,000						555,521
Nevada Queen, Nev.	25,000	25,000		5,000				270,000
North Banner, Cal					10,000			21,794
North Belle Isle, Nev. (*h*)	20,000	38,075			10,000			523,075
North Bloomfield, Cal							25,000	25,000
North Commonwealth, Nev. (*i*)		10,000						120,000
North Eureka, Utah					1,500			1,500
North Gould & Curry, Nev	10,000	10,000	20,000		10,000		20,000	375,000
North Mercur, Utah							10,000	10,000
Northern Light, (g.) Utah							80,000	80,000
Northern Spy, Utah							10,000	10,000
Occidental Cons., Nev	50,000	55,000	30,000	30,000	65,000	30,000	20,000	499,179
Old Home, Cal							5,000	5,000
Omaha Con., Cal							10,000	10,000
Opohonga, Utah							1,500	1,500
Ophir, Nev	50,000	100,000	100,000	100,000	25,200	25,200	43,200	4,602,568
Original Keystone, Nev	10,000							250,000
Orleans, Cal							15,000	15,000
Osceola Cons., Cal						1,000	1,000	10,924
Overman, Nev. (*j*)	126,720	60,000	34,560	23,040	34,560	17,280	17,280	4,129,690
Peer, Ariz	20,000	10,000	5,000					215,000
Peerless, Nev	5,000	5,000	5,000					410,000
Phœnix, Utah						1,000		1,000

ASSESSMENTS LEVIED BY MINING COMPANIES—*Continued.*

Name and Location of Company.	1892.	1893.	1894.	1895.	1896.	1897.	1898.	Total Levied.
Pine Hill, Cal.		3,000	3,000	5,000	5,000	10,000		$30,000
Potosi, Nev.	56,000	84,000	112,000	56,000	28,000	50,400	50,400	2,157,200
Rainbow, S. Dak.	1,250			438	425	100		5,213
Red Cap, Cal.							90,000	90,000
Reddick, Cal.							5,000	5,000
Rescue Gold, Nev.						3,000	2,000	5,000
Reward, Cal.						7,040	3,840	63,680
Richmond, Cal.							10,000	19,000
Ridge Copper, Mich.						50,000		50,000
Rockland, Cal.							3,000	3,000
Ropes, Mich.		20,000						20,000
Rose Creek, Cal.							5,000	5,000
St. Mary's Copper, Mich.		2,000		2,000				4,000
Salt Lake & Nevada, Utah							1,000	1,000
Savage, Nev.	122,000	112,000	100,805	67,200	67,200	67,200	44,800	7,298,600
Scorpion, Nev.	5,000		5,000			5,000		420,000
Seg. Belch. & Mides, Nev.	25,000	35,000	20,000	20,000	10,000	10,000	8,000	368,000
Sevier, Utah						22,500		50,000
Sheep Rock, Utah						5,250		5,250
Sierra Nevada, Nev.	55,000	45,000	50,000	50,000	50,000	400,000	20,000	6,706,910
Silver Bell, Mont.							5,000	5,000
Silver Hill, Nev.	16,200	5,400	5,400			5,000	5,400	2,220,200
Silver King, Ariz.	25,000		100,000		75,000	50,000	50,000	440,000
Silver State, Utah						1,000		1,000
Siskiyou Cons., Cal.	7,000	9,000	14,000	4,000	2,000		3,000	3,000
Snowflake, Utah							2,000	46,000
Soulsby Con., Utah						13,125		17,500
South Fork Con., Utah							5,000	5,000
Spanish Bar, Cal.							2,000	2,000
Star, (g. s.) Utah							1,000	1,000
Sterling, Cal.							15,000	15,000
Sunbeam Con., Utah						11,250	5,000	23,125
Telegraph, Cal.	975							3,575
Tetro, Utah							18,000	18,000
Thorpe, Cal.							5,000	5,000
Trinity, Cal.							1,000	1,000
Troy, Alaska						2,000	6,250	10,500
Union Cons., Nev.	50,000	45,000	35,000	20,000	40,000	20,000	30,000	2,630,000
Utah Cons., Nev.		25,000	5,000	5,000	10,000	30,000	20,000	470,000
Vallejo, (q). Cal.							2,000	2,000
Victory, S. Dak.					375			2,025
Wall Street, Mont.	900							1,500
Waterloo, Cal.		30,000						30,000
Weldon, Ariz.	5,000	10,000						65,000
Wood River, Idaho	3,000							3,000
Yellow Jacket, Nev.	156,000	90,000	90,000	90,000	90,000		24,000	5,788,000
Yuba River, Cal.							2,500	2,500
Zuba, Cal.							5,000	5,000

NOTE—All the Comstock companies reduced their capitalization between September and December, 1898.

(*a*) Previous to the consolidation in August, 1884, the California had paid $31,320,000 in dividends and the Consolidated Virginia $42,390,000. Since then, and to February, 1895, the consolidated company paid $3,894,000 in dividends.

(*b*) Dexter paid $100,000 in dividends to 1893.

(*c*) Eureka Con. paid $5,112,500 in dividends to 1892.

(*d*) Great Western paid $288,366 in dividends to 1893.

(*e*) Hale & Norcross paid $5,776,200 in dividends to 1898.

(*g*) Mt. Diablo paid $225,000 in dividends to 1893.

(*h*) North Belle Isle paid $230,000 in dividends to 1888.

(*i*) North Commonwealth paid $25,000 in dividends to 1891.

(*j*) Overman reincorporated in September, 1898.

(*k*) Mayflower paid $166,824 in dividends to December, 1895.

(*i*) Pennsylvania Cons. levied assessments to 1892 amounting to $50,051.

(*j*) Maid of Erin Silver Mines, Limited, formerly Henriette & Maid Consolidated Mining Company. The dividends for 1887, 1888, 1889, and 1890 were paid by the old company, and those for 1891 and 1892 by the new company.

(*k*) Enterprise Mining Company, of Rico, Colo.; there is also an Enterprise Mining Company in Aspen and one in Leadville.

(*l*) Including dividends paid on preferred stock and common stock.

(*m*) Previous to consolidation in 1896 the Smuggler Mining Company had paid $1,140,000 in dividends.

(*n*) Bodie, Bulwer, and Mono transferred to Standard Con., January, 1897. Previous to consolidation Bodie paid $1,677,572, Bulwer paid $100,000, and Mono $12,500, while Standard Con. levied assessments to 1890, amounting to $99,887.

(*o*) Yankee Girl Silver Mines, Limited, formerly Yankee Girl Mining Company. The above statement includes the payments by both the old and new companies.

(*p*) Silver King levied assessments to 1897 amounting to $3,000.

(*q*) Wolverine levied assessments to 1895 amounting to $180,000.

(*r*) W. Y. O. D. levied assessments to January, 1890, amounting to $22,500.

(*s*) Lillie transferred to an English company in October, 1898; old company paid $134,110 in dividends to September 1, 1898.

AUSTRALASIA.

The most important articles of mineral production in the seven colonies of Australasia are gold, silver, lead, copper, tin, iron and coal. These industries are referred to specifically under the respective captions elsewhere in this volume. The statistics of production, imports and exports as reported in the official statistics are summarized in the following tables:

MINERAL PRODUCTION OF NEW SOUTH WALES. (*a*) (IN METRIC TONS AND DOLLARS; £1=$5.)

Year.	Alunite.		Antimony and Ore.		Bismuth		Chrome Ore		Coal.		Coke.		Cobalt Ore.		Copper Ingot.	
1893...	834	$16,420	1,802	$125,460	..	......	.....	....	3,307,812	$5,858,610	18,144	$101,165	26	$1,525	1,068	$221,175
1894...	876	17,240	1,270	38,720	..	......	3,067	$61,800	3,730,829	5,777,865	35,009	166,045	3	50	1,582	305,170
1895...	845	16,640	486	36,255	..	.....	4,677	65,240	3,798,325	5,476,635	28,072	123,415	6	130	2,838	596,500
1896...	1,393	20,580	135	9,170	41	$2,450	3,913	56,400	3,972,068	5,627,405	26,773	109,255	..	......	4,524	1,001,180
1897...	736	10,860	172	18,060	3	4,000	3,433	51,345	4,453,729	6,150,205	65,229	226,960	..	......	6,864	1,499,145

Year.	Copper Ore and Regulus		Fire-clay.		Gold—Kg.		Iron. (*b*)		Iron Oxide		Lead, Argentiferous.			Limestone Flux.	
											Ore.	Metal	Value.		
1893....	1,032	$70,955	21	$230	5,576·5	$3,256,430	2,226	$73,930	1,280	$7,630	158,353	59,335	$14,767,945	132,625	$555,205
1894....	590	62,235	24	300	10,102·2	5,783,585	2,406	85,850	440	3,350	140,018	43,225	10,976,695	91,430	346,445
1895....	1,075	107,925	20	275	11,202·6	6,579,646	2,442	78,100	155	1,740	193,266	30,162	7,805,050	105,861	340,800
1896....	15	375	35	345	9,221·2	5,366,800	4,798	166,415	381	4,005	271,640	19,886	8,794,665	90,346	271,305
1897....	169	4,255	..	....	9,088·9	5,442,065	3,291	109,310	234	2,680	275,249	18,395	8,407,640	68,671	208,990

Year.	Man. Ore.		Opal—Kg.		Shale, Kerosene.		Silver—Kg.		Tin Ingot.		Tin Ore.		Zinc Ore.		Sundry Minerals
1893..	..	.....	206·8	$41,575	56,551	$506,105	16,546	$390,655	2,679	$1,115,695	150	$33,020	......	....	$2,785
1894..	14	$220	91·2	28,420	21,510	158,905	26,339	470,750	2,653	897,225	193	38,760	......	....	4,460
1895..	3	50	153·3	30,000	60,377	376,095	17,112	409,290	2,235	680,400	78	12,715	......	....	23,185
1896..	..	.....	630·4	125,000	32,348	171,010	6,307	132,590	1,737	496,060	98	14,525	......	....	4,620
1897..	..	....	2400·4	475,000	34,635	203,060	4,666	83,555	1,159	350,640	14	2,800	29,303	118,440	87,160

(*a*) From the Annual Report of the Department of Mines and Agriculture, New South Wales, 1897. (*b*) Manufactured from old iron.

MINERAL IMPORTS OF NEW SOUTH WALES. (*a*) (IN METRIC TONS AND DOLLARS; £1=$5.)

Year.	Cement—Barrels.		Chrome Ore.		Coal.		Coke.		Copper.			
									In Matte and Regulus.		Ingot.	
1894....	145,662	$361,115	895	$16,835	379	$1,225	48,347	$183,420	187	$29,225	353	$71,040
1895....	162,621	330,760	3,840	66,705	378	1,870	43,206	134,160	94	19,155	246	50,480
1896....	126,242	239,025	5,537	109,920	807	4,870	43,824	675,535	93	19,380	5	1,085
1897....	184,831	403,210	2,731	49,240	1,755	6,065	33,970	511,980	42	8,680	6	1,585

Year.	Copper—*Continued.*			Gold.		Iron and Steel.			Iron.			
	Ore.		Rod, Sheet, and Wire.	Coin.	Bullion.	Certain Manufactures of. (*c*)		All Other.	Pig.		Ore.	
1894....	669	$32,535	$16,070	$1,498,790	$13,133,480	50,987	$2,654,950	$162,660	3,744	$59,880	1,409	$7,275
1895....	379	20,620	11,400	279,225	10,051,195	55,004	2,642,300	142,035	6,568	98,055	1,522	6,965
1896....	315	12,875	27,895	2,407,100	9,943,070	67,978	3,626,075	....	8,155	124,255	103	555
1897....	1,401	92,440	96,445	5,749,095	10,208,230	71,469	3,904,105	68,495	16,256	269,470	1,298	8,065

Year.	Lead.				Nickel Ore.		Quicksilver. (*No. of Flasks.*)		Salt.			
	Pig.		Pipe, Sheet and Old.						Rock.		In Bags.	
1894....	7,016	$334,545	53	$4,445	1,432	$39,690	1,078	$33,535	5,900	$35,245	26,099	$230,045
1895....	4,775	224,330	18	1,955	317	9,865	797	29,595	10,811	70,515	26,956	223,810
1896....	7,440	362,720	268	19,720	3	120	261	9,250	7,275	39,825	25,784	207,380
1897....	5,281	289,590	303	20,235	203	5,000	470	12,255	4,654	34,110	36,722	261,230

Year.	Saltpeter.		Silver.							Slate.			
			Ingot—Kg.		Ore.		In Matte—Kg.		Coin.	Roofing—*Number*		Slabs. *Number.*	
1894....	131	$12,480	415	$8,350	2,958	$270,140	(*b*)	....	$50,000	1,077,501	$41,555	2,755	$5,525
1895....	138	15,410	95	1,945	922	63,275	503	$10,110	109,470	954,449	30,635	2,150	7,405
1896....	162	19,905	180	3,900	15,073	672,770	3,958	79,245	49,000	2,219,681	90,595	3,602	11,675
1897....	176	18,455	288	5,060	1,288	38,680	1,928	35,500	131,630	1,908,475	78,990	2,605	8,575

Year.	Soda Ash.		Soda.						Stone, All Kinds, Excepting Marble.	Sulphur.	
			Bicarbonate.		Caustic.		Crystals.				
1894....	1,057	$30,120	488	$17,070	1,598	$96,550	1,577	$28,445	$11,425	2,067	$43,340
1895....	1,068	27,370	636	24,520	850	46,985	1,164	24,895	69,740	1,728	32,800
1896....	1,749	60,300	611	25,080	1,010	48,465	745	18,305	40,925	1,390	22,235
1897....	1,452	50,583	788	31,395	1,213	66,900	495	7,940	27,065	2,287	33,955

Year.	Tin.					Yellow Metal.	Zinc.			
	Ingot.		Ore.		Plates.		Slabs.		Sheet and Manufactures.	
1894....	1,097	$379,760	2,203	$513,480	$299,665	$63,920	683	$57,610	278	$29,060
1895....	946	272,700	1,493	270,980	372,870	80,525	693	53,880	320	28,110
1896....	910	265,505	1,073	169,875	353,345	92,895	1,416	116,115	368	35,500
1897....	693	205,135	662	104,510	200,350	69,290	1,341	122,815	561	58,010

(*a*) From the *New South Wales Statistical Register*. (*b*) No returns given. (*c*) The figures for "certain manufactures" of iron and steel in the years 1894, 1895 and 1897 include only those articles the weight of which is given in the official returns. The figures for 1896 include all manufactures of iron and steel. "All other" manufactures comprise the values of those articles for which no weights are given in the reports.

MINERAL EXPORTS OF NEW SOUTH WALES. (*a*) (IN METRIC TONS AND DOLLARS; £1 = $5.)

Year.	Antimony						Acid, Sulphuric.		Bismuth Ore.		Cement. *Barrels.*		Coal.	
	Ore.		Auriferous.		Metal.									
1894....	679	$32,240	561	$56,945	30	$4,535	140	$20,720	2·0	$50	6,927	$16,060	2,159,127	$4,023,845
1895....	225	9,945	209	19,850	44	6,460	247	19,430	0·6	210	10,279	23,275	2,200,890	3,862,775
1896....	87	2,810	25	3,160	23	3,200	100	5,535	45·0	3,350	33,987	74,650	2,514,506	4,501,220
1897....	49	2,510	21	1,500	102	14,050	52	3,495	3·3(*f*)	4,150	15,272	39,185	2,739,769	4,760,270

Year.	Coke.		Chrome Ore.		Cobalt Ore.		Copper.								
							Ingot.		In Matte.		Regulus.		Ore.		Rod, Sheet, Wire.
1894....	11,858	$53,330	3,977	$80,855	590	$15,415	1,932	$373,195	373	$70,640	(*e*)	....	301	$7,050	$8,670
1895....	8,948	42,625	8,151	140,535	235	7,265	2,926	619,730	194	37,710	1,086	115,925	55	1,730	3,370
1896....	5,521	22,615	9,465	145,165	284	9,750	4,026	904,540	595	117,595	(*e*)	....	15	375	3,000
1897....	32,221	135,225	5,959	101,320	73	2,500	4,126	964,880	2,778	549,440	(*e*)	....	169	4,255	5,770

Year.	Gold.				Iron and Steel.				Lead.					
	Ore.	Coin.	Bullion.		Pig.		Manufactures. (d)		Pig.		Pipe and Sheet.		Argentiferous.	
1894....	$227,315	$9,877,865	2,848	$1,663,455	400	$6,585	9,646	$587,685	3,836	$165,020	326	$21,060	43,193	$7,716,065
1895....	946,845	13,552,800	3,758	2,174,985	201	3,825	14,504	726,680	2,737	131,400	577	39,945	30,162	4,798,680
1896....	488,425	18,014,930	2,576	1,417,520	1,470	26,350	17,289	1,039,375	3,150	167,425	750	52,535	19,897	3,974,335
1897....	282,040	21,733,235	3,295	2,043,690	1,183	22,495	19,515	1,297,320	2,100	119,395	1,043	77,725	18,395	3,227,385

Year.	Nickel Ore.		Saltpeter.		Salt.				Silver.				
					Rock.		Other, in Bags.		Ore.		In Matte—Kg.		Coin.
1894....	2,148	$53,545	5	$770	199	$1,855	2,660	$34,795	143,200	$3,416,320	7,019	$133,575	$63,820
1895....	343	9,175	17	2,480	855	7,995	3,909	46,035	194,158	3,068,985	5,855	118,190	38,330
1896....	(e)	....	6	790	457	3,605	2,820	32,440	284,524	5,447,925	7,550	152,960	66,450
1897....	203	4,900	42	5,730	330	3,065	4,752	58,380	275,807	5,205,525	2,454	41,860	99,545

Year.	Soda Ash.		Soda.				Stone.	Sulphur.		Tin.				
			Bicarbonate.		Caustic.					Ore.		Ingot.		Plate.
1894....	178	$6,560	50	$2,400	214	$14,675	$4,665	34	$1,305	214	$42,835	3,749	$1,276,725	$51,585
1895....	175	5,800	56	2,500	67	4,000	4,705	42	2,105	80	12,955	3,181	970,410	87,445
1896....	207	7,010	39	1,790	148	7,545	7,015	139	4,240	99	14,525	2,646	763,595	79,100
1897....	117	4,010	61	2,795	90	4,660	3,545	32	1,300	143	2,800	1,837	556,345	78,205

(a) From the *New South Wales Statistical Register.* (d) Values including those of manufactures for which no quantity is stated, as follows: In 1894, $5,955; 1895, $17,735; 1896, $17,970; 1897, $9,125. (e) Not stated in the reports. (f) Probably bismuth metal. (g) There was also exported from N. S. W. in 1897, iron oxide: 246 metric tons, $2,830, and iron ore, 5 metric tons, $35.

MINERAL PRODUCTION OF NEW ZEALAND. (a) (b) (IN METRIC TONS; £1=$5.)

Year.	Antimony Ore.		Coal.		Coke.		Gold—Kg.		Manganese Ore.	
1893............	336	$17,335	702,613	$1,919,525	52	$265	7,054	$4,565,690	324	$4,715
1894............	45	3,805	731,059	1,978,545	109	800	6,893	4,439,195	544	5,780
1895............	55	7,430	752,680	2,053,810	293	3,575	9,129	5,810,820	213	2,625
1896............	21	2,250	805,537	2,143,240	107	1,315	8,202	5,207,140	66	1,025
1897............	10	785	854,164	2,259,575	....	....	7,827	4,901,020	183	2,705

MINERAL PRODUCTION—*Continued.* ‖ MINERAL IMPORTS. (c) (IN METRIC TONS; £1=$5.)

Year.	Mixed Ores.		Silver—Kg.		Coal.		Iron.	Machinery.	Railway Materials	Tools and Implements.
1893............	38	$3,250	1,962	$48,715	119,323	$559,780	$2,040,680	$1,060,220	$274,705	$337,900
1894............	25	1,765	1,685	33,485	114,768	525,955	1,903,210	889,980	213,985	309,685
1895............	63	4,400	2,644	53,395	109,929	488,105	1,739,640	770,045	234,940	299,325
1896............	38	6,675	2,933	52,945	103,384	470,690	2,274,165	1,417,200	163,270	350,690
1897............	1,586	29,460	5,716	104,360	112,681	490,695	2,359,525	2,005,265	682,855	450,905

(a) From *New Zealand Mines Statements*, by the Hon. A. J. Cadman, Minister of Mines, Wellington. (b) The exports are stated to be identical with the production, except in the case of coal, of which substance the exports were as follows: in 1893, 70,242 tons, value $363,495; in 1894, 76,204 tons, value $367,190; in 1895, 87,363 tons, value $416,710; in 1896, 80,796 tons, value, $359,920; in 1897, 77,280 tons, value, $347,975 (c) From *British Statistical Abstracts.*

MINERAL PRODUCTION OF QUEENSLAND. (a) (IN METRIC TONS AND DOLLARS; £1=$5.)

Year.	Antimony Ore.		Bismuth Ore.		Building Stone. (b)		Coal.		Copper Ore.		Gems.	Gold—Kg.		Lead.	
1893........	30	$1,440	75	$53,380	27,510	$21,820	268,632	$626,700	302	$19,110	$22,500	19,189	$10,796,450	127	$4,600
1894........	28	1,400	66	31,350	42,997	53,115	275,036	572,965	422	47,910	b12,085	21,135	11,891,442	e458	21,850
1895........	(c)	....	60	16,290	52,206	53,305	328,237	662,650	441	65,485	b29,575	19,647	11,054,435	369	18,170
1896........	(c)	....	(c)	....	(c)	....	377,332	774,935	589	105,210	(c)	19,917	11,206,737	628	30,900
1897........	(c)	....	1	670	(c)	....	364,142	699,445	293	63,225	(c)	25,129	12,765,705	391	20,585

Year.	Manganese Ore		Opal.	Silver Ore.		Silver—Kg.		Tin Ore.		Wolfram Ore	
1893........	(c)	....	(c)	696	$86,545	10,552	$212,040	2,473	$534,765	e4	$240
1894........	142	$2,000	$60,000	138	9,375	5,697	110,385	2,917	511,385	107	3,550
1895........	361	5,515	163,750	77	10,060	6,999	150,210	2,148	340,665	25	2,710
1896........	305	4,500	116,500	(c)	....	8,687	160,810	1,579	245,090	3	300
1897........	403	7,530	51,250	(c)	....	7,280	125,590	1,222	187,545	13	975

MINERAL IMPORTS OF QUEENSLAND. (*d*) (IN METRIC TONS; £1—$5.)

Year.	Cement and Plaster of Paris—*Barrels.*		Coal.		Coke.		Glass and Glassware.	Iron and Steel.
1893..........	27,353	$45,215	14,688	$38,030	74	$455	$74,175	$749,070
1894..........	25,715	49,035	13,166	44,980	50	565	99,675	1,020,590
1895..........	47,244	116,620	18,023	91,575	(c)	....	147,615	1,468,785
1896..........	68,607	163,860	18,743	88,225	5	20	176,330	1,519,595
1897..........	65,230	159,375	21,624	98,370	81	505	103,060	1,586,605

Year.	Petroleum—*Gallons.*		Rails, Track Material.	Gold Bullion—Kg.		Gold Specie.	Silver Bullion, Kg.		Silver Specie.
1893..........	1,375,836	$172,690	$16,060	15·1	$9,330	$4,342,435	(c)	....	$10,580
1894..........	1,311,129	201,890	110,305	24·0	12,925	1,452,930	1·5	$45	2,965
1895..........	1,638,975	282,530	139,980	56·3	33,375	3,001,000	7·1	210	1,750
1896	1,271,910	265,645	138,230	189·2	108,405	400,000	5·0	120	13,380
1897..........	1,369,178	278,070	121,950	176·9	91,490	59,350	11·1	265	113,145

MINERAL EXPORTS OF QUEENSLAND. (*a*) (IN METRIC TONS; £1—$5.)

Year.	Antimony Ore.		Bismuth Ore.		Copper Ore.		Copper Ingot.		Copper Regulus.		Copper Matte.		Gold—Kg.		Gold Ore.	
1893....	204	$17,030	68	$40,105	106	$6,420	29	$6,225	22	$1,775	289	$26,480	19,335	$10,821,385	95	$8,000
1894....	109	9,055	37	23,510	64	4,900	148	25,400	376	41,870	127	18,500	20,030	11,909,580	33	2,190
1895....	44	1,935	44	24,760	102	10,840	315	62,695	211	51,885	225	31,200	21,529	11,360,545	199	9,555
1896....	(c)	....	0·7	750	209	21,590	377	71,045	237	56,970	142	12,400	19,496	10,571,285	68	5,290
1897....	25	2,500	39	20,150	241	15,425	196	34,515	143	35,075	150	21,925	25,293	12,843,510	220	19,285

Year.	Silver Ore.		Silver-Lead Bullion.		Silver Precipitate.		Doré Silver Bullion. Kg.		Mundic Ore.		Precious Stones	Tin Ore.		Tin.		Wolfram Ore.	
1893....	670	$80,545	2,560	$783,875	(c)	......	(c)	....	7	$45	$24,370	1,744	$445,520	245	119,615	4	$240
1894....	136	9,375	1,008	473,085	(c)	......	(c)	....	6	355	12,085	1,783	365,745	473	173,400	69	6,570
1895....	77	10,060	561	198,710	1	$6,785	489	$26,340	(c)	....	29,575	1,180	224,175	459	146,760	34	2,965
1896....	181	46,350	174	103,630	0·8	1,500	3,163	144,460	5	770	11,730	751	136,410	355	97,485	23	2,365
1897....	259	35,080	301	37,350	(c)	......	1,823	210,565	(c)	....	5,700	636	115,750	181	675	(c)	.....

(*a*) From *Annual Reports of the Under Secretary of Mines*, Queensland, when not otherwise stated. (*b*) From *Mineral Statistics of the United Kingdom*. (*c*) Not reported. (*d*) From *Statistical Abstracts for the Several Colonial and other Possessions of the United Kingdom*. (*e*) Quantity exported.

MINERAL IMPORTS OF SOUTH AUSTRALIA. (*a*) (IN METRIC TONS; £1—$5.)

Year.	Coal.		Coke.		Iron.				Silver-Lead.				Gold and Silver Bullion and Specie.
					Bar, Sheet, Hoop, and Rod.		Galvanized, Plain, and Corrugated.		Metal.		Ore.		
1893....	244,589	$745,860	68,112	$446,790	5,394	$194,475	4,831	$347,680	57,735	$10,056,205	149,018	$4,321,140	$3,805,735
1894....	259,695	768,670	119,557	751,525	4,848	175,620	6,236	450,995	38,018	6,750,430	109,386	2,511,160	307,670
1895....	261,117	675,450	91,293	531,875	5,158	163,205	4,153	286,970	25,079	4,293,305	134,878	1,989,950	978,140
1896....	343,583	895,480	68,454	315,600	7,656	263,610	7,110	527,475	25,806	4,379,645	234,297	4,143,515	1,782,730
1897....	378,495	1,012,140	75,205	402,620	6,505	229,970	6,438	454,750	17,466	2,989,810	301,990	5,221,195	78,340

(*a*) From *British Statistical Abstracts*, except the figures for 1896, which are from the *Statistical Register of South Australia*.

MINERAL EXPORTS OF SOUTH AUSTRALIA. (*a*) (IN METRIC TONS AND DOLLARS; £1 — $5.)

Year.	Coal.		Copper.		Copper Ore.		Gold—Kg.		Lead.		Silver Lead.	
1893	(*g*)	....	4,389	$1,044,835	1,279	$29,040	1,051·9	$642,580	17,074	$794,155	56,909	$9,977,440
1894	19	$55	5,023	1,043,195	314	9,815	1,051·9	642,580	19,978	917,800	40,098	6,619,780
1895	876	1,725	5,251	1,132,470	268	10,085	1,152·5	643,960	20,766	961,220	25,134	8,026,310
1896	37,423	110,480	*b*4,689	1,097,320	354	15,750	265·1	157,905	24,358	1,183,620	18,869	1,416,555
1897	39,480	121,210	4,784	1,192,835	554	23,200	556·5	345,670	21,283	1,099,020	16,474	2,548,515

Year.	Silver-Lead Ore.		Manganese Ore.		Matte.		Mica.	Salt.		Slate, Roofing.
1893	3,937	$84,340	2,467	$31,795	(*g*)	....	(*g*)	7,229	$46,705	(*g*)
1894	7,971	194,110	177	2,585	87	$14,605	$12,650	7,747	55,235	(*g*)
1895	44,839	1,132,015	49	730	(*g*)	....	(*g*)	(*g*)	....	(*g*)
1896	129,438	3,652,360	(*g*)	....	3,385	465,910	1,660	*d*19,332	127,280	$4,925
1897	180,870	5,394,000	(*g*)	....	2,324	342,235	5,690	*d*29,374	174,805	18,865

Year.	Spelter.		Tin Ore.		Various Ores.	
1893	73	$4,630	(*g*)	....	21	950
1894	46	2,895	(*g*)	....	......	2,155
1895	26	1,610	70	$9,075	6	360
1896	36	2,020	(*e*)	....	2	90
1897	11	590	(*e*)	....	52	2,095

(*a*) From the *Mineral Statistics of the United Kingdom*, except the figures for 1896, which are from the *Statistical Register of South Australia*, Part IV. and those for 1897, which are from *British Statistical Abstracts*. (*b*) Includes yellow metal. (*d*) Exported in 1896 also rock salt, 3 metric tons ($50). In 1827, 2 metric tons ($30). (*e*) No tin ore exported in 1896; but tin, block and sheet, 5 metric tons ($1,795). In 1897, 2 metric tons ($825). (*g*) Not reported.

MINERAL PRODUCTION OF TASMANIA. (*a*) (IN METRIC TONS AND DOLLARS; £1 — $5.)

Year.	Coal.		Copper Ore.		Gold—Kg.		Iron Ore.		Lead-Silver Ore.		Stone.				
											Bluestone.			Limestone.	
											Cubic Yards	Tons.	Total Value.	*Tons*	Total Value
1893	34,587	$170,210	59	$12,255	1,158	$729,375	(*b*)	.....	14,581	$993,050	(*b*)	(*b*)	....	1,546	$3,965
1894	31,417	65,920	127	25,000	1,806	1,127,425	(*b*)	.....	21,401	1,405,215	(*b*)	(*b*)	....	1,433	5,270
1895	33,703	70,145	34	2,600	1,709	1,061,645	(*b*)	.....	18,194	879,785	7,144	7,961	$13,540	1,419	7,500
1896	44,280	86,770	52	7,450	1,917	1,187,870	208	$250	21,150	1,114,740	4,256	(*b*)	3,435	2,621	3,101
1897	43,210	84,640	113,261	1,618,250	2,399	1,446,205	909	370	17,806	986,125	(*b*)	(*b*)	....	1,702	6,200

Year.	Stone—*Continued.*						Tin Ore.					
	Freestone, Flagstones and Building Stones.			Rubble or Metal.			Quantity.			Value.		
	Cubic Feet.	*Loads*	Total Value.	*Cubic Feet.*	*Tons.*	Total Value.	Alluvial.	Lode.	Total.	Alluvial.	Lode.	Total.
1893	37,470	(*b*)	$3,965	517,860	700	$16,405	4,491	291	4,782	$968,570	$63,485	$1,032,055
1894	48,019	506	5,895	103,050	(*b*)	11,770	4,352	1	4,353	784,125	200	784,325
1895	21,470	(*b*)	6,170	1,140	6,200	5,175	3,887	39	3,926	726,230	180	766,410
1896	13,575	4,220	7,590	*c*4,556	(*b*)	1,635	3,867	(*b*)	3,867	804,395	(*b*)	804,395
1897	82,197	(*b*)	5,805	95,202	13,274	11,010	3,281	1	3,282	545,545	85	545,630

(*a*) From *Statistics of the Colony of Tasmania*, Part V., Production. (*b*) Not reported. (*c*) Represents cart-loads.

MINERAL IMPORTS OF TASMANIA. (*a*) (IN METRIC TONS AND DOLLARS; £1—$5.)

Year.	Cement and Whiting.		Coal and Coke.		Copper and Lead.	Earthen and Glassware	Glass, Sheet, Crown, and Plate.	Gold Specie.	Iron and Tin.	Jewelry and Plate.	Railway Material	Silver Specie and Bullion.
1893	(*b*)	......	32,167	$121,840	$8,205	$51,175	$10,860	$91,750	$121,100	$18,375	$57,000	$2,230
1894	1,033	$8,490	39,881	60,320	5,090	55,605	8,220	170,000	115,050	18,620	26,745	5,000
1895	1,605	14,855	36,139	58,615	5,590	36,140	8,125	485,000	151,120	11,960	58,465	15,065
1896	897	8,775	28,145	11,838	5,105	69,010	10,525	(*b*)	179,835	22,250	148,960	11,100
1897	1,607	16,895	41,436	105,475	4,950	84,845	15,795	250,000	254,500	44,740	89,190	3,135

(*a*) From *Statistics of the Colony of Tasmania*. (b) Not reported.

MINERAL EXPORTS OF TASMANIA. (*a*) (IN METRIC TONS AND DOLLARS; £1=$5.)

Year.	Bismuth Ore.		Coal and Coke.		Copper Ore.		Earthen and Glassware	Galena.	Gold.				
									Ore.		Bullion—Kg.		Coin.
1893 (*b*).	(*c*)	....	(*c*)	....	(*c*)	$229,830	(*c*)	40	(*c*)	....	1,029	655,520	$2,19[illegible]
1894....	3·5	$4,115	391	$5,555	571	553,065	$345	(*c*)	547·5	$29,380	1,644	1,095,295	2,5[illegible]
1895....	(*c*)	....	101	705	205	48,385	40	(*c*)	395·0	28,890	1,664	1,062,650	16,8[illegible]
1896....	1·0	400	380	680	86	8,295	545	(*c*)	1,032·0	42,835	1,759	1,118,065	(*c*)
1897....	(*c*)	....	2,271	3,095	53	4,560	360	(*c*)	421·6	19,970	1,829	1,131,440	(*c*)

Year.	Iron.						Lead.		Nickel Ore.		Silver.			
	Oxide.		Old Metal.		Rails.						Ore.		Bullion.	
1893 (*b*).	(*c*)	....	(*c*)	....	(*c*)	$12,960	(*c*)	....	(*c*)	....	13,261	769,260	(*c*)	....
1894....	29	$170	1	$15	(*c*)	9,670	(*c*)	....	110	$3,560	20,018	1,076,970	109	$22,25[illegible]
1895....	66	325	96	670	117	1,395	7·0	$380	(*c*)	....	20,575	1,139,580	(*c*)	....
1896....	29·	145	300	2,710	61	900	2·5	100	(*c*)	....	21,150	1,114,740	(*c*)	....
1897....	(*c*)	....	198	1,450	57	50	19·3	725	(*c*)	....	19,828	1,084,465	(*c*)	....

Year.	Tin.				Zinc.	
	Ore.		Metal.			
1893 (*b*).	140	....	3,178	$1,330,795	(*c*)	....
1894....	121	$20,780	2,981	991,490	8·0	$345
1895....	13	1,465	2,771	837,305	9·0	500
1896....	2	10	2,743	795,180	10·4	380
1897....	5	2,960	2,462	749,970	42·9	1,890

(*a*) From *Statistics of the Colony of Tasmania*. Additional products in 1897: Jewelry and plate, $6,1[illegible]; silver coin, $12,875; unclassified ore, 30 metric tons, $1,450; mineral sand, ½ metric ton, $5; copper metal 1½ tons, $40; blister copper, 4,585 tons, $1,825,565; iron ore, 904 tons, $4,015; manganese ore, 2 tons, $30; mineral oxide, 5 tons, $45. (*b*) The complete data for 1893 are not available, and the values of the exports stated for those years represent aggregate valuations, viz.: Under gold, the value given is that of ore, pyrites, and metal; under silver, that of ore and bullion; under copper, that of ore and pyrites; under iron, that of ore, oxide, old metal, and rails; and under tin, the value is that of ore and metal. (*c*) Not stated in the reports.

MINERAL PRODUCTION OF VICTORIA. (*a*) (IN METRIC TONS AND DOLLARS; £1=$5.)

Year.	Antimony Ore.		Coal.		Lignite.		Copper Ore		Gold—Kg. (*b*)		Lead Ore.		Slate and Flagging.		Tin Ore	
1893..	90	$2,155	93,194	$245,835	4,572	$11,035	(*d*)	....	20,874	$13,422,520	(*d*)	......	171	$225	54	$10,85[illegible]
1804..	36	875	174,406	474,995	3,571	10,040	492	$73,810	22,299	14,339,080	(*d*)	......	315	425	61	11,0[illegible]
1895..	(*d*)	....	197,334	592,000	1,988	5,239	7	1,050	23,019	14,801,720	20	$500	366	450	76	15,0[illegible]
1896..	(*d*)	....	230,187	565,060	5,908	10,705	(*d*)	....	25,041	16,101,740	(*d*)	......	386	485	47	8,9[illegible]
1897..	5	100	240,057	543,200	4,894	6,885	(*d*)	....	25,280	16,255,320	(*d*)	......	(*d*)	*e*125,000	48	8,2[illegible]

MINERAL IMPORTS OF VICTORIA. (*c*)

Year.	Coal.		Iron and Steel.	
1893..	611,826	$2,092,420	23,141	$1,753,310
1894..	550,709	977,075	22,795	1,073,520
1895..	553,343	1,005,235	42,788	1,422,400
1896..	511,020	970,175	45,126	1,996,845
1897..	535,812	1,143,235	56,832	2,294,645

MINERAL EXPORTS OF VICTORIA. (*c*)

Year.	Coal.		Coke and Charcoal.		Gold.		Silver. (Specie)
					Bullion.	Specie.	
1893..	452	$2,115	456	$7,290	$479,960	$13,775,935	$56,9[illegible]
1894..	495	1,555	193	1,485	178,990	18,414,385	76,0[illegible]
1895..	161	600	210	2,060	612,505	18,141,180	41,8[illegible]
1896..	319	1,770	295	3,050	323,025	16,171,535	53,185
1897..	283	1,980	1,212	15,690	969,055	31,392,535	11,6[illegible]

(*a*) From Annual Reports of the Secretary for Mines of the Colony; additional products in 1897: Bricks, estimated value, $11,250; pottery, estimated value, $8,900. (*b*) The values are not separately stated in the report, and are estimated at $20 per oz. = $643·02 per kg. (*c*) From *British Statistical Abstracts*. (*d*) Not stated in the reports. (*e*) Represents estimated value of building stones (basalt, sandstone, granite, slate, flagging, etc.)

MINERAL IMPORTS OF WESTERN AUSTRALIA. (*a*) (IN METRIC TONS AND DOLLARS; £1=$5.)

Year.	Alkali.				Blue-stone.	Brass-ware.	Bricks. *Number.*		Cement. *No. of Barrels.*		Coal.	
	Soda Crystals.		Caustic Soda.	Soda Ash, Nitrate of Potash, etc.								
1893......	36	$570	$1,745	$1,575	$440	$4,765	44,838	$1,175	6,902	$20,690	19,117	$76,010
1894......	(*c*)	....	2,505	2,920	775	1,530	5,714	325	7,650	18,035	26,348	108,910
1895......	(*c*)	1,270	2,460	1,240	455	9,220	941,041	6,305	15,048	30,290	39,125	147,[illegible]
1896......	(*c*)	2,150	2,420	4,940	735	18,980	(*c*)	19,340	(*c*)	70,845	71,680	197,515
1897......	143	2,730	3,955	1,195	625	23,935	1,223,055	12,120	47,457	81,305	139,369	474,[illegible]

Year.	Coin.			Coke and Patent Fuel. *No. of Sacks.*		Copper.		Earthen and Chinaware.	Glass and Glassware	Gold and Silver Leaf.
	Copper.	Silver.	Gold.			Ingot.	Rod, Sheet, and Wire.			
1893......	(*c*)	$21,500	$675,000	2,162	$1,440	(*c*)	$2,015	$22,465	$25,510	(*c*)
1894......	$1,500	58,500	1,139,500	44	80	(*c*)	4,095	33,575	31,385	$475
1895......	1,850	92,000	4,540,000	368	510	(*c*)	4,335	68,220	64,410	2,765
1896......	4,250	147,095	4,751,850	(*c*)	2,485	$8,375	9,440	87,485	103,210	2,565
1897......	1,250	103,000	225,000	(*c*)	10,825	1,695	12,295	101,320	137,815	2,505

Year.	Iron.					Iron and Steel Wire.		Lead—Sheet, Pig and Pipe.		Marble and Stone, Rough and Wrought.	Metal Sheathing.
	Pig.		Galvanized (Corrugated Sheet).		Other Manufactures.						
1893......	381	$4,440	1,618	$119,465	$99,965	1,209	$59,475	66	$4,875	$4,280	$3,520
1894......	36	1,100	2,135	164,835	172,465	(*c*)	69,095	34	2,255	1,895	6,285
1895......	289	3,870	3,848	289,935	230,670	(*c*)	53,995	96	6,040	7,140	3,085
1896......	639	12,455	7,467	525,050	771,945	(*c*)	67,715	321	20,250	22,070	1,740
1897......	953	14,095	11,689	897,035	661,630	(*c*)	82,545	537	37,115	20,190	6,890

Year.	Mineral Oil and Turpentine. *No. of Gallons.*		Nails.	Paints and Colors.	Paraffine Wax.	Plaster of Paris.		Quicksilver. *No. of Flasks*		Salt.			
										Rock. *No of Sacks*		Other Kinds.	
1893......	208,555	$52,135	$16,260	$15,750	$3,945	(*c*)	....	56	$2,750	14	$215	414	$4,070
1894......	277,968	48,630	31,960	14,050	1,050	58	$1,035	165	6,435	157	90	649	8,020
1895......	495,024	82,730	47,840	17,750	2,640	52	1,050	323	9,880	212	595	833	9,710
1896......	853,649	159,765	120,790	45,140	3,780	245	3,545	840	20,235	(*c*)	260	997	10,690
1897......	1,058,022	163,735	151,560	42,570	3,750	496	6,225	623	22,705	(*c*)	410	1,211	11,568

Year.	Steel.	Stone.			Sulphur. *No. of Casks.*		Tiles.	Tin.		Whiting.	Zinc, Sheet.
		Grindstones. *Number.*		Other Kinds.				Ingot.	Block, Foil, Plate, and Wares.		
1893......	$5,610	2,006	$1,780	$1,325	49	$380	$925	$530	$5,810	$595	$3,445
1894......	18,470	*b*718	680	615	537	1,715	100	(*c*)	16,125	1,030	795
1895......	21,765	*b*700	660	265	344	1,230	2,815	(*c*)	15,900	1,065	3,970
1896......	82,935	(*c*)	2,055	1,490	(*c*)	720	7,585	4,090	34,395	1,950	6,295
1897......	80,340	(*c*)	2,445	665	(*c*)	600	16,365	6,360	24,835	1,530	7,565

(*a*) From the *Blue Books for Western Australia*, except the figures for 1896, which are from the *Annual Report of the Collector of Customs*, wherein only the values of the imports are stated. (*b*) Number of packages. (*c*) Not stated in the reports.

MINERAL EXPORTS OF WESTERN AUSTRALIA. (*a*) (IN METRIC TONS AND DOLLARS; £1 = $5.

Year.	Coal.		Gold.			Iron.		Mica.	Ores.						Quartz.
			Bullion—Kg.		Coin.				Copper.		Lead.		Tin.		
1893......	6,343	$43,750	3,449	$2,106,925	$100,300	(*d*)	.....	$20	51	$3,030	(*d*)	.....	232	$55,670	$2,075
1894......	8,322	45,030	6,446	3,935,495	22,500	(*d*)	.....	(*d*)	(*d*)	....	(*d*)	.....	396	76,370	2,425
1895......	15,095	92,675	7,201	4,398,740	460	(*d*)	.....	15	839	64,760	(*d*)	.....	281	48,515	(*d*)
1896......	(*d*)	75,205	8,748	5,344,040	(*d*)	(*d*)	$130	(*d*)	(*d*)	500	(*d*)	.....	(*d*)	21,690	(*d*)
1897......	23,756	129,010	20,994	12,824,885	3,130,400	(*d*)	4,165	1,045	86	5,165	(*d*)	$20	96	16,375	(*d*)

(*a*) From the *Blue Books for Western Australia*, except the figures for 1896, which are from the *Annual Report of the Collector of Customs*, Perth. In 1896 there were also exported brassware, $320; silver plate, $85; and salt, $80. (*d*) Not stated in the reports.

AUSTRIA-HUNGARY.

The latest official statistics of the mineral production, imports, and exports of Austria-Hungary are summarized in the following tables:

MINERAL PRODUCTION OF AUSTRIA. (*a*) (IN METRIC TONS AND FLORINS.)
(1 florin = 40·52 cents.)

Year.	Alum and Alum Shale.		Antimony Ore.		Arsenic Ore.		Asphalt.		Bismuth Ore.		Coal.		Lignite.	
1892......	20,480	12,863	97	11,765	*Nil.*	....	78	1,407	856	22,308	9,241,126	31,680,036	16,190,273	30,096,801
1893......	13,370	10,862	441	50,708	*Nil.*	....	88	1,301	797	21,818	9,732,651	33,549,863	16,815,955	34,048,927
1894......	10,854	13,495	686	68,632	2	100	116	1,907	570	15,194	9,572,952	33,182,691	17,332,538	32,290,905
1895......	5,716	9,955	695	55,600	*Nil.*	....	404	8,118	185	....	9,722,679	34,104,407	18,389,147	34,923,288
1896......	25,184	20,086	905	80,630	*Nil.*	....	390	11,342	*Nil.*	....	9,899,522	35,254,925	18,882,537	36,227,898
1897......	3,599	2,141	864	73,449	*Nil.*	....	300	8,868	1	....	10,492,771	38,404,864	20,458,093	40,084,423

Year.	Copper Ore.		Gold Ore.		Graphite.		Iron Ore.		Iron Pyrites		Lead Ore.		Manganese Ore.	
1892......	8,636	329,824	164	14,886	20,978	637,012	993,290	2,325,088	1,210	25,292	13,265	922,260	4,558	54,530
1893......	8,576	316,232	477	43,787	23,807	637,870	1,109,111	2,482,191	1,221	25,199	10,696	849,869	5,411	58,358
1894......	7,235	278,849	86	9,907	24,121	881,980	1,214,763	2,676,114	(*b*)	......	12,060	836,744	5,055	55,427
1895......	7,435	286,897	104	38,907	28,443	985,771	1,384,911	2,971,384	(*b*)	......	12,919	883,244	4,352	41,600
1896......	6,823	273,269	416	48,412	35,972	1,216,458	1,448,615	3,446,479	(*b*)	......	14,563	1,058,564	3,950	34,299
1897......	7,405	279,393	647	32,938	38,504	1,352,646	1,613,876	3,766,048	(*b*)	......	14,145	1,161,896	6,012	47,961

Year.	Nickel and Cobalt Ore		Petroleum.		Quicksilver Ore		Salt.		Silver Ore.		Sulphur, Crude Rock		Tin Ore.	
1892......	0·27	.....	89,871	2,742,724	79,447	1,007,829	288,424	20,796,030	14,171	2,672,606	1,804	20,987	33	2,309
1893......	*Nil.*	.....	96,331	3,008,819	76,215	785,493	305,586	20,440,426	18,018	3,415,704	1,950	20,485	26	2,712
1894......	0·55	.....	111,930	3,252,554	84,127	988,754	311,597	24,281,350	18,338	3,000,171	2,435	34,484	21	2,403
1895......	*Nil.*	.....	188,634	4,464,353	86,683	797,218	278,875	23,901,782	18,113	2,294,043	830	8,298	24	2,534
1896......	*Nil.*	.....	262,356	5,188,855	83,305	778,455	308,933	22,985,353	18,701	1,921,533	643	8,271	15	2,473
1897......	*Nil.*	.....	275,204	5,876,692	88,238	814,059	331,084	25,561,441	20,628	1,871,801	530	4,837	16	2,422

Year.	Uranium Ore.		Wolfram Ore.		Zinc Ore.		METALLURGICAL PRODUCTION. (*a*)									
							Alum.		Antimony.		Bismuth.		Copper.		Copperas.	
1892......	18	24,889	72	19,806	33,944	580,365	1,096	69,902	114	44,489	0·55	4,765	837	502,593	1,085	21,628
1893......	21	28,647	43	10,066	30,531	465,230	837	54,344	175	74,090	0·58	5,327	944	568,221	1,220	25,199
1894......	26	59,404	40	10,908	28,491	439,780	1,150	73,656	279	107,341	0·02	2,023	1,341	744,772	1,480	27,809
1895......	31	51,761	35	9,154	25,862	384,330	885	54,935	296	92,031	*Nil.*	......	865	460,990	160	3,328
1896......	30	28,435	22	6,980	26,887	474,033	919	56,555	422	130,532	*Nil.*	......	1,001	552,954	170	4,172
1897......	44	45,011	31	19,248	27,464	530,141	851	55,411	425	113,230	*Nil.*	......	1,083	614,684	125	3,125

Year.	Copper Sulphate.		Gold—Kg.		Iron, Pig.		Lead.		Litharge.		Nickel and Cobalt Products.		Mineral Paint.	
1892......	133	24,587	13·0	17,580	630,790	24,417,266	7,252	1,125,493	2,520	393,356	0·15	196	2,022	57,429
1893......	177	35,275	35·0	52,506	663,345	24,186,083	7,212	1,103,840	2,411	352,648	0·12	163	3,020	75,329
1894......	140	28,832	61·0	96,779	742,372	26,750,603	7,570	1,113,088	2,057	301,514	0·01	156	3,002	76,379
1895......	246	49,231	75·0	116,753	660,549	22,858,257	8,085	1,204,980	2,034	297,708	*Nil.*	.....	3,164	73,632
1896......	265	52,557	69·8	98,913	693,188	23,737,250	9,769	1,527,216	1,738	271,508	*Nil.*	.....	3,979	83,760
1897......	276	58,010	67·6	93,676	762,685	26,626,238	9,680	1,576,210	1,626	265,669	*Nil.*	.....	3,653	84,049

Year.	Quicksilver.		Silver—Kg.		Sulphur.		Sulphuric Acid.		Tin.		Uranium Salts.		Zinc.	
1892......	542	1,148,320	36,658	3,293,746	53	4,338	11,038	315,433	72	85,184	2·0	28,298	5,237	1,264,587
1893......	512	1,068,515	37,344	3,330,265	44	3,613	10,248	336,509	66	76,717	5·0	74,267	5,870	1,212,709
1894......	519	1,056,718	38,246	3,031,631	76	536	9,938	264,280	80	80,860	5·0	69,742	6,810	1,268,940
1895......	535	1,168,512	40,081	2,524,993	*Nil.*	....	7,431	204,485	60	50,536	4·5	64,009	6,456	1,096,008
1896......	564	1,149,695	39,904	2,140,913	*Nil.*	....	7,972	221,182	54	45,650	4·2	47,059	6,888	1,285,391
1897......	532	1,190,121	40,026	1,952,413	*Nil.*	....	*c*8,515	201,054	48	42,014	4·4	42,522	6,236	1,216,609

(*a*) From *Statistisches Jahrbuch des K. K. Ackerbau Ministeriums; Der Bergwerksbetrieb Oesterreiche.* (*b*) Not stated in the report. (*c*) Includes 686 metric tons of oleum.

MINERAL AND METALLURGICAL PRODUCTION OF HUNGARY. (*a*) (IN METRIC TONS AND FLORINS.

(1 florin = 40·52 cents.)

Year.	Alum Ore.		Antimony.				Asphaltum.		Briquettes.		Carbon Bisulphide.	
			Ore. (*b*)		Crude and Regulus.							
1893......	934	1,400	881	67,000	361	144,700	(*c*)	....	34,190	269,800	249	44,800
1894......	634	1,141	1,265	68,733	385	124,642	2,370	157,700	30,057	233,247	248	44,568
1895......	634	1,141	1,240	66,330	465	141,922	2,285	114,800	29,421	234,783	237	42,244
1896......	*d* 20	240	1,361	55,705	500	146,379	2,740	142,200	31,179	247,314	352	63,324
1897......	*d* 60	600	1,800	85,299	523	156,368	3,057	160,100	27,022	216,208	432	17,600

Year	Coal.		Coke.		Copper.		Copperas.		Copper Sulphate.		Galena.	
1893......	982,798	5,161,900	3,188	29,300	343	174,400	762	12,300	2·3	300	214	40,500
1894......	1,037,322	5,447,201	10,250	88,767	271	135,367	795	7,950	0·8	8,000	224	38,700
1895......	1,068,046	5,640,514	12,033	11,302	286	144,220	521	2,086	(c)	....	80	13,600
1896......	1,132,625	5,845,791	25,550	145,396	159	77,258	595	2,380	3·0	712	e405	42,200
1897......	1,118,024	6,034,445	(*c*)	....	213	113,749	592	2,959	6·5	1,542	e525	52,900

Year.	Gold—Kg.		Iron, Pig.		Iron Pyrites.		Lead.		Lignite.	
1893......	2,000·00	2,790,200	319,362	12,103,200	68,189	277,900	2,514	348,600	2,917,899	9,394,300
1894......	2,687·07	4,497,627	312,148	11,216,865	76,870	334,000	2,113	295,015	3,181,071	10,301,700
1895......	3,187·27	4,869,959	322,206	11,802,398	69,195	280,400	2,277	323,174	3,517,901	11,214,391
1896......	3,208·04	5,261,268	383,698	13,942,007	52,697	226,382	1,911	265,822	3,761,728	12,573,124
1897......	3,068·00	5,030,168	402,503	14,478,163	44,454	176,457	2,527	368,466	3,870,530	12,541,026

Year.	Litharge.		Manganese Ore.		Mineral Paints.		Nickel and Cobalt.				Petroleum.	
							Ore.		Products.			
1893......	227	39,600	1,238	1,800	321	8,600	202	18,700	202	18,700	20	690
1894......	689	117,512	3,748	6,700	608	13,853	25	8,106	23	9,374	2,061	66,100
1895......	615	119,592	3,381	1,500	371	11,211	55	21,182	18	5,429	2,083	70,300
1896......	465	77,095	1,991	2,087	334	6,010	46	4,515	18	7,333	2,168	54,011
1897......	155	34,275	(*c*)	....	460	7,370	32	3,133	7·9	2,362	2,299	56,504

Year.	Quicksilver—Kg.		Salt.		Silver—Kg.		Sulphur.		Sulphuric Acid.		Zinc Blende	
1893......	2,500	4,800	167,209	14,605,000	23,631·0	2,120,000	70	5,900	2,337	36,700	77	[illegible]
1894......	1,837	4,699	169,282	14,903,000	20,155·0	1,217,186	93	7,809	4,018	57,046	(c)	......
1895......	1,129	2,400	169,395	13,451,000	20,432·3	1,231,584	102	7,826	4,223	91,500	(c)	......
1896......	1,100	2,236	180,133	15,281,000	19,916·0	1,178,868	138	9,331	3,550	65,253	(c)	......
1897	700	924	171,711	13,267,000	26,790·0	1,536,608	112	8,186	3,397	64,803	(c)	......

(*a*) From *Magyar Statisztikai Evkönyv.* (*b*) This does not include the ore produced in Hungary from which the metallurgical products given in the table were derived. It gives only the amount of ore which was sold, and presumably was not smelted in the country. (*c*) Not stated in the report. (*d*) Represents refined alum. (*e*) Galena and other ores of lead.

MINERAL IMPORTS OF AUSTRIA-HUNGARY. (*a*) (IN METRIC TONS AND FLORINS.)

(1 florin = 40·52 cents.)

Year.	Alum.		Aluminum and Alloys.		Amber.		Ammonia.				Ammoniacal Liquor.		Antimony.			
							Chloride and Sulphate of.		Caustic.				Ore.		Regulus Kg.	
1893......	229·0	20,610	43·3	163,509	63·2	936,840	230·4	70,928	53·9	6,468	856·3	12,845	36·2	4,344	2,400	1,128
1894......	404·0	40,400	32·2	74,760	65·7	990,010	227·1	78,016	69·6	9,048	1,048·0	15,720	71·6	8,592	1,300	481
1895......	338·5	30,465	47·8	93,330	81·8	379,270	305·1	100,797	103·4	13,442	876·7	13,151	15·0	3,150	2,100	777
1896......	359·5	32,355	50·3	94,790	63·6	1,862,400	322·7	116,866	71·1	9,243	506·9	7,604	15·5	3,255	700	231
1897......	346·2	31,158	67·1	125,240	64·9	726,000	339·3	106,074	128·4	16,692	564·6	6,775	8·0	1,680	600	195

Year.	Arsenic, Arsenious Acid, and Sulphide.		Asbestos.				Asphalt.				Barytes and Baryta-White.	
			Crude.		Manufactured.		Crude Rock.		Mastic and Bitumen.			
1893......	287·8	67,144	196·5	26,430	150·8	157,630	3,156	130,391	1,140	79,821	4,468	114,682
1894......	320·0	74,516	555·8	63,541	108·7	153,090	3,918	72,256	758	53,060	4,876	92,380
1895	292·5	67,369	431·6	56,550	108·3	110,175	2,410	44,873	872	52,326	5,008	134,565
1896......	309·2	76,318	185·2	29,526	165·1	167,615	4,715	77,952	1,621	89,145	5,377	142,126
1897......	258·9	60,052	625·3	82,602	134·4	121,935	5,824	97,236	1,309	72,012	4,947	130,323

Year.	Borax.				Brass, German Silver and Tombac				Brass.			
	Crude, and Boracic Acid.		Refined.		Crude, Old, and Remnants.		Bars, Sheets, Wire, etc.		Crude and Old.		Bars, Sheets, etc.	
1893......	1,451	183,004	34·4	11,008	2,081	894,210	52·6	50,909	1,750	700,120	35·4	33,989
1894......	1,320	158,364	64·1	17,948	2,286	894,698	72·7	34,686	1,955	723,461	52·8	34,576
1895......	1,908	282,883	61·9	14,237	2,742	1,166,927	131·4	63,425	2,246	909,673	112·3	67,420
1896......	1,363	208,055	76·5	17,595	3,118	1,345,163	113·4	58,398	2,417	966,960	80·9	58,266
1897......	1,206	171,930	63·4	12,838	2,660	1,222,631	151·9	126,598	2,011	864,816	121·5	79,758

Year.	Calamine and Other Zinc Ores.		Cement.		Chloride of Lime.		Chrome Ore.		Coal and Briquettes.		Coke and Briquettes.	
1893......	3,739	93,694	36,874	665,939	1,459	116,712	570	19,950	3,840,874	25,556,342	309,861	3,144,385
1894......	6,760	214,663	33,286	599,832	1,967	157,336	1,081	51,624	4,048,477	26,135,871	437,299	4,389,109
1895	7,691	312,891	32,012	514,413	2,538	188,616	1,827	94,905	4,503,008	27,790,484	533,402	5,605,297
1896......	9,022	297,946	35,290	566,240	1,989	159,088	1,891	104,005	5,174,321	31,390,639	491,028	5,110,085
1897......	7,863	283,054	32,479	521,891	1,820	145,640	1,109	55,470	5,121,475	31,382,217	533,463	5,699,121

Year.	Colored Earths.		Copper.						Copperas.		Copper Sulphate.		Copper and Iron Vitriol, Mixed.	
			Ore.		Crude and Old.		Bars, Sheets, Wire, etc.							
1893......	4,827	167,892	637·9	210,507	11,822	7,014,931	113·8	84,926	799·8	21,596	1,862·6	335,268	81·3	7,317
1894......	4,205	144,696	24·8	2,976	13,383	7,286,434	97·2	66,375	717·8	18,663	1,916·4	325,788	85·6	7,704
1895......	4,244	145,183	31·0	3,720	11,747	6,922,160	97·9	72,106	871·1	13,068	895·3	170,107	56·2	5,058
1896......	4,362	145,840	0·7	56	13,666	8,372,122	126·2	98,256	575·5	8,633	2,084·4	449,987	40·0	3,600
1897......	4,553	154,244	81·4	11,396	15,926	9,952,699	94·4	77,453	401·3	6,020	6,822·5	1,444,976	153·1	13,779

Year.	Cryolite.		Fertilizers, Mineral.		Fluorspar.		Glass.						Glauber Salt.	
							Enamel and Glazed Ware.		Remnants.		Manufactures.			
1893……	230·3	55,552	8,958	768,731	3,010	43,902	186·5	74,690	4,719	82,592	3,422	1,570,000	6,687·6	167,191
1894……	219·0	54,196	10,217	306,519	3,618	88,745	198·4	79,636	4,736	82,892	4,177	1,754,000	8,503·4	217,723
1895……	229·4	120,485	8,929	267,858	3,528	86,836	236·1	94,440	4,213	73,727	4,038	1,976,000	6,617·5	132,350
1896……	265·5	139,300	9,658	289,728	3,821	94,267	255·8	102,320	3,744	65,515	3,989	2,119,000	4,678·1	81,867
1897……	211·4	110,485	7,583	227,502	4,201	103,360	269·7	107,880	4,651	81,396	3,782	2,187,000	2,879·5	50,395

Year.	Gold.						Graphite.		Gypsum. Crude.	
	Bullion—Kg.		Old and Dross. Kg.		Coin—Kg.					
1893……	48,255	74,181,800	13,888	946,036	47,356	68,397,640	854·5	39,541	862	9,478
1894……	3,827	5,950,830	6,260	50,200	17,416	25,422,862	732·9	36,645	1,375	15,130
1895……	5,223	8,053,617	4,761	190,725	30,279	42,820,107	639·6	26,066	850	9,350
1896……	13,053	19,398,565	5,051	43,150	30,149	41,764,178	696·8	28,102	821	9,032
1897……	33,668	50,326,926	5,572	57,800	32,752	44,739,232	948·0	38,305	980	10,781

Year.	Gypsum Burned.		Iron.						Iron and Steel. Bars, Sheets, Wire, etc.		Jet—Kg.	
			Ore.		Pig and Old.		Manufactures.					
1893……	8,074	161,486	73,248	814,996	75,622	2,491,000	23,323	8,982,000	22,725	2,009,000	850	12,240
1894……	10,629	265,725	88,252	927,746	133,849	3,957,000	20,171	9,334,000	24,313	2,058,000	550	6,875
1895……	10,916	272,909	117,600	1,345,728	175,400	5,127,000	23,078	9,976,000	30,909	2,556,000	600	7,500
1896……	11,736	258,181	107,018	1,327,983	148,217	5,143,000	21,910	10,646,000	27,809	2,635,000	940	11,750
1897……	12,101	266,217	134,778	1,798,413	164,433	5,977,000	28,876	11,456,000	18,625	1,745,000	190	2,280

Year.	Lead.											
	Red and Yellow.		White.		Ore.		Alloys, Crude.		Bars.		Ash—Kg.	
1893……	283·7	56,740	304·4	91,320	298·7	21,507	5,925	725,790	262·8	40,078	3,200	160
1894……	470·2	94,040	243·4	68,152	290·4	19,602	8,842	1,038,914	200·3	30,045	500	25
1895……	371·3	74,260	187·5	46,406	416·4	32,479	8,974	1,166,607	207·9	41,580	900	45
1896……	431·7	82,023	156·1	34,734	540·2	39,435	7,221	974,849	218·3	39,294	2,500	200
1897……	543·3	102,771	111·4	24,229	441·4	35,312	5,887	841,784	147·9	28,101	6,700	469

Year.	Lignite, Brown Coal, and Briquettes.		Lime, Crude and Burned, Inclusive of Caustic and Hydraulic.		Litharge.		Lithographic Stone.		Magnesium.			
									Sulphate.		Chloride.	
1893……	20,111	108,905	26,213	233,652	316·1	60,059	751·2	71,364	53·8	2,152	1,388·9	66,666
1894……	17,453	94,247	26,571	234,825	358·7	64,644	670·8	63,726	47·7	1,908	1,147·8	55,095
1895……	16,797	90,702	19,726	145,087	355·5	63,990	683·9	67,707	48·7	1,948	1,353·4	54,136
1896……	19,981	107,898	25,518	219,795	233·4	44,346	647·3	64,083	53·7	2,148	1,333·0	53,320
1897……	19,609	107,720	28,939	259,041	224·0	40,320	523·7	51,847	7·5	300	1,530·2	57,383

Year.	Manganese Ore.		Marble.		Meerschaum.		Millstones.		Mineral Oil.			
									Crude.		Refined.	
1893……	354·2	11,866	1,816	54,480	200·5	529,850	1,147·7	160,678	147,061	4,832,842	16,764	1,222,298
1894……	2,444·5	95,011	1,888	56,634	229·1	620,250	1,160·4	162,456	126,159	4,430,006	17,019	1,289,322
1895……	2,772·3	98,329	1,886	56,592	185·1	522,700	1,229·5	172,130	120,479	6,078,067	16,876	1,416,150
1896……	7,371·3	264,044	2,347	70,398	202·1	454,520	1,204·9	168,686	69,013	2,822,506	17,943	1,303,280
1897……	8,018·0	304,684	2,333	69,987	214·0	588,500	1,274·9	178,486	70,573	2,271,287	21,249	1,481,831

Year.	Muriatic Acid.		Nickel.				Nitrate of Soda.		Nitric Acid.		Non-Precious Metals, Manufactures of.	
			Crude and Old.		Sheet, Wire, etc.							
1893……	437·6	8,752	29·6	69,443	11·0	28,561	37,627	4,327,049	19·0	3,230	1,579	3,085,000
1894……	498·0	9,952	130·5	261,000	2·8	7,250	39,522	4,149,791	19·3	3,281	1,732	3,205,000
1895. ….	466·7	9,334	168·4	255,968	4·9	14,240	43,059	4,521,176	16·0	2,240	1,890	3,626,000
1896……	528·8	10,576	160·7	252,299	5·2	13,360	33,086	3,308,630	20·7	2,898	1,979	3,708,000
1897……	720·6	14,412	157·4	254,988	6·9	18,090	39,600	3,960,040	22·7	3,178	2,056	4,034,000

Year.	Ozokerite.		Phosphorus and Phosphoric Acid.		Polishing Stones, Whetstones, etc.		Porcelain Earth and Feldspar.		Potash.		Potassium Chloride.	
1893......	10·0	3,200	238·1	285,720	3,487	257,713	4,146	133,755	208·3	41,727	1,869	144,824
1894......	(b)	....	334·8	401,760	3,673	269,816	5,793	172,913	102·1	18,679	2,406	186,428
1895......	2·2	638	206·1	247,320	3,559	265,978	6,532	190,383	285·4	53,590	2,679	200,888
1896......	2·0	600	223·9	268,680	3,851	289,022	7,425	206,652	987·3	164,868	2,475	185,603
1897......	2·4	720	209·5	248,385	4,151	269,822	6,913	118,439	333·5	62,761	2,206	198,655

Year.	Potassium Chromate.		Potters' and Other Clays.		Pyrites.		Quartz and Quartz Sand.		Quicksilver. Kg.		Salt.	
1893......	199·1	71,676	25,149	251,487	45,530	546,360	50,464	347,874	3,400	7,820	26,610	268,023
1894......	75·9	34,155	24,981	249,813	59,085	634,333	61,544	436,737	11,100	18,426	32,822	328,828
1895......	29·5	12,685	27,493	249,254	54,610	655,314	58,494	411,225	4,200	8,316	40,396	405,359
1896......	34·1	14,663	30,072	270,645	50,691	608,289	59,150	449,831	1,300	2,418	53,680	537,267
1897......	33·8	14,534	28,925	261,638	49,462	593,543	61,532	487,895	1,000	1,930	46,057	460,651

Year.	Silver.						Slag and Slag Wool.		Slate, Roofing and Other.		Soda. Caustic.	
	Bullion—Kg.		Old and Dross—Kg.		Coin—Kg.							
1893......	57,340	3,497,965	400	4,150	58,550	3,362,823	1,659	7,799	18,887	1,147,000	1,447	173,586
1894......	73,140	3,705,004	4,210	78,620	56,130	2,370,840	1,065	5,008	17,607	1,069,000	1,668	166,974
1895......	49,370	2,482,147	60	1,740	4,400	225,882	981	4,612	15,667	982,000	1,155	112,567
1896......	138,420	7,219,890	550	2,150	5,730	343,110	240	1,129	15,932	970,000	835	71,794
1897......	99,900	4,535,460	1,000	6,520	3,670	189,860	4,717	22,168	16,758	1,015,074	1,450	123,796

Year.	Soda. Crude and Crystallized.		Sodium Chromate.		Sulphate of Sodium.		Stone.				Sulphur.	
							Paving.		Not Elsewhere Specified.			
1893......	94·6	3,312	183·6	56,916	114·6	16,044	10,607	127,285	71,302	1,141,767	13,090	621,778
1894......	258·2	9,038	248·4	94,392	140·5	16,860	24,860	248,598	97,609	1,161,016	14,410	684,482
1895......	39·9	1,197	152·7	53,445	137·2	6,860	5,459	54,590	79,869	822,608	14,709	661,902
1896......	57·5	1,725	40·3	14,105	144·1	7,205	8,476	84,759	87,796	726,813	15,221	684,945
1897......	45·3	1,359	10·6	3,710	91·3	4,565	16,961	169,607	118,848	922,149	21,406	899,856

Year.	Sulphuric Acid.		Tin.				Tombac.		Zinc.			
			Ingot, Crude, Old, etc.		Salt.				Metal.		Bars, Sheets, Wire, etc.	
1893......	2,090·4	62,712	2,929	3,281,272	72·1	54,075	40·9	20,450	15,083	3,318,348	452·3	122,979
1894......	1,530·4	46,656	3,136	2,823,456	84·7	59,290	13·5	6,345	15,315	3,139,519	979·0	245,517
1895......	1,566·0	43,053	3,038	2,430,745	46·0	28,520	15·4	7,238	17,156	3,053,715	611·0	148,087
1896......	3,521·8	96,850	3,344	2,455,002	27·7	15,789	324·4	152,468	17,539	3,507,720	551·6	144,285
1897......	5,876·9	147,418	3,467	2,652,332	22·3	12,488	440·8	229,111	16,599	3,485,748	355·8	93,096

From *Statistik des Answärtigen Handels Oesterreichisch Ungarischen Zollgebiets.* Although the present monetary system of Austria-Hungary, inaugurated in 1892, has for its unit the crown, the florin of the former system still continues in use during the transitional period, and the imports, exports, and production are stated in florins in the Austrian and Hungarian statistical publications. Since 1892 the florin is accepted as equivalent in value to two crowns, the value of the gold crown being 20·26 cents in United States money, while the silver crown is equal to 17·4 cents at the United States coining rate. The present value of the florin in gold is therefore equal to 40·52 cents in United States money, which equivalent should be used for converting the values in the table into United States currency. (b) Not reported.

MINERAL EXPORTS OF AUSTRIA-HUNGARY. (*a*) (IN METRIC TONS AND FLORINS.)

(1 florin = 40·52 cents.)

Year.	Alum.		Aluminum and Alloys. Unman'fac'd		Amber.		Ammoniacal Liquor.		Antimony.				Arsenic, Arsenious Acid, and Orpiment.	
									Ore.		Regulus.			
1893......	105·1	7,885	19·5	34,920	6·7	206,760	386·2	5,793	100·3	22,066	230·4	103,680	5·3	1,416
1894......	107·0	8,829	1·4	3,010	5·7	181,200	323·0	4,845	87·9	19,338	354·6	120,564	3·5	941
1895......	59·6	5,068	0·2	350	6·9	144,000	412·6	6,189	193·5	42,570	368·9	129,115	6·4	1,688
1896......	47·0	3,995	(b)	....	6·7	185,400	603·7	9,056	218·0	47,960	440·7	143,228	26·1	6,616
1897......	69·9	5,767	0·4	690	6·2	169,200	592·2	8,883	289·4	63,668	358·8	113,022	16·5	4,072

Year.	Asbestos.				Asphalt.				Barium Sulphate.		Borax.			
	Crude.		Manufactured.		Rock and Earth.		Mastic and Bitumen.				Crude, and Boracic Acid.		Refined.	
1893......	234·4	17,996	5·8	8,000	245·7	8,034	594·9	59,490	224·1	19,802	67·7	20,310	47·6	16,184
1894......	113·2	8,416	6·3	14,000	376·1	11,858	985·6	98,560	277·7	25,879	28·1	8,430	26·7	8,010
1895......	122·2	12,569	9·7	15,000	145·3	2,915	1,183·1	29,579	391·6	21,522	30·9	9,270	14·1	3,948
1896......	48·5	4,988	9·8	20,000	134·1	2,642	1,692·2	42,304	202·7	16,733	5·7	1,767	1·5	405
1897......	56·2	4,111	18·6	30,000	102·4	1,934	2,593·0	64,825	70·2	7,308	8·6	2,580	2·2	556

Year.	Brass, German Silver, and Tombac.		Calamine and Other Zinc Ores.		Cement.		Chloride of Lime.		Chrome Ore.		Clay Wares.	
1893......	985·0	922,000	7,578	200,344	21,262	303,217	216·4	19,476	326·8	19,608	44,460	8,841,000
1894......	966·2	926,000	6,979	187,680	16,856	274,866	314·3	26,244	240·8	15,532	50,329	7,930,000
1895......	1,286·8	920,000	7,491	269,676	12,804	180,492	266·7	22,136	384·6	24,999	51,767	8,014,000
1896......	1,980·7	1,102,000	9,453	378,104	16,721	238,228	113·8	9,559	142·1	9,237	67,270	8,591,000
1897......	2,499·4	1,512,000	12,914	619,876	19,786	281,339	111·0	8,880	153·4	9,204	55,895	8,200,000

Year.	Coal and Briquettes.		Cobalt and Nickel Ores.		Coin.				Coke and Briquettes.		Colored Earths.	
					Gold—Kg.		Silver—Kg.					
1893......	648,424	4,980,779	304·9	30,490	9,450	13,840,205	86,979	4,314,772	108,576	1,482,378	2,330	88,155
1894......	629,670	4,843,901	129·1	12,007	15,266	22,455,120	102,010	4,228,926	110,460	1,504,569	2,580	95,952
1895......	640,963	4,883,839	138·6	27,720	15,452	21,886,245	47,275	2,014,565	119,051	1,613,155	2,244	86,453
1896......	658,368	5,001,742	113·1	23,186	23,926	33,388,428	182,950	7,688,706	116,608	1,550,016	1,700	64,565
1897......	701,919	5,333,127	116·7	24,507	33,316	45,810,175	114,560	4,203,750	145,056	1,976,486	1,621	60,994

Year.	Copper.								Cryolite.		Fluorspar.	
	Ore.		Crude and Old.		Bars, Sheets, Plates, etc.		Sulphate.					
1893......	13·7	411	434·5	234,630	248·0	180,127	28·7	5,740	2·9	660	221	6,463
1894......	15·3	459	254·8	129,948	469·1	319,266	48·7	9,499	15·9	2,520	70	1,829
1895......	16·6	1,328	151·1	86,127	353·8	250,426	161·9	33,999	10·9	5,585	44	1,252
1896......	12·4	992	227·8	134,402	189·2	148,717	46·9	10,553	2·3	1,235	40	1,272
1897......	0·1	8	159·3	95,580	180·5	147,478	14·5	3,263	9·8	5,110	27	962

Year.	Glass.				Glauber Salt.		Gold.					
	Remnants.		Manufactures.				Ore.		Bullion—Kg.		Old, Remnants, and Dross—Kg.	
1893......	1,604	28,070	45,040	18,296,000	570·5	17,400	73·2	5,856	847	1,278,970	44,667	224,790
1894......	1,568	27,445	47,507	19,276,000	692·1	20,097	(*b*)	....	85	82,450	38,473	227,680
1895......	1,070	18,725	50,843	23,048,000	660·8	19,163	1·1	440	306	338,620	116,911	582,480
1896......	1,638	28,669	52,752	24,293,000	2.290·8	51,543	45·5	7,280	381	445,770	131,942	581,120
1897......	2,086	36,498	50,082	22,370,000	6,910·1	138,202	36·9	5,535	239	277,240	119,336	881,275

Year.	Graphite.		Gypsum.		Iron.							
					Sulphate.		Ore.		Pig, Old, etc.		Manufactures.	
1893......	11,536	627,916	2,360	41,933	476·2	20,240	106,259	743,814	12,461	487,000	19,506	11,058,000
1894......	11,536	442,374	2,430	42,781	256·3	7,049	144,309	1,010,164	11,583	467,000	20,884	11,099,000
1895......	11,923	500,262	5,069	163,014	300·6	6,915	165,402	1,488,615	9,786	441,000	18,698	12,423,000
1896......	13,091	549,190	2,275	40,157	391·6	7,832	214,390	1,929,509	11,712	510,000	17,674	10,599,000
1897......	14,229	571,496	2,466	46,973	648·1	17,176	247,856	2,354,631	12,084	609,000	21,064	10,906,000

Year.	Iron and Steel. Bars, Sheets, Wire, etc.		Lead.						Lignite and Briquettes.		Limestone.	
			Ore.		Metal and Alloys.		White.					
1893......	13,699	1,298,000	1,608	112,539	232·6	35,499	180·5	64,980	6,763,181	23,844,949	28,019	140,098
1894......	10,963	989,000	1,232	86,219	242·4	35,512	132·2	47,592	6,902,494	23,412,076	38,318	191,592
1895......	9,993	1,732,000	3,758	225,618	207·8	33,197	232·7	79,118	7,143,234	23,902,409	33,479	167,398
1896......	12,428	2,175,000	3,076	215,313	272·3	47,006	171·0	54,720	7,562,721	24,320,812	23,249	116,245
1897......	17,387	3,242,000	2,438	195,056	241·1	39,657	47·5	15,793	8,108,975	26,857,181	11,373	56,867

Year.	Lime. (Hydraulic, Burned, etc.)		Litharge.		Manganese Ore.		Marble.		Meerschaum.		Millstones.	
1893......	66,543	741,966	1,310·7	275,247	3,697	91,750	2,638	79,152	74·3	272,450	2,076	332,112
1894......	63.526	698,789	1,466·3	309,030	2,099	52,509	2,823	84,684	53·6	223,050	1,853	296,400
1895......	34,698	381,675	781·7	156,340	425	14,888	2,737	82,113	45·6	196,555	1,977	296,450
1896......	44,642	491,060	597·0	119,400	701	21,740	3,595	107,838	42·1	186,018	1,831	274,605
1897......	49,399	543,387	354·7	70,940	622	20,536	2,994	89,829	42·8	167,400	1,773	265,965

Year.	Mineral Oil.				Muriatic Acid.		Nitric Acid.		Non-Precious Metals, Manufactures of (Exclusive of Iron).		Other Earths and Mineral Substances, Prepared.	
	Crude.		Refined.									
1893......	613·0	21,456	306·6	17,632	1,841	53,047	387·7	74,647	3,436	6,760,000	32,616	1,304,624
1894......	974·7	34,117	207·7	13,505	1,908	51,196	400·7	64,926	3,686	7,107,000	28,172	1,126,880
1895......	1,454·3	54,344	3,862·7	280,123	1,460	31,832	418·1	69,188	3,807	6,854,000	58,853	2,354,108
1896......	1,814·7	51,811	23,106·2	993,567	1,246	25,090	359·8	59,520	4,362	7,360,000	25,803	1,022,108
1897......	1,600·4	38,410	13,082·0	534,249	1,439	30,222	310·1	52,256	4,486	7,827,000	20,758	830,324

Year.	Other Non-Precious Metals, in Bars, Sheets, Wire, etc		Other Ores, Not Specified.		Ozokerite.		Peat and Peat Coke.		Porcelain Earth and Feldspar.		Potash.	
1893......	249·1	291,000	619·3	81,438	4,660	1,351,400	5,441	39,718	49,816	1,623,429	7,300	1,355,166
1894......	603·9	1,080,000	1,041·1	136,773	5,166	1,433,678	3,823	27,906	53,638	1,304,979	8,189	1,393,074
1895......	336·5	373,000	1,007·8	132,526	5,054	1,493,942	3,753	27,395	56,203	1,606,557	5,665	966,586
1896......	636·1	988,000	702·8	56,224	5,722	1,916,703	2,701	19,715	67,381	2,241,422	4,164	706,084
1897......	400·8	559,000	1,314·9	118,341	5,153	1,906,425	1,655	12,080	68,609	2,296,891	5,997	953,488

Year.	Potassium Chloride.		Potters' and Other Clays.		Precious Metals, in Wire and Sheets. Kg.		Pyrites.		Quartz and Quartz Gravel and Sand.		Quicksilver.	
1893......	1,118·0	111,800	31,307	313,066	44	17,600	3,042	36,500	9,882	72,304	414·4	953,129
1894......	627·4	48,937	33,008	330,075	16	6,400	597	8,062	26,748	199,799	509·5	881,435
1895......	1,073·9	75,173	37,667	376,665	150	39,000	383	5,173	30,213	226,032	493·3	976,734
1896......	1,026·2	71,834	41,276	417,759	129	33,540	341	4,605	32,844	246,818	569·4	1,059,084
1897......	1,004·6	90,414	46,968	469,680	29	7,540	255	3,447	31,347	236,818	529·4	1,021,742

Year.	Salt.		Saltpeter. (Chile.)				Silver.					
			Crude.		Refined.		Ore.		Bullion—Kg.		Old and Dross. Kg.	
1893......	7,678	115,175	15·4	1,926	69·6	16,704	14·4	1,152	584	34,166	115,684	264,366
1894......	6,811	102,167	121·1	13.019	52·0	11,960	0·1	7	1,630	74,980	151,740	220,800
1895......	6,554	98,318	148·7	17,102	224·1	56,025	......	....	2,130	96,915	88,770	133,750
1896......	11,811	177,170	54·9	5,765	159·7	39,925	16·9	2,080	2,360	107,380	123,220	192,280
1897......	182	2,727	61·0	6,405	120·1	24,320	......	....	4,120	165,624	116,700	171,620

Year.	Slag and Slag Wool.		Soda.		Stone.						Sulphur.	
					Paving.		Lithographic		Not Elsewhere Specified.			
1893......	45,478	309,253	931·5	55,702	36,728	532,559	9·0	1,127	226,555	1,998,965	1,507·8	79,139
1894......	58,219	395,891	666·5	39,[illegible]81	41,328	599,260	27·8	3,477	321,167	2,660,915	1,231·6	64,660
1895......	52,688	421,504	1,781·2	90,874	29,392	426,187	3·6	452	241,040	1,455,482	989·3	50,136
1896......	83,187	665,500	2,914·1	162,698	30,577	443,369	5·7	713	216,666	1,324,424	1,230·9	62,056
1897......	65,966	329,828	3,970·7	234,639	38,256	554,715	22·3	2,788	223,321	1,321,344	947·5	47,875

Year.	Sulphuric Acid.		Tin.				Zinc.					
			Ingot and Old.		Bars, Plates, Sheets, etc.		White.		Metal.		Sheets, etc.	
1893......	3,906·1	324,105	78·7	234,630	71·6	87,352	2,148·4	515,616	717·6	182,989	1,225·8	223,892
1894......	8,209·8	360,307	78·0	129,948	104·2	102,116	1,626·2	390,288	446·7	89,340	1,201·7	312,825
1895......	6,465·5	219,637	53·2	86,127	90·4	76,840	1,688·1	371,382	503·8	94,463	1,157·7	293,039
1896......	6,211·7	197,449	130·2	134,402	77·7	63,714	1,825·3	383,313	1,256·2	238,678	1,139·2	315,488
1897......	7,903·1	235,383	87·0	52,200	75·1	63,835	1,673·0	351,330	770·4	120,456	993·1	265,384

(a)From *Statistik des Answärtigen* Handels des Oesterreichisch Ungarischen Zollgebiets. (b) Not reported.

MINERAL PRODUCTION OF BOSNIA AND HERZEGOIVINA. (*a*) (METRIC TONS AND FLORINS.)

Year.	Antimony, Crude.		Chrome Ore.		Copper.		Iron, Pig.		Lead-Silver Ore.		Lignite.	
1893......	3	193	965	19,996	101	19,796	2,816	36,613	(*b*)	....	118,263	113,434
1894......	(*b*)	....	1,808	30,890	270	135,197	3,287	107,477	(*b*)	....	160,800	414,100
1895......	(*b*)	....	707	(*b*)	105	(*b*)	2,569	(*b*)	(*b*)	....	195,422	(*b*)
1896......	(*b*)	....	443	18,566	206	(*b*)	10,120	295,158	(*b*)	....	222,724	469,674
1897......	(*b*)	....	396	....	135	....	15,606	....	(*b*)	....	229,643	....
1898......	(*b*)	....	458	....	156	....	(*b*)	....	(*b*)	....	270,752	....

Year.	Manganese Ore.		Quick-silver.		Salt.		Zinc Ore.	
1893......	7,403	44,265	0·2	600	8,517	228,476	(*b*)	......
1894......	6,588	104,085	(*b*)	......	10,250	779,000	(*b*)	......
1895......	8,145	(*b*)	(*b*)	......	12,758	(*b*)	(*b*)	......
1896......	6,821	112,548	(*b*)	......	13,720	960,393	(*b*)	......
1897......	5,344	84,429	(*b*)	......	13,919	974,323	(*b*)	......
1898......	5,320	....	4·0	......	14,496	....	(*b*)	......

(*a*) From *Oesterr. Zeits. für Berg-, Hütten-, und Salinenwesen*, except the figures for 1897 and 1898, which were furnished by the "Bosnisches Bureau des K. und K. gemeinsamen Ministeriums." Besides the substances specified in the table there was also produced: In 1893: Quicksilver ore, 0·886 tons; copper ore, 2,365 tons; iron ore, 6,874 tons; antimony ore, 0·089 tons; fahlore, 0·318 tons; iron pyrites, 0·700 tons; coal, 122,396 tons; iron (pig), 45·16 tons; and salt, 85·51 tons. In 1897, iron ore, 37,095 tons. In 1898, 57,935 tons. Copper ore, in 1897, 3,487 tons; in 1898, 3,760 tons. Fahlore, in 1897, 40 tons; in 1898, 452 tons. Iron pyrites, in 1898, 3,670 tons. (*b*) Not reported.

BELGIUM.

The most important articles of mineral production in Belgium are coal, iron, lead, zinc, manganese ore and phosphate of lime. Developments in these industries are described specifically under the respective captions elsewhere in this volume. The official statistics of production, imports and exports are summarized in the following tables:

MINERAL PRODUCTION OF BELGIUM. (*a*) (IN METRIC TONS AND DOLLARS; 5 f.—$1.)

Year.	Blende.		Calamine.		Iron Ore.		Iron Pyrites.		Lead Ore.		Manganese Ore.		Totals.
1893........	7,300	$83,720	4,010	$43,440	284,465	$295,580	6,301	$9,800	67	$1,520	16,820	$40,100	$474,160
1894........	7,570	76,000	4,015	39,700	311,222	316,440	3,050	5,980	160	3,380	22,048	55,540	497,040
1895........	8,080	72,510	4,150	40,340	312,637	296,090	3,510	7,230	220	5,100	22,478	57,254	478,524
1896........	7,070	71,280	4,560	48,970	307,031	283,564	2,560	5,370	70	1,610	23,265	69,004	479,780
1897........	6,804	74,340	4,150	41,270	240,774	252,902	1,828	3,990	108	3,230	28,372	68,540	444,272

PRODUCTION OF MINERAL FUEL IN BELGIUM.

Year.	Coal.							Coke.		
	Quantity. Metric Tons.	Value.		Profit.		Number of Workmen.	Average Annual Wages.	Ovens. Active.	Number of Workmen.	Consumption of Coal. Metric Tons
		Total.	Per Ton	Total.	Per Ton					
1893........	19,410,519	$36,281,180	$1·87	$1,279,000	$0·066	116,861	$177	3,310	2,251	2,284,796
1894........	20,534,501	38,258,420	1·86	1,619,560	·078	117,103	188	3,201	2,108	2,381,896
1895........	20,457,604	38,671,540	1·89	1,659.480	·082	118,957	189	3,233	2,130	2,258,063
1896........	21,252,370	40,402,020	1·90	2,179,400	·102	119,246	196	3,555	2,415	2,709,720
1897........	21,492,446	44,134,420	2·05	3,911,350	·182	120,382	205	3,845	2,566	2,968,620

Year.	Coke—*Continued.*			Briquettes from Coal.					
	Quantity. Metric Tons.	Value.		Number of Works, Active.	Number of Workmen.	Consumption of Coal. Metric Tons	Quantity. Metric Tons.	Value.	
		Total.	Per Ton					Total.	Per Ton
1893........	1,683,702	$4,488,750	$2·67	36	(*b*)	(*b*)	1,256,265	$2,836,566	$2·26
1894........	1,756,622	4,546,138	2·59	37	(*b*)	(*b*)	1,326,226	3,095,411	2·33
1895........	1,749,109	4,810,050	2·75	38	(*b*)	(*b*)	1,217,795	2,956,806	2·43
1896........	2,004,430	5,700,599	2·84	36	1,334	1,092,340	1,213,760	2,910,596	2·40
1897........	2,207,840	7,572,891	3·43	37	(*b*)	1,129,791	1,245,114	3,112,785	2·50

METALLURGICAL PRODUCTION OF BELGIUM. (*a*) (IN METRIC TONS AND DOLLARS; 5 f. = $1.)

Year.	Iron, Crude.								
	Forge Pig.			Foundry Pig.			Bessemer Pig.		
	Quantity.	Value.		Quantity.	Value.		Quantity.	Value.	
		Total.	Per Ton		Total.	Per Ton		Total.	Per Ton
1893........	428,480	$3,855,460	$9·00	74,630	$711,760	$9·54	165,077	$1,913,620	$11·59
1894........	378,045	3,525,100	9·32	80,110	757,200	9·45	170,420	1,948,440	11·43
1895........	329,750	2,949,180	8·94	85,450	747,560	8·75	161,606	1,809,020	11·07
1896........	362,451	3,734,840	10·30	84,275	805,880	9·56	193,518	2,284,720	11·81
1897........	426,332	4,653,486	10·91	78,410	912,300	11·63	183,701	2,377,210	12·94

Year.	Iron, Crude—*Continued.*						Iron, Manufactures of.			
	Thomas Pig.			Total Pig.						
	Quantity.	Value.		Quantity.	Value.		Bar, Crude.		Rails.	
		Total.	Per Ton		Total.	Per Ton				
1893........	77,077	$729,660	$9·47	745,264	7,210,500	$9·68	135,193	$3,072,320	1,616	$39,560
1894........	190,022	1,934,880	10·18	818,597	8,165,620,	9·97	107,881	2,508,080	1,285	32,300
1895........	252,428	2,536,020	10·05	829,234	8,041,780	9·65	76,101	1,718,760	525	12,820
1896........	307,779	3,336,580	10·84	959,414	10,316,180	10·75	81,394	2,030,500	1,027	27,720
1897........	333,958	4,001,380	11·98	1,035,037	12,144,076	11·73	108,608	2,801,200	1,443	40,400

Year.	Iron, Manufactures of—*Continued.*						Lead, Crude.		Silver—Kg.	
	Sheet and Plate.		Wrought.		Other Manufactures.					
1893........	111,986	$3,428,180	2,448	$140,680	233,778	$5,693,880	12,006	$615,120	26,717	$691,080
1894........	118,596	3,545,520	1,236	70,420	224,292	5,361,540	14,120	704,620	28,961	605,220
1895........	109,209	3,189,728	741	56,390	259,323	6,168,140	15,573	840,760	31,543	686,000
1896........	112,597	3,365,698	851	50,240	298,163	7,326,770	17,222	1,029,980	28,509	637,900
1897........	100,252	3,148,800	872	47,800	263,644	6,834,000	17,023	1,101,760	30,073	631,422

Year.	Steel.									
	Ingots, Cast and Blooms.		Rails.'		Tires.		Sheet.		Forge.	
1893........	273,113	$4,585,740	104,496	$2,250,480	7,648	$303,360	64,782	$1,704,520	6,132	$209,320
1894........	405,661	6,602,140	113,661	2,326,300	9,769	328,360	166,981	3,829,680	5,627	170,060
1895........	454,619	6,885,200	122,357	2,508,160	7,359	239,790	179,249	3,916,260	4,551	141,830
1896........	598,974	10,102,490	147,183	3,174,780	10,497	367,670	268,009	6,070,020	6,702	204,910
1897........	616,541	11,104,800	136,911	3,295,800	10,870	427,200	272,839	6,748,400	23,104	586,400

Year.	Steel—*Continued.*						Zinc.			
	Plates.		Wire.		Total.		Crude.		Sheet.	
1893........	30,614	$1,032,900	10,249	$273,080	224,922	$5,773,660	95,665	$7,920,420	32,388	$3,030,140
1894........	37,980	1,166,280	8,300	211,420	341,318	8,040,100	97,041	7,233,360	31,724	2,560,260
1895........	42,444	1,349,144	11,987	308,670	367,947	8,483,854	107,664	7,699,340	34,081	2,688,400
1896........	64,653	2,216,640	22,267	591,820	519,311	12,625,840	113,361	9,182,440	36,238	3,244,610
1897........	64,366	2,354,000	19,567	558,400	527,617	13,965,600	116,067	9,936,090	37,011	3,450,710

QUARRY PRODUCTION OF BELGIUM. (*a*) (VALUES IN DOLLARS; 5 f. = $1.)

Year.	Building Stone—Cubic Meters.						Chalk, Marl. Cubic Meters.		Dolomite. Cubic Meters.		Feldspar. Cubic Meters.		Flagstones. Square Meters	
	Conglomerate.		Freestone.		Limestone.									
1893....	(*b*)	....	144,795	$2,886,800	2,500,827	$1,898,200	46,295	$28,400	3,500	$1,400	1,800	$3,200	90,780	$52,800
1894....	(*b*)	....	135,094	2,049,100	2,586,305	2,089,000	83,700	25,140	(*b*)	......	2,150	3,560	96,109	52,800
1895....	245	$6,680	137,353	2,385,060	2,488,840	2,135,940	100,160	31,200	(*b*)	......	1,900	1,560	95,137	29,640
1896....	160	4,000	152,420	2,742,300	2,646,305	2,143,500	191,100	66,400	21,500	7,200	2,000	3,200	131,400	91,720
1897....	220	5,500	181,746	3,076,924	3,010,877	2,730,730	204,600	90,680	52,720	13,908	1,100	1,940	107,372	71,848

Year.	Flint for Earthenware. Cubic Meters		Flint and Gravel for Ballast. Cubic Meters.		Limestone for Flux. Cubic Meters.		Marble. Cubic Meters.		Ochers. Cubic Meters.		Paving Stones. *Pieces.*		Phosphate of Lime Cubic Meters.	
1893....	32,850	$40,400	7,280	$3,800	166,425	$70,200	13,147	$506,800	880	$1,400	96,041,650	$1,714,000	331,230	$881,000
1894....	28,625	34,860	6,640	3,180	77,900	44,980	11,849	383,760	250	1,000	84,309,000	1,441,660	371,776	879,300
1895....	24,870	20,000	202,590	103,200	163,800	87,400	12,790	424,240	500	2,000	92,378,800	1,575,160	506,730	683,140
1896....	23,450	19,160	244,050	123,140	164,900	75,360	16,315	544,680	700	2,800	102,295,950	1,793,720	297,470	537,320
1897....	23,050	17,720	235,495	110,819	225,300	89,085	17,797	471,954	350	1,450	95,542,700	....	350,056	436,792

Year.	Plastic Earth. Metric Tons.		Sand. Cubic Meters.		Slate.			Sulphate of Baryta. Metric Tons.		Whetstones and Hones—*Pieces.*	
					Pieces.	Cubic Meters	Total Value.				
1893....	192,262	$287,200	389,970	$130,800	32,508,500	615	$212,800	41,500	$58,200	100,000	$9,200
1894....	172,010	241,160	374,700	141,640	32,011,000	475	221,540	40,000	56,000	87,000	7,350
1895....	195,485	272,940	381,170	158,120	33,652,000	690	251,720	32,750	45,860	70,000	11,120
1896....	83,020	84,500	418,720	162,960	35,980,000	1,150	269,500	25,000	35,000	45,850	6,820
1897....	270,715	359,952	559,141	237,196	41,422,000	1,445	335,780	23,000	32,200	43,150	16,740

(*a*) From *Statistiques des Mines, Minières, Carrières, et Usines Métallurgiques*, by Emil Harzé. Belgium also produced, in 1896, manganese pig-iron, 11,391 tons, $154,160; kaolin, 130,960 cu. m., $213,560; and in 1897, manganese pig-iron, 12,636 tons, $199,700. (*b*) Not stated in the report.

MINERAL IMPORTS OF BELGIUM. (*a*) (IN METRIC TONS AND DOLLARS; 5 f. = $1.)

Year.	Ashes.		Cement.		Chemical Products.			Coal.		Coal Briquettes.	
					Soda Salts.		Not Specified.				
1893.........	4,285	$17,139	....	....	147,254	$10,594,780	$3,927,768	1,288,640	$3,865,920	5,545	$17,467
1894.........	4,650	18,599	23,736	$142,413	175,704	13,521,762	4,269,520	1,377,099	4,227,418	4,317	13,814
1895.........	6,173	24,694	23,198	139,190	184,306	14,630,218	4,390,884	1,530,364	4,713,521	3,452	11,048
1896.........	6,747	26,988	30,565	183,392	194,202	11,989,011	5,438,285	1,693,376	5,249,466	1,561	5,151
1897.........	10,870	43,481	17,681	113,160	181,676	7,419,315	6,048,423	2,017,344	6,455,501	632	2,212

Year.	Coke.		Copper and Nickel.					Glass and Glassware.		
			Crude.		Hammered, Drawn or Rolled.		Wrought.	Common (Bottles, Broken Glass, etc.)		All Other Kinds.
1893.........	287,560	$983,455	6,422	$1,541,285	1,009	$308,892	$118,586	4,150	$106,726	$380,834
1894.........	326,188	1,154,706	9,600	2,112,060	926	268,497	155,098	4,728	126,810	429,615
1895.........	362,834	1,327,972	10,480	2,431,424	926	268,462	168,720	6,078	187,035	441,358
1896.........	260,273	976,024	15,506	3,597,466	1,109	321,718	188,931	6,980	213,380	541,225
1897.........	269,606	1,159,306	14,821	3,556,943	1,418	405,576	193,242	4,699	143,130	664,176

Year.	Gold (Including Platinum).						Guano.		Iron and Steel Filings.	
	Ore—Kg.		Unwrought—Kg.		Coin.	Jewelry, etc.				
1893.........	2,037	$13,444	1,090	$750,792	$1,241,860	$562,970	31,790	$1,462,329	263·3	$2,106
1894.........	....	....	1,389	956,743	2,310,740	522,884	27,810	1,223,650	23·9	191
1895.........	65	429	2,714	1,869,403	616,280	523,491	43,017	1,806,725	61·6	492
1896.........	93	614	4,923	3,390,962	599,540	757,507	25,946	1,037,846	11·1	93
1897.........	....	....	3,824	2,633,971	1,726,700	701,535	5,162	185,825	26·4	227

Year.	Iron.									
	Ore.		Pig.		Old.		Hammered, Drawn or Rolled.		Wrought.	
1893.........	1,684,679	$3,200,891	161,767	$1,882,635	27,537	$330,447	20,826	$553,078	4,473	$327,482
1894.........	1,942,883	3,691,478	228,598	2,404,880	24,704	296,443	21,007	566,201	4,542	336,023
1895.........	1,857,624	3,529,485	225,037	2,326,123	22,817	273,809	17,616	462,134	4,922	362,567
1896.........	2,069,676	3,932,385	324,389	3,897,069	53,802	688,668	22,812	646,150	6,118	469,081
1897.........	2,544,377	4,834,316	302,193	3,815,918	51,985	670,608	26,054	792,873	6,336	491,296

Year.	Lead.			Lime.		Petroleum.			
	Pig.		Manufactures.			Crude.		Refined.	
1893........	38,028	$1,901,421	$37,963	29,460	$58,920	6,327	$88,574	145,139	$4,063,881
1894........	43,491	2,087,566	97,081	19,209	37,458	3,718	52,055	142,429	3,703,151
1895........	45,594	2,370,874	21,097	9,083	17,893	1,824	25,540	159,980	5,759,266
1896........	35,221	1,937,173	17,231	11,522	22,699	95	1,336	158,979	5,405,275
1897........	43,840	2,630,429	91,580	13,184	26,236	988	13,831	149,501	4,843,844

Year.	Pottery (*e*).						Resins and Bitumens, Not Specified.		Salt.			
	Terra Cotta. *Pieces.*		Common.		Earthenware.	Porcelain			Crude.		Refined.	
1893........	33,339	$200,032	2,418	$120,896	$206,809	$131,167	185,466	$9,273,289	84,524	$507,145	39,561	$395,612
1894........	39,113	234,677	2,243	112,169	237,003	145,262	194,015	9,700,769	92,215	553,288	44,414	444,143
1895........	40,531	243,185	2,344	117,216	254,547	155,283	226,133	11,306,672	81,188	487,126	40,625	406,245
1896........	*b*85,486	629,221	2,065	103,231	*c*272,894	*d*177,207	216,278	10.813,881	92,408	554,449	38,785	387,852
1897........	*b*86,493	549,631	2,115	105,756	*c*298,535	*d*188,434	237,570	11,878,487	96,805	580,827	39,193	391,926

Year.	Silver.						Steel.					
	Ore—Kg.		Bullion—Kg.		Coin.	Jewelry, etc.	Ingot.		Bars, Sheets, and Wire.		Wrought.	
1893........	1,269,590	$152,351	8,482	$220,532	$8,321,760	$311,596	18,859	$339,459	9,210	$269,030	1,750	$437,513
1894........	1,300,333	156,040	17,471	454,246	40,920	323,802	18,518	296,287	12,040	352,063	1,028	257,119
1895........	1,297,070	155,648	7,936	206,336	2,793,680	343,840	18,405	294,478	17,582	538,880	1,136	283,031
1896........	1,476,582	177,190	8,980	233,480	6,461,840	415,967	28,435	561,744	15,263	479,341	936	233,924
1897........	2,533,237	253,324	467,851	9,357,020	*f* 2,083,040	460,244	25,370	518,271	25,869	834,766	1,147	286,415

Year.	Stone.									Sulphur.	
	Roofing Slate. *1,000 Pieces.*		Building Stone, Including Marble and Alabaster.		Cut, Polished, etc.	Paving.		All Other Kinds.			
1893........	37,326	$149,305	29,230	$584,591	$112,314	7,660	$22,980	$12,759	$140,345	17,838	$642,156
1894...	39,504	158,018	29,546	590,916	59,684	2,281	6,843	75,502	830,525	24,982	899,337
1895........	37,720	150,879	31,156	623,114	53,450	4,249	14,023	83,569	919,263	18,438	590,029
1896........	38,209	152,834	40,511	891,237	61,769	6,163	22,187	81,360	894,964	14,399	403,177
1897........	38,754	162,765	47,929	1,054,432	111,313	13,197	47,509	182,950	1,097,417	13,261	371,307

Year.	Tin.			Tin Plate.			Zinc.			Ores, Crude, Not Elsewhere Specified.
	Ingot.		Manufactures.	Unwrought.		Wrought.	Ingot.		Manufactures.	
1893........	3,100	$1,456,839	$1,426	2,411	$183,251	$18,897	5,833	$501,632	$9,237	$8,480,880
1894........	4,724	2,220,253	4,245	1,919	145,819	18,781	9,130	675,633	10,554	9,801,923
1895........	3,216	1,286,230	3,501	2,046	151,511	23,712	8,551	615,643	13,095	8,361,581
1896........	4,617	1,662,044	2,845	3,203	243,403	27,425	20,182	1,614,588	11,230	10,227,349
1897........	1,609	547,118	2,634	3,875	294,533	22,972	16,320	1,387,218	10,661	13,048,392

(*a*) From Statistique de la Belgique: Tableau General du Commerce avec les Pays Etrangers, Brussels. (*b*) In metric tons. (*c*) Corresponding in 1896 to 3,668 metric tons, and in 1897 to 3,713 metric tons. (*d*) Corresponding in 1896 to 824 metric tons, and in 1897 to 815 metric tons. (*e*) There was imported in 1897 also slabs of pottery for paving, building, etc., 7,755 metric tons, $172,795. (*f*) Corresponding to 52,076 kg.

MINERAL EXPORTS OF BELGIUM. (*a*) (IN METRIC TONS AND DOLLARS; 5 f. = $1.)

Year.	Ashes.		Cement.		Chemical Products.			Coal.		Coal Briquettes.	
					Soda Salts.		Not Specified.				
1893........	255	$1,020	....	....	38,668	$2,767,447	$5,915,821	4,849,887	$14,549,661	489,225	$1,541,059
1894........	776	3,105	195,533	$1,173,197	44,678	3,470,711	6,235,984	4,539,525	13,936,342	573,463	1,835,082
1895........	1,030	4,121	273,807	1,642,845	55,089	3,965,041	5,555,916	4,661,477	14,357,349	459,702	1,471,046
1896........	1,084	4,338	277,615	1,665,690	42,857	2,298,603	6,819,396	4,649,799	14,414,377	459,974	1,517,914
1897........	2,675	10,701	322,024	2,060,951	59,054	2,202,874	7,236,116	4,448,544	14,236,341	615,974	2,152,759

Year.	Coke.		Copper and Nickel.							Glass and Glassware.		
			Crude.		Hammered, Drawn or Rolled.		Wrought.	Foreign Coin. Kg.		Common (Bottles, Broken Glass, etc.).		Plate.
1893	941,663	$3,220,487	3,473	$833,449	2,007	$614,140	$109,640	2,326	$2,326	3,824	$55,565	$2,361,408
1894	879,278	3,112,644	4,936	1,085,995	1,856	538,237	164,448	494,422	494,422	3,620	33,066	2,582,004
1895	870,983	3,187,798	5,205	1,207,508	2,558	741,944	129,892	676	676	4,181	42,923	2,768,243
1896	863,067	3,236,501	11,700	2,714,477	2,073	601,277	168,524	94	94	3,647	26,574	3,468,668
1897	909,486	3,910,789	9,994	2,398,659	1,996	570,761	198,665	266	266	3,546	26,468	3,761,219

Year.	Glass—*Continued.*		Gold (Including Platinum).				Guano.		Iron and Steel Filings.	
	All Other Kinds.		Unwrought—Kg.		Coin.	Jewelry, etc.				
1893	158,498	$7,485,843	714	$491,803	$591,480	$32,479	22,359	$1,028,502	1,847	$14,771
1894	160,319	7,468,692	375	258,300	29,760	45,569	17,820	784,068	2,217	17,738
1895	156,304	7,400,444	893	615,098	3,547,640	77,426	18,968	796,644	1,488	11,904
1896	178,611	12,194,434	3,713	2,557,514	2,666,620	75,570	14,633	585,308	7,025	59,007
1897	174,232	11,448,839	2,547	1,754,374	605,120	126,385	14,044	505,585	534	4,595

Year.	Iron.									
	Ore.		Pig.		Old.		Hammered, Drawn or Rolled.		Wrought.	
1893	171,120	$672,127	38,380	918,445	9,436	$113,229	271,743	$6,758,620	45,792	$2,467,668
1894	258,529	491,205	34,386	906,266	8,491	101,894	279,891	7,043,983	33,375	2,349,299
1895	325,809	619,036	33,237	912,301	12,519	150,228	271,066	6,697,454	39,985	2,983,007
1896	389,235	739,546	38,465	1,132,480	18,416	235,725	343,072	9,072,611	44,952	3,497,287
1897	410,817	780,552	42,554	1,325,352	17,590	226,911	356,835	10,097,254	44,521	3,501,959

Year.	Lead.			Lime.		Petroleum.			
	Pig.		Wrought.			Crude.		Refined.	
1893	29,947	$1,497,358	$11,036	404,039	$812,247	1,345·0	$18,835	25,761	721,316
1894	34,690	1,665,122	16,037	411,803	803,017	1,403·0	19,639	26,500	689,006
1895	39,996	2,079,800	18,192	436,635	860,172	2,312·0	32,364	29,239	1,052,591
1896	31,366	1,725,144	36,821	477,213	940,110	1·6	23	29,321	996,913
1897	35,988	2,159,293	33,286	520,588	1,035,969	1·1	15	18,088	586,067

Year.	Pottery.								Resins and Bitumens, not Specified.	
	Terra Cotta. *No. of Pieces.*		Common.		Earthenware.		Porcelain.			
1893	125,358	$752,150	3,731	$186,527	4,636	$927,144	338·9	$298,226	58,447	$2,922,338
1894	101,149	606,891	3,833	191,664	5,724	1,144,827	565·5	497,647	62,197	3,251,887
1895	127,413	764,481	3,463	173,140	4,048	809,509	400·7	352,589	76,087	3,804,331
1896	*b*302,526	1,913,054	2,628	131,394	4,692	513,950	586·7	54,332	86,906	4,345,315
1897	*b*294,815	1,773,837	3,197	159,867	4,905	534,532	264·0	45,395	92,591	4,629,538

Year.	Salt.				Silver.						
	Crude.		Refined.		Ore—Kg.		Bullion—Kg.		Coin.		Jewelry, etc.
1893	556	$3,339	149	$1,484	....	....	45,486	$1,182,636	37,623	$1,504,920	48,668
1894	871	5,224	158	1,579	47,260	$5,671	70,145	1,823,770	32,244	1,289,760	35,597
1895	2,136	12,814	117	1,172	19,400	2,328	45,299	1,177,774	48,326	1,933,040	51,452
1896	1,434	8,604	129	1,287	19,400	2,328	40,118	1,043,068	16,696	667,840	137,137
1897	493	2,959	231	2,309	423,160	42,316	57,933	1,158,660	521	20,851,320	189,959

Year.	Steel.						Stone.				
	Ingot.		Bars, Sheets and Wire.		Wrought.		Roofing Slate. *1,000 Pieces.*		Building Stone, Including Marble and Alabaster.		Cut, Polished, etc.
1893	1,332	$23,983	91,290	$2,132,521	21,908	$5,476,943	13,882	$108,280	237,878	$1,614,168	$727,617
1894	659	10,541	120,768	2,604,402	34,839	8,709,782	11,395	88,880	130,754	889,130	687,326
1895	1,315	21,038	170,328	4,052,108	27,438	5,508,067	15,237	106,657	147,679	1,004,216	645,690
1896	1,145	23,272	179,873	4,597,877	38,306	7,843,777	15,435	111,130	161,298	1,290,387	922,147
1897	1,201	22,302	183,386	4,842,221	37,353	7,764,848	17,304	138,428	187,180	1,591,033	934,286

Year.	Stone—*Continued.*				Sulphur.		Tin.		
	Paving.		All Other Kinds.				Ingot.		Manufactures.
1893	153,112	$918,672	721,783	$2,887,133	4,609	$165,916	247	$116,051	$574
1894	152,720	916,323	736,611	2,946,445	3,550	127,811	1,194	561,283	2,566
1895	184,838	889,930	730,863	2,923,451	4,576	146,423	1,051	420,453	1,176
1896	154,737	1,145,052	796,231	3,184,926	5,335	149,371	1,055	379,732	878
1897	153,504	1,197,335	773,531	3,248,832	6,041	169,157	347	118,089	889

Year.	Tin Plate.			Zinc.			Ores Not Specified.
	Unwrought.		Wrought.	Ingot.		Manufactures.	
1893	662.1	50,317	$213,295	75,828	$6,521,191	$93,573	$6,125,735
1894	473.4	35,975	210,937	81,248	6,012,364	54,486	3,717,273
1895	1,750.2	133,016	21,693	88,316	6,358,748	54,210	4,161,023
1896	3,952.3	309,625	11,750	100,369	8,029,501	56,349	4,236,703
1897	1,191.	90,545	15,216	100,228	8,519,380	90,749	6,216,589

(*a*) From *Statisque de la Belgique: Tableau General du Commerce avec Pays Etrangers.* (*b*) Metric tons.

CANADA.

The mineral statistics of the Dominion of Canada as collected by the Geological Survey and the Bureaus of Mines of the various provinces, are summarized in the following tables:

MINERAL PRODUCTION OF THE DOMINION OF CANADA. (*a*) (IN METRIC TONS AND DOLLARS.)

Year.	Arsenic.		Asbestos.		Barytes.		Bricks. *Thousands.* (*b*)		Building Material.		
									Stone.	Flagstones. *Sq. Ft.*	
1894....	6·4	$420	6,922	$420,825	980	$2,830	(*q*)	$1,800,000	$1,200,000	152,700	$5,298
1895....	*Nil.*	....	7,943	368,175	(*q*)	....	*c*308,836	1,670,000	1,095,000	80,005	6,697
1896....	*Nil.*	....	11,113	429,856	131	715	(*q*)	1,600,000	1,000,000	(*q*)	6,710
1897....	*Nil.*	....	27,617	445,368	518	3,060	(*q*)	1,600,000	*b*1,000,000	(*q*)	7,190
1898....	*Nil.*	....	21,577	486,227	971	5,258	(*q*)	....	*c*3,600,000	(*q*)	4,250

Year.	Building Material—*Continued.*				Cement. *Barrels.*		Fireclay.		Pottery	Tile and Sewer Pipe. (*d*)	Coal.		Coke. (*e*)	
	Granite.		Marble.											
1894....	14,877	$109,936	*Nil.*	......	108,142	$144,637	489	$2,167	$162,144	$450,325	3,490,039	$7,429,468	52,657	$148,551
1895....	17,453	84,838	181	$2,000	128,294	173,675	1,206	3,492	151,588	467,045	3,155,533	6,739,153	48,404	143,047
1896....	16,980	106,709	203	2,405	149,090	201,651	764	1,805	163,427	378,875	3,398,091	7,226,462	45,014	110,257
1897....	9,385	61,934	(*q*)	......	205,213	275,273	1,921	5,759	129,629	389,250	3,434,756	7,303,597	55,044	176,457
1898....	(*q*)	73,573	(*q*)	......	250,209	397,580	*b*1,969	5,000	135,000	166,421	3,785,408	8,227,958	65,721	219,200

Year.	Copper, Fine. (*f*)		Feldspar. (*g*)		Gold—Kg. (*h*)		Graphite.		Grindstones.		Gypsum.	
1894....	3,509	$739,659	*Nil.*	....	1,698	$1,128,688	63	$223	3,408	$32,717	202,877	$202,031
1895....	3,987	945,714	......	$*i*2,545	3,135	2,083,674	199	6,150	3,153	31,932	205,187	202,608
1896....	4,260	1,021,960	890	*i*2,633	4,145	2,754,774	126	9,455	3,368	33,310	187,818	178,061
1897....	6,033	1,501,660	1,270	3,290	9,380	6,027,016	396	16,240	4,148	42,340	217,322	244,531
1898....	8,146	2,159,556	2,268	6,250	(*q*)	13,700,000	(*q*)	11,098	(*q*)	39,465	198,908	230,440

Year.	Iron Ore.		Chrome Iron Ore.		Pyrites.		Iron, Pig. (*j*)		Lead (in Ore). (*k*)	
1894....	99,783	$226,611	907	$20,000	36,766	$121,581	45,327	$646,447	2,587	$187,536
1895....	93,257	238,070	2,882	41,301	31,024	102,594	47,586	696,440	7,467	532,738
1896....	83,377	191,557	2,124	27,004	30,586	101,155	(*q*)	....	10,977	721,159
1897 ...	50,000	130,290	2,392	32,474	35,299	116,730	(*q*)	....	17,698	1,396,853
1898....	52,763	152,510	1,833	24,252	29,228	128,872	(*q*)	....	14,477	1,206,399

Year.	Lime—*Bushels.* (*b*)		Limestone for Flux.		Manganese Ore.		Mica.	Nickel (in Ore). Kg.		Ochers.	
1894....	(*q*)	$900,000	31,843	$34,347	67	$4,180	*l*$45,581	2,225,995	$1,870,958	554	$8,690
1895....	5,225,000	*c*700,000	31,370	32,916	113	8,464	65,000	1,763,823	1,360,984	1,215	14,600
1896....	(*q*)	650,000	33,985	36,140	112	*g*3,975	60,000	1,540,920	1,188,990	2,043	16,045
1897....	(*q*)	650,000	27,371	30,258	14	1,166	76,000	1,813,321	1,399,176	3,542	23,560
1898....	(*q*)	....	30,766	31,153	45	1,600	117,598	2,502,808	1,820,838	2,124	18,600

Year.	Petroleum, Crude. *Barrels.* (*m*)		Phosphate (Apatite).		Platinum.	Precious Stones.	Roofing Cement.		Sand and Gravel (Exports).		Molding Sand.	
1894....	829,104	$835,322	5,224	$41,166	$950	$1,500	739	$3,978	294,526	$86,940	5,637	$12,428
1895....	726,138	1,086,738	1,653	9,565	3,800	(*q*)	(*q*)	3,153	251,440	118,359	6,137	13,530
1896....	726,822	1,155,647	517	3,420	750	(*q*)	78	430	203,909	80,110	5,207	11,478
1897....	709,857	1,011,546	824	3,984	1,600	(*q*)	(*q*)	....	118,768	76,729	4,976	10,931
1898....	700,790	981,106	665	3,665	(*q*)	(*q*)	(*q*)	....	(*q*)	....	9,591	21,038

Year.	Salt.		Silver—Kg. (*n*)		Slate.		Soapstone.		Terra Cotta. (*o*)	Various Products. (*p*)	Whiting. *Barrels.*	
1894....	51,890	$170,687	26,366	$534,049	(*q*)	$75,550	831	$1,640	$65,600	$703,794	500	$750
1895....	47,515	160,455	55,229	1,159,166	(*q*)	58,900	431	2,138	195,123	801,080	(*q*)	....
1896....	39,880	169,693	99,696	2,149,503	(*q*)	53,370	372	1,230	83,855	647,997	(*q*)	....
1897....	46,583	225,730	172,891	3,323,395	(*q*)	42,800	142	350	155,595	336,197	(*q*)	....
1898,....	51,839	248,639	137,911	2,583,298	(*q*)	40,791	(*q*)	....	167,902	491,660	(*q*)	....

(*a*) From *Reports Compiled by the Geological Survey of Canada.* (*b*) Estimated. (*c*) Includes building stone, brick, tile, lime, sand and gravel, estimated as for previous year. (*d*) Value of tile in 1894, $200,000, sewer pipe, $250,325; in 1895, tile, $210,000, sewer pipe, $257,045; in 1896, tile, $225,000, sewer pipe, 153,875; in 1897, tile, $225,000, sewer pipe, $164,250. In 1898 sewer pipe is alone reported. (*e*) Oven coke production of Nova Scotia and British Columbia. (*f*) Copper contents of ore, matte, etc., at the following values per lb.: 1894, 9⅜c.; 1895, 10⅜c.; 1896, 10$\frac{2}{10}$c.; 1897, 11$\frac{3}{10}$c.; 1898, 12$\frac{3}{10}$c. (*g*) Figures for 1896 includes quartz, 9 tons, $50. (*h*) Fine ounces calculated at $20·67 per oz., or $664·60 per kgm. (*i*) Export returns. (*j*) In 1894 there were 98,766 metric tons of native iron ore ($223,861) converted into pig iron; in 1895, 84,558 metric tons ($218,336). (*k*) Lead contents of ores at the following values per lb.: 1894, 3·3c.; 1895, 3·25c.; 1896, 3c.; 1897, 3·6c.; 1898, 3·78c. (*l*) Exports, plus quantity sold to Canadian electrical and stove manufacturers. (*m*) Calculated from the inspection returns at 100 gals. crude to 38 gals. refined. The value of the crude per bbl. of 35 imp. gals. was, in 1894, $1·00¼; in 1895, 1·49⅜; in 1896, $1·59: in 1897, $1·42; in 1898, $1·40. (*n*) Silver contents of ores, values for production and exports per oz.: 1894, $0·63; 1895, $0·653; 1896, $0·67; 1897, $0·598; 1898, $0·583. (*o*) In 1894, includes porous fireproof terra cotta and other structural and ornamental forms. (*p*) In 1894, includes natural gas ($313,754), mineral waters, 767,402 gals. ($100·040), lithographic stone, 163 metric tons ($30,000); in 1895, natural gas ($423,032), mineral waters, 739,382 gals. ($126,048), lithographic stone ($2,000); in 1896, natural gas ($276,301), mineral waters, 706,872 gals. ($111.736), tripoli, 602 metric tons ($9,960); in 1897, mercury, 312 kg. ($324), natural gas ($325,873); also sundry minerals, estimated in part and including actinolite, graphite, manganese, soapstone and tripolite; in 1898, natural gas ($320,000), mineral waters ($155,000), tripolite, 922 metric tons ($16,660). (*q*) Not reported.

MINERAL IMPORTS OF THE DOMINION OF CANADA. (*a*) (IN METRIC TONS AND DOLLARS.)

Year. (*h*)	Alumina.		Asphalt.		Brass. (*f*)	Bricks and Tiles.	Brimstone.		Buhrstones. *Number.*		Cement.	Chalk.
1894..	(*i*)	$1,686	8,215	$111.441	$453,671	$138,727	2,647	$61,558	2,042	$3,029	$284,471	$11,308
1895..	33	3,248	2,789	41,817	427,454	123,461	2,222	56,965	1,612	2,172	251,926	10,267
1896..	7	7,537	3,843	33,875	477,279	193,594	3,145	63,973	1,572	2,049	255,029	9,075
1897..	8	5,717	312	9,012	457,342	83,292	3,932	87,712	1,499	1,827	260,842	11,826
1898..	13	2,011	6,006	55,164	560,014	74,587	17,248	373,786	889	1,813	376,315	14,362

Year. (*h*)	Chloride of Lime.		Cryolite.		Clays.	Coal, Anthracite. (*b*)		Coal and Coke.		Coal Tar. *Barrels.*		Coin and Bullion.
1894..	1,236	$54,182	16	$94	$70,731	1,389,387	$6,354,040	1,405,656	$3,515,845	9,609	$17,229	$4,023,079
1895..	1,649	63,486	3	386	62,730	1,274,019	5,350,627	1,515,309	3,523,042	19,200	36,581	4,576,620
1896..	1,565	58,994	21	2,699	62,984	1,428,255	5,667,096	1,642,462	3,556,593	18,467	31,209	5,296,319
1897..	1,361	49,368	16	2,106	59,386	1,322,049	5,695,168	1,604,860	3,313,826	23,661	36,942	4,676,094
1898..	1,765	55,987	17	2,315	72,795	1,325,139	5,847,685	1,735,888	3,225,151	26,702	35,194	4,390,844

Year.	Copper. (*f*)	Earthenware and China.	Emery (Wheels and Ground).	Flint and Stones.		Fuller Earth.		Glass.	Gold and Silver. (*f*)	Gravel and Sand.		Gunpowder.
1894..	$281,801	$695,514	$19,552	254	$3,263	58	$1,566	$1,209,203	$283,645	37,723	$33,506	$112,781
1895..	258,214	547,935	22,344	235	2,705	102	1,904	1,120,986	341,476	17,789	24,779	142,355
1896..	294,410	575,498	18,840	287	3,027	(*i*)	1,834	1,104,481	340,241	17,193	24,604	136,518
1897..	270,036	595,822	11,231	475	5,244	(*i*)	1,552	1,139,764	296,143	19,330	25,222	131,562
1898..	867,452	674,874	15,478	389	5,344	(*i*)	3,330	1,024,706	297,242	29,164	43,287	138,430

Year.	Gypsum.		Iron Sand.		Iron and Steel. (*f*)	Lead. (*f*)	Lime. *No. of Bbls.*		Litharge.		Lithographic Stones.	Metal Wares.
1894..	(*i*)	$1,660	11	$863	$11,310,771	$203,644	6,766	$4,907	1,753	$28,685	$4,665	$317,145
1895..	947	960	10	1,300	9,249,749	192,046	12,008	5,748	542	32,953	10,078	301,925
1896..	547	848	15	3,003	10,952,607	239,109	10,239	7,331	481	32,817	4,964	307,201
1897..	(*i*)	772	17	1,624	8,666,497	233,125	16,108	10,529	546	34,538	6,360	292,921
1898..	1,040	1,742	34	2,999	12,691,772	335,278	12,850	9,002	519	32,904	7,791	534,565

Year.	Mineral Oils. *Gallons.*		Ochers.		Ores of Metals.		Paints and Colors.	Phosphorus.		Plaster of Paris. *Barrels.*		Potash.		Precious Stones and Jewelry.
1894..	6,666,323	$446,492	616	$18,955	16	$165	$182,270	10	$9,367	202	$2,584	448	$45,345	$427,796
1895..	6,852,425	442,402	73	2,077	10	261	161,726	8	8,035	953	1,707	344	34,525	301,150
1896..	7,130,061	628,371	158	2,995	......	24,038	204,583	10	9,525	1,205	2,190	292	41,037	675,420
1897..	7,696,258	614,932	293	5,533	22,021	105,072	519,445	16	8,575	3,383	4,612	26	2,390	506,728
1898..	8,175,006	623,364	448	9,428	69,766	155,583	659,193	5	4,618	(*i*)	2,318	244	32,502	402,158

Year. (*h*)	Pumice Stone.	Quicksilver.		Kainite.		Sal Ammonia		Salt.		Saltpeter.		Silex.		Slate.
1894..	$4,160	17	$14,483	87	$1,642	130	$10,126	96,406	$381,636	87	$8,881	111	$1,521	$29,267
1895..	3,609	29	25,703	294	4,459	83	7,083	95,342	362,592	350	32,282	130	1,881	19,471
1896..	3,721	35	32,343	180	1,978	52	7,876	96,467	363,438	598	55,628	149	2,174	24,175
1897..	2,903	34	33,534	206	5,984	69	7,258	103,309	345,587	456	43,066	116	3,415	21,615
1898..	3,829	3	36,425	49	1,008	38	3,721	96,935	326,202	627	51,236	141	2,773	24,927

Year.	Soda.		Spelter.		Stone and Marble. (*f*)	Sulphate of Copper.		Tin. (*f*)	Ultramarine.		White and Red Lead.		Whiting.	Zinc.
1894..	11,689	$339,702	372	$35,615	$223,930	408	$29,035	$1,310,389	119	$15,973	4,925	$353,053	$26,649	$96,882
1895..	14,775	347,808	419	30,245	200,786	877	62,777	972,078	112	15,252	3,984	282,353	24,441	68,554
1896..	12,326	376,910	494	40,548	219,298	710	57,380	1,236,640	103	14,079	5,312	367,569	27,322	87,074
1897..	13,938	375,651	378	32,826	173,822	516	40,469	1,272,600	124	15,044	4,677	347,539	22,541	58,751
1898..	16,026	446,799	4	1,407	189,386	738	57,497	1,548,346	113	12,842	5,753	448,659	25,761	116,325

MINERAL EXPORTS OF DOMESTIC PRODUCE FROM THE DOMINION OF CANADA. (*a*) (IN METRIC TONS AND DOLLARS.)

Year. (*h*)	Agric. Implements.	Antimony Ore.		Asbestos.		Barytes	Brick. *Thousands.*		Cement	Charcoal.	Cinders.	* Clay, M'f's of.	Coal.	
1894..	$465,682	(*i*)	....	5,651	$339,756	......	4,502	$33,013	$1,268	$33,191	$3,941	$67	903,563	$3,321,565
1895..	663,718	(*i*)	....	7,796	493,075	$900	1,417	6,774	443	21,823	3,163	671	1,007,506	3,578,195
1896..	593,464	(*i*)	....	8,698	482,679	168	1,216	7,788	1,035	36,086	2,125	496	1,029,918	3,249,069
1897..	761,842	(*i*)	....	9,951	510,916	(*i*)	906	5,236	1,332	26,930	922	796	999,789	3,330,017
1898..	1,443,140	1,118	$15,295	16,714	510,368	(*i*)	276	1,553	609	5,181	(*i*)	343	981,696	3,273,415

Year.	Coin and Bullion.	Coke.		Copper. (*c*)		Explosives.		Fertilizers.	Glass and Glassware.	Grindstones	Gypsum, Crude.		Gypsum, Ground.	Iron and Steel, M'f's of	Lime.
1894..	310,006	2	$10	541	$88,352	35	$29,050	$31,413	$4,433	$19,330	147,337	$160,082	$14,616	$295,924	$71,171
1895..	256,571	24	108	1,722	222,657	173	98,083	32,174	4,738	12,042	145,875	156,897	24,556	308,711	83,595
1896..	207,532	17	47	1,788	194,771	120	92,763	36,137	6,881	18,853	182,217	205,641	23,332	506,946	76,451
1897..	327,298	1,692	3,802	2,933	540,439	93	76,578	36,584	7,208	15,760	163,785	183,376	18,710	522,988	56,720
1898..	1,045,923	3,275	7,341	4,203	832,546	92	74,305	46,864	7,494	18,785	163,612	193,515	2,587	606,082	48,307

Year. (*h*)	M'f'es of Metals Not Iron and Steel,	Mica, Crude and Cut.		Mica, Ground.	Mineral Oils. *Gallons.*		Nickel in Ore.		Ores.							
									Gold Quartz, etc.	Iron.		Lead. (*e*)		Manganese.	
1894..	$24,169	155	$26,553	(*i*)	68,740	$2,722	3,315	$808,790	$318,258	1,686	$9,026	1,503	$65,337	53	$4,352
1895..	18,932	335	46,894	$575	63,543	3,572	3,647	599,568	612,729	4,290	43,088	8,191	333,763	116	7,693
1896..	31,308	251	55,514	113	18,241	2,971	3,173	486,651	1,099,053	2,666	39,999	9,209	408,625	79	1,988
1897..	41,601	215	61,565	433	1,831	230	3,415	498,515	2,804,801	1,198	2,492	13,632	522,327	73	4,348
1898..	53,317	231	69,513	(*i*)	9,530	2,061	6,696	970,531	3,587,953	236	402	19,939	1,008,147	7	1,371

Year. (*h*)	Ores—*Continued.*		Phosphate of Lime.		Plumbago.		Pyrites.		Salt.		Sand and Gravel.		Slate.		Stone and Marble.	Tin Man'f's
	Silver (*d*).															
1894..	20	$423,707	4,488	$40,400	(*i*)	$38	12,867	$46,788	127	$1,280	274,664	$96,153	67	$1,390	$46,883	$6,157
1895..	35	651,737	3,800	33,810	4	291	7,664	35,452	137	1,136	273,924	90,093	201	3,964	60,405	9,531
1896..	78	1,595,548	479	5,280	94	10,771	6,706	32,893	76	767	259,585	119,847	275	8,983	49,679	5,777
1897..	127	2,613,173	21	275	78	3,584	14,219	29,403	4,702	1,168	171,057	75,134	(*i*)	......	4,983	2,764
1898..	211	3,519,786	232	1,090	348	6,428	18,752	221,564	5,559	1,139	150,270	80,952	(*i*)	......	61,030	5,578

(*a*) From the tables of the *Trade and Navigation of the Dominion of Canada.* The imports figures are for home consumption only. The exports are those of domestic produce. (*b*) Including anthracite coal dust. (*c*) Represents fine copper contained in ore, matte, regulus, etc. (*d*) Silver contents of ores. (*e*) Represents lead contained in ore. (*f*) Includes manufactures. (*g*) Represents barrels. (*h*) Fiscal years ending June 30. (*i*) Not reported.

MINERAL PRODUCTION OF BRITISH COLUMBIA. (*a*) (METRIC TONS AND DOLLARS.)

Year.	Coal. (*b*)		Coke.		Copper.		Gold—Kg.				Lead. (*c*)		Silver—Kg.	
							Lode.		Placer.					
1894	1,029,160	$3,038,859	......	....	147	$16,234	194	$125,014	......	$405,516	2,568	$169,875	23,214	470,219
1895	954,688	2,818,962	459	$2,260	432	47,642	1,221	785,271	749	481,683	7,473	532,255	46,546	977,229
1896	859,775	2,327,145	625	3,075	1,732	190,926	1,936	1,244,180	846	544,026	10,977	721,384	97,519	2,100,689
1897	896,980	2,648,562	18,117	89,155	2,415	266,258	3,301	2,122,820	798	513,520	17,618	1,390,517	170,227	3,272,836
1898	1,154,039	3,407,595	35,560	175,000	3,298	874,781	3,423	2,201,217	1,600	643,346	14,376	1,077,581	133,200	2,375,841

(*a*) From the *Annual Report of the Minister of Mines.* (*b*) The figures of production of coal from 1867 to 1892 are to be found in THE MINERAL INDUSTRY, Vol. II., p. 220. (*c*) Attention should be called to the discrepancy between certain of these figures and those of the Canadian Geological Survey; practically the total production of lead in the Dominion is from British Columbian ore.

MINERAL PRODUCTION OF NOVA SCOTIA. (*a*) (*b*) (IN METRIC TONS AND DOLLARS.)

Year.	Coal.	Coke.	Copper	Gold. Kg.	Grindstones.	Gypsum.		Iron Ore	Limestone.	Manganese Ore-		Molding Sand.	
1893......	1,709,636	52,437	1,134	436	$6,971	89,129	$95,509	60,634	(*e*)16,904	103	$10,533	(*b*)	....
1894......	2,235,438	60,592	(*b*)	621	6,581	96,317	97,529	75,761	27,215	22	1,376		725
1895......	2,122,672	42,160	(*b*)	625	17,189	120,929	(*b*)	72,236	27,375	99	(*b*)	68	....
1896......	2,271,140	59,681	10	813	30,317	131,577	122,614	57,235	31,670	131	8,798	....	....
1897......	2,358,051	45,720	(*b*)	827	32,400	126,500	(*b*)	48,852	25,300	102	(*b*)	....	....
1898......	2,317,957	42,672	(*b*)	967	38,000	133,096	....	31,547	24,384	76	....	....	....

(*a*) From the *Annual Reports of the Department of Mines of Nova Scotia.* In 1896 also produced graphite, 178 net tons. (*b*) Not stated in the official reports. (*e*) There was also produced 2,050 bbl. lime, value $2,050.

MINERAL PRODUCTION OF ONTARIO. (*a*) (IN METRIC TONS AND DOLLARS.)

Year.	Brick—*Number.*						Building Stone.					
	Common.		Pressed.				Dimension Stone. *Cubic Feet.*		Heads and Sills—*Cu. Ft.*		Coursing Stone—*Sq. Yds*	
			Plain.		Fancy.							
1892....	175,000,000	$980,000	20,342,000	$198,350	1,323,000	$32,253	2,600,000	$680,000	50,000	$26,000	64,000	$42,000
1893....	162,350,000	932,500	20,208,000	*b*217,373	1,373,700	(*c*)	1,400,000	260,000	44,700	21,000	170,000	180,000
1894....	131,500,000	690,000	22,460,000	198,510	2,896,000	34,160	1,340,000	360,470	47,070	15,900	22,000	36,000
1895....	126,245,000	705,000	15,253,370	115,695	2,312,497	24,075	(*d*)	....	(*d*)	....	(*d*)	....
1896....	105,000,000	577,000	10,774,400	88,945	1,256,600	9,900	(*d*)	....	(*d*)	....	(*d*)	....
1897....	(*g*)	....	7,148,908	53,727	895,000	9,350	(*g*)	....	(*g*)	....	(*g*)	....

Year.	Building Stone. *Continued.*			Calcined Plaster, etc. Metric Tons.		Cement—*No. of Barrels.*				Cobalt. Metric Tons		Copper. Metric Tons.	
	Rubble, etc. *Cubic Yards.*		Total Value.			Natural Rock.		Portland.					
1892....	730,000	$132,000	$880,000	*e*2,258	$35,690	54,155	$38,580	20,247	$47,417	7·7	$3,713	1,756	$232,135
1893....	410,000	260,000	721,000	*f* 145	15,000	74,353	63,567	31,924	63,848	17·2	9,400	1,298	115,200
1894....	223,000	142,000	554,370	1,036	22,697	55,323	48,774	30,580	61,060	2·9	1,500	2,493	195,750
1895....	(*d*)	....	438,000	402	13,095	55,219	45,145	58,699	114,332	(*g*)	....	2,146	160,913
1896....	(*d*)	....	394,000	635	10,250	60,705	44,100	77,760	138,230	(*g*)	....	1,695	130,069
1897....	(*g*)	....	....	(*g*)	....	84,670	76,123	96,825	170,302	(*g*)	....	2,495	200,067

Year.	Gold.			Gypsum. Metric Tons.		Lime—*Bushels.*		Mica. Metric Tons.		Natural Gas. *1,000 Cubic Feet.*		Nickel. Metric Tons.	
	Ore. Met. Tons.	Metal.											
		Kg.	Value.										
1892....	3,365	(*g*)	$36,900	3,510	$14,100	2,600,000	$350,000	6·3	$1,500	(*g*)	$160,000	1,888	$590,902
1893....	5,044	52·7	32,960	2,556	7,363	2,700,000	364,000	63·5	8,600	2,342,000	238,200	1,489	454,702
1894....	2,202	62·9	32,776	2,951	9,760	2,150,000	280,000	(*g*)	....	1,653,500	204,179	2,322	612,728
1895....	5,896	94·2	50,281	3,059	7,471	2,090,000	280,000	22·6	2,900	3,320,000	282,986	2,100	404,861
1896....	12,059	222·5	121,848	3,175	10,500	1,880,000	220,000	2·2	2,425	(*g*)	276,710	1,768	357,000
1897....	(*g*)	354·9	190,244	1,568	17,905	(*g*)	....	(*g*)	....	(*g*)	308,448	1,813	359,651

Year.	Petroleum.										
	Crude. *Imperial Gallons.* (*h*)		Refined Products. (*h*)								
			Illuminating Oils. *Imp. Gals.*		Lubricating Oils. *Imp. Gals.*		All Other Oils. *Imp. Gals.*		Paraffine Wax. Metric Tons.		Fuel Products
1892....	28,000,000	$1,000,000	10,862,894	$919,315	3,457,570	$138,304	7,654,723	$272,577	29·4	$70,289	(*g*)
1893....	34,055,000	1,099,868	13,322,320	1,372,209	4,239,847	277,500	11,220,705	323,156	102·0	143,325	$72,500
1894....	34,912,360	1,094,852	14,349,472	1,337,040	3,817,181	242,688	10,632,141	343,416	1,249·0	152,467	71,228
1895....	33,351,997	1,403,960	10,924,826	1,237,328	2,400,404	205,591	7,081,717	285,308	891·0	86,608	79,589
1896....	27,380,588	1,222,307	11,342,880	1,263,230	2,283,047	204,946	7,821,262	340,054	695·2	76,250	70,815
1897....	25,556,591	(*g*)	10,891,337	1,131,083	1,959,810	199,755	8,970,974	358,375	970·4	88,378	(*g*)

Year.	Pottery.	Salt. Metric Tons.		Sewer Pipe.	Silver Ore. Metric Tons.		Terra Cotta.	Tiles—*Thousands.*				Total Value.
								Drain.		Roofing.		
1892....	$80,000	39,360	$162,700	(*g*)	9·07	$732	$20,119	10,000	$100,000	383	$8,613	$5,374,139
1893....	115,000	43,953	149,850	$230,000	453·6	2,500	(*c*)	17,300	190,000	53	(*c*)	6,120,753
1894....	134,000	31,947	115,551	207,000	*Nil.*	....	52,360	25,000	280,000	100	1,200	6,086,758
1895....	108,000	46,275	188,101	133,159	*Nil.*	....	38,500	14,330	157,000	375	6,200	5,170,138
1896....	104,000	40,657	204,910	49,875	*Nil.*	....	24,190	13,200	144,000	170	6,800	5,235,603
1897....	(*g*)	49,611	249,880	73,551	*Nil.*	....	35,800	(*g*)	....	(*g*)	73,551	3,869,821

(*a*) From the *Annual Reports of the Bureau of Mines of Ontario*, Toronto. In 1894 there was also produced sand and gravel 733,500 cubic yards, value $203,450. (*b*) Comprising the values of plain and fancy pressed brick, roofing tile, and a quantity of ornamental and porous terra cotta. (*c*) Value included with that of plain pressed brick (*d*) Quantity not stated, and value included in the total of building stone. (*e*) Comprising 2,160 metric tons of phosphate of lime, value $23,810, and 98 metric tons of alabastine and plastico, value $11,880. (*f*) Comprising 127 metric tons of alabastine and plastico, value $14,800, and 18 metric tons of phosphate of lime, value $200. (*g*) Not reported. (*h*) One barrel of crude oil is reckoned at 35 imperial gallons, and 1 barrel refined at 42 imperial gallons.

CHILE.

There are no official statistics of mineral production in Chile. The exports and imports are summarized in the subjoined table. With respect to the most important articles of mineral production, namely, copper and nitrate of soda, the exports practically represent the production. This is also the case with respect to iodine and borate of lime.

MINERAL EXPORTS OF CHILE. (*a*) (IN METRIC TONS AND CHILEAN DOLLARS.)

Year.	Borate of Lime.		Borax.		Clay.		Coal.		Cobalt Ore.		Copper Matte.	
1893....	4,587	$229,828	(*b*)	....	1	$10	209,711	$1,677,904	1·6	$78	2,966	$296,579
1894....	6,700	335,008	(*b*)	....	(*b*)	....	205,201	1,641,608	4·6	230	342	34,220
1895....	4,425	221,263	107	$21,380	(*b*)	....	195,115	1,560,920	13·4	915	417	41,673
1896....	7,486	374,294	(*b*)	....	40	600	204,858	1,434,253	(*b*)	....	2,528	284,897
1897....	3,154	283,770	14	2,860	20	998	243,968	1,869,310	6·0	312	2,519	251,915

Year.	Copper and Silver Matte.		Copper, Silver and Gold Matte		Copper Ore.		Copper and Silver Ore.		Copper, Silver and Gold Ore—Kg.		Copper, in Bars.	
1893....	1,813	$322,034	(*b*)	....	8,299	$414,950	165·1	$16,507	(*b*)	....	19,828	$4,844,853
1894....	1,508	245,223	2·5	$503	11,106	444,227	90·3	11,027	460	$53	19,640	4,372,238
1895....	664	134,424	15·3	3,060	6,963	278,515	84·4	8,849	2,012	402	20,042	4,602,114
1896....	1,059	213,115	7·6	1,520	6,159	307,943	62·3	9,462	29,542	2,954	20,592	5,148,015
1897....	904	378,340	(*b*)	....	3,396	169,810	161·8	25,455	(*b*)	....	19,011	5,226,199

Year.	Earth for Smelting Furnaces. Kg.		Gold Bullion. Kg.		Gold Ore.		Iodine.		Iron Ore.		Lead and Argentiferous Lead, in Bars.	
1893....	5,800	$116	753·4	$602,761	168	$33,596	595	$5,953,420	(*b*)	....	73	$4,912
1894....	53,155	1,066	1,475·4	1,172,355	192	39,243	323	3,332,780	(*b*)	....	87	6,072
1895....	(*b*)	....	1,184·5	247,749	270	54,063	144	1,442,580	805	$15,255	93	4,627
1896....	(*b*)	....	1,061·3	849,036	367	73,475	206	2,063,950	(*b*)	....	594	59,415
1897....	8,200	82	1,131·7	905,168	64	28,535	243	2,429,370	5·1	256	369	66,696

Year.	Lime.		Manganese Ore.		Other Ores.	Chile Saltpeter.		Silver Ore.		Silver and Gold Ore.	
1893....	0 8	$16	36,741	$367,410	$70	947,023	$39,211,913	330	$206,996	88	$9,225
1894....	7·4	148	47,994	479,939	1,150	1,061,337	44,327,518	370	401,449	56	12,383
1895....	1·2	26	24,075	241,383	2,800	1,220.427	45,528,510	2,137	1,051,034	118	28,255
1896....	4·0	80	26,151	261,515	700	1,111,757	43,931.667	2,750	1,257,870	666	130,182
1897....	0·8	24	23,529	1,411,648	20,300	1.057,640	37,461,559	984	434,231	260	46,631

Year.	Silver, in Bars, etc. Kg.		Silver, Old. Kg.		Silver-Lead Ore.		Silver Sulphuret Ore.		Tin.	
1893....	140,369	$4,418,052	2,353	$70,587	28	$1,982	140	$175,337	(*b*)	....
1894....	153,723	3,857,481	2,175	43,556	15	1,000	127	165,098	4·2	$1,26
1895....	148,747	3,867,412	375	7,494	21	2,050	99	123,155	(*b*)	....
1896....	151,226	4,183,108	988	19,754	0·8	83	160	418,785	(*b*)	....
1897....	143,541	3,781,125	443	12,518	6·0	636	183	338,062	345	25,88

(*a*) From the *Estadística Commercial de la República de Chile*, Valparaiso. (*b*) Not reported.

MINERAL IMPORTS OF CHILE. (*a*) (IN METRIC TONS AND CHILEAN DOLLARS.)

Year.	Bitumen for Pavements.		Brass.		Cement, Roman.		Coal.		Copper.			
									Sheets.		For Various Uses	
1894....	118	$25,600	86	$41,914	12,302	$252,314	595,639	$4,752,966	37	$20,151	31	$18,366
1895....	190	38,901	82	36,776	18,236	365,550	574,063	4,578,067	18	9,780	57	15,473
1896....	254	52,198	118	50,518	14,431	326,395	683,196	5,462,954	16	9,484	54	29,853
1897....	237	55,892	60	26,206	18,205	365,129	513,421	4,122,918	55	28,333	43	23,824

Year.	Earth for Smelting Furnaces.		Gold Coin.	Iron.								Lead, Bars and Sheets.	
				Bars and Ingots		Hoops.		Sheets, Ungalv.		Sheets, Galv.			
1894....	88	$2,422	$179,595	14,397	$678,305	959	$62,012	4,113	$333,082	10,451	$1,260,622	96	$15,117
1895....	177	3,980	392,900	16,506	772,723	1,198	84,794	4,714	375,848	5,972	719,721	194	27,759
1896....	69	1,386	5,635	13,970	698,611	1,452	101,605	1,256	99,985	7,917	942,449	525	73,519
1897....	100	3,542	4,440	7,604	354,592	802	63,943	1,859	153,732	10,756	1,292,176	281	37,882

Year.	Quicksilver.		Salt.				Silver.					Soda, Caustic.	
			Common.		Refined.		Bars—Kg.		Coin.	Ore.			
1894....	142	$144,564	5,417	$39,789	109	$8,353	......	....	$89,241	66	$15,557	1,433	$85,475
1895....	106	106,277	3,192	23,042	246	12,116	17·3	$954,255	112,430	58	21,383	2,111	123,102
1896....	54	54,350	7,311	53,486	353	19,077	......	....	89,853	40	14,640	2,570	149,222
1897....	40	39,758	13,179	90,742	320	16,191	......	....	36,930	9	3,030	2,604	155,086

Year.	Steel, Bars and Sheets.		Sulphate of Copper.		Sulphur.		Tin.		Whiting and Gypsum.		Zinc.			
											Bars.		Sheets.	
1894....	3,198	$382,223	258	$36,275	2,291	$136,596	447	$61,936	522	$11,973	74	$9,714	352	45,714
1895....	7,142	857,400	134	18,855	1,679	100,961	804	113,046	796	20,674	7	944	309	55,830
1896....	3,480	422,078	111	16,643	1,775	110,742	506	71,648	1,492	47,831	3	426	315	45,451
1897....	2,225	268,181	85	10,737	2,655	157,671	608	82,149	492	13,683	24	3,314	252	50,236

(*a*) From the *Estadistica Commercial de la Republic de Chile*, Valparaiso.

CHINA.

There are no official statistics of mineral production in China. The exports and imports as reported by the Imperial Maritime Customs officials are summarized in the following tables:

MINERAL IMPORTS OF CHINA. (*a*) (IN METRIC TONS AND DOLLARS.)

Year.	Brass Wire.		Cement.		Chinaware, Fine and Coarse.		Coal.		Colors.		Copper.	
											Bar, Rod, Sheets, Plates, Nails.	
1893....	142·2	$41,060	(*b*)	....	(*b*)	....	435,803	$2,012,220	914·9	$63,429	569·1	$133,293
1894....	217·6	50,080	9,950·5	$123,694	514·0	$70,586	494,075	2,480,434	1,529·5	124,010	811·7	178,633
1895....	159·8	38,343	(*b*)	105,943	208·3	37,518	581,590	2,715,418	2,675·6	122,542	589·5	105,414
1896....	192·5	46,780	(*b*)	170,041	403·1	53,083	634,912	2,867,241	3,153·7	233,470	539·3	123,741
1897....	183·6	33,915	(*b*)	79,056	401·3	109,119	558,149	2,658,722	2,185·6	138,723	490·6	108,317

Year.	Copper—*Continued.*						Flint Stones.		Glass.		
	Slabs and Ore, Unmanufactured		Wire.		Wares, Unclassified.				Window—*Boxes.*		Wares.
1893....	2,092·1	$491,033	82·8	$26,318	98·5	$31,290	732·6	$6,663	111,812	$267,624	....
1894....	1,596·9	296,917	91·8	21,904	73·5	21,702	1,642·3	12,011	103,470	194,996	$162,117
1895....	930·3	224,237	69·9	18,915	102·4	27,930	2,261·7	17,387	80,881	165,335	185,748
1896....	2,285·4	511,674	113·8	29,985	90·4	40,589	2,535·4	33,123	111,586	281,983	278,652
1897....	2,525·8	606,602	79·9	20,743	34·4	13,975	2,652·9	30,707	138,553	335,532	238,784

Year.	Iron.											
	Bar.		Hoop.		Nail-Rod.		Old.		Pig and Kentledge.		Sheets and Plates.	
1893....	6,512·4	$251,580	930·8	$37,605	17,950	$622,115	30,313	$598,026	3,753·4	$77,049	2,316·2	$142,673
1894....	7,565·4	234,472	1,455·4	52,775	20,856	573,825	30,810	540,846	5,453·4	102,694	3,442·7	157,716
1895....	8,745·9	309,540	1,401·2	48,544	16,358	480,478	25,009	419,298	7,378·5	138,497	3,664·4	184,394
1896....	9,881·5	314,737	1,949·4	81,797	26,677	761,381	52,435	946,837	8,752·3	155,880	4,861·1	229,416
1897....	6,367·9	209,284	1,480·8	54,326	11,611·4	353,935	32,881	587,278	1,458·5	36,678	3,489·6	164,803

Year.	Iron—*Continued.*			Jadestone.		Lead.				Machinery
	Wire.		Wares, Unclassified			Pig.		Tea and Sheet.		
1893....	3,717·7	$205,575	$266,793	253·8	$147,660	10,694	$648,536	141·9	$12,247	$898,425
1894....	2,078·1	114,532	123,185	152·8	90,402	12,799	720,873	120·5	9,110	862,298
1895....	2,214·3	133,506	115,700	272·0	225,497	10,858	646,253	122·3	8,199	1,908,164
1896....	2,991·3	173,474	421,083	264·2	216,728	10,114	622,690	248·1	17,328	1,672,197
1897....	2,529·1	168,455	299,645	227·5	174,467	7,709	494,721	102·2	7,271	1,956,051

Year.	Metals, Unclassed.	Nickel.		Paints.		Petroleum—*Liters.*		Quicksilver.		Spelter.	
1893....	$288,006	(b)	....	2,501·8	$614,522	227,181,608	$5,348,347	78·1	$79,469	4,734·0	$470,781
1894....	253,727	38·6	$29,855	2,025·0	393,839	316,674,095	6,164,092	67·5	59,543	570·5	42,231
1895....	322,425	120·3	93,220	(b)	420,138	236,319,349	5,220,238	73·7	69,792	190·3	14,095
1896....	273,153	104·1	67,634	(b)	624,690	304,190,319	7,357,490	91·0	101,680	1,116·0	97,080
1897....	197,659	37·0	23,334	(b)	457,603	451,387,582	9,575,378	55·3	54,023	1,819·5	169,686

Year.	Steel.				Tin.				White Metal. (German Silver)		Yellow Metal. Bar, Rod, Sheets, and Nails.	
	Common.		Mild, or Iron Ingots.		Plates.		Slabs.					
1893....	7,382·2	$334,965	(b)	....	940·7	$66,909	4,080·6	$1,798,956	(b)	....	1,705·7	$350,532
1894....	3,575·7	212,699	3,115·9	$95,251	1,429·0	81,105	4,435·2	1,495,009	172·3	$83,125	1,537·4	243,724
1895....	2,766·0	168,062	3,573·4	118,324	3,318·0	195,276	4,590·3	1,540,592	139·7	74,793	1,569·4	236,978
1896....	5,181·9	369,762	9,758·9	220,547	1,880·7	123,901	5,516·4	1,785,541	182·1	82,953	1,719·3	305,514
1897....	3,206·9	198,455	711·4	22,223	3,543·7	209,447	4,128·9	1,396,302	241·4	122,901	1,753·6	317,187

RE-EXPORT OF FOREIGN GOODS FROM CHINA. (a) (IN METRIC TONS AND DOLLARS.)

Year.	Brass Wire.		Cement.		Chinaware.		Coal.		Colors.		Copper Ore and Manufactures.	
1893....	0·477	$174	(b)	....	(b)	....	61·381	$290,116	(b)	....	0·125	$87
1894....	(b)	....	(b)	....	(b)	....	47·950	254,031	10·8	$2,906	14·330	83
1895....	(b)	....	(b)	....	(b)	....	48·889	234,380	143·7	18,265	47·300	3,663
1896....	(b)	....	11·6	$156	1·3	$203	42·082	271,957	31·8	7,939	193·400	41,723
1807....	(b)	....	147·0	1,935	0·1	123	47·254	233,561	33·8	8,578	....	....

Year.	Glass.			Iron.							Lead.	
	Window. Boxes.		Wares.	Bar, Hoop, Nail-Rod, Sheets, Plates, and Wire.		Old.		Pig and Kentledge.		Unclassed.		
1893....	444	$1,023	(b)	755·4	$30,803	140·3	$3,188	$178·4	3,684	$2,192	280·6	$16,045
1894....	758	1,354	(b)	704·0	29,738	170·1	2,604	150·6	2,878	3,351	18·7	1,071
1895....	1,046	2,018	$1,394	1,628·8	64,041	210·1	3,743	152·0	2,815	484	80·5	4,866
1896....	222	580	2,252	1,412·3	58,783	334·5	7,197	154·4	2,524	4,544	98·4	6,890
1897....	1,024	2,580	1,992	1,691·5	69,655	258·5	5,098	1,181·6	33,774	6,340	68·6	4,981

Year.	Metals, Unclassed.	Nickel.		Paints.		Petroleum—*Liters.*		Quicksilver.		Spelter.	
1893....	$2,186	(b)	....	19·7	$9,323	4,770	$98	(b)	....	(b)	....
1894....	3,351	(b)	....	34·4	11,925	500	8	(b)	....	(b)	....
1895....	14,790	1·269	$1,005	(b)	5,506	3,262,668	72,576	8·6	$7,709	(b)	....
1896....	10,325	1·330	936	(b)	11,935	2,302,730	61,587	3·7	4,048	(b)	....
1897....	5,650	3·443	1,874	(b)	15,996	1,982,607	42,623	2·6	2,639	12·08	$1,122

RE-EXPORT OF FOREIGN GOODS FROM CHINA. *Concluded.*

Year.	Steel.		White Metal. (German Silver)		Yellow Metal.	
1893....	29·7	$2,262	(b)	....	4·8	$912
1894....	29·3	1,761	(b)	....	24·6	4,261
1895....	173·6	7,603	7·1	$3,596	13·2	2,946
1896....	377·7	15,733	(b)	....	2·1	446
1897....	591·0	20,695	1·3	711	0·2	52

EXPORT OF NATIVE GOODS FROM CHINA. (a) (IN METRIC TONS AND DOLLARS.)

Year.	Chinaware, Earthenware, and Pottery.		Glassware, Bangles, etc.		Gold and Silver Ware.	
1893...	19,195	$1,131,681	1,397·2	$378,278	2·037	$85,412
1894...	18,663	947,975	1,347·8	304,160	2·176	74,346
1895...	32,783	1,232,906	1,510·6	353,929	3·143	95,697
1896...	21,786	1,318,955	1,111·9	316,891	3·083	120,149
1897...	20,453	996,722	1,066·9	252,573	2·597	93,573

(a) From the *Returns of Trade and Trade Reports of the Imperial Maritime Customs*, Shanghai, China. In these reports the unit of quantity is chiefly the "picul," and that of value the "Haikwan-tael." In converting the original data to metric tons and dollars, the following relations were used: 1 picul = 0·060453 metric ton; in 1892, 1 Haikwan-tael = $1·07 American gold; in 1893, 96c.; in 1894, 77c.; in 1895, 80c.; in 1896, 81c., and in 1897, 72c., at the average sight exchange on New York, London, Paris, Berlin, Calcutta, and Hong-Kong respectively.
(b) Not reported.

FRANCE.

The official statistics of mineral production in France and her Colonies are summarized in the subjoined tables. With respect to the most important substances, reference should be made to the respective captions elsewhere in this volume.

MINERAL PRODUCTION OF FRANCE. (*a*) (IN METRIC TONS; 5f. = $1.)

Year.	Alunite.		Antimony Ore.		Asphaltum.		Bauxite.		Bituminous Substances. (*c*)		Copper Ore.	
1892..	(*b*)	....	5,103	$135,299	14,690	$128,702	23,298	$39,305	224,311	$335,649	251	$6,078
1893..	253	$2,530	7,171	130,069	13,608	123,165	33,923	52,249	221,063	322,441	(*b*)	....
1894..	354	3,540	6.144	81,231	14,336	129,550	26,032	32,588	230,603	351,561	(*b*)	....
1895..	330	3,300	5,396	68,004	*e*15,705	138,055	17,958	24,282	266,660	368,632	(*b*)	....
1896..	171	1,710	5,675	68,544	17,717	150,263	33,820	48,833	225,784	348,187	106	704
1897..	(*b*)	....	4,685	64,915	17,982	146,272	41,740	55,211	233,328	339,898	956	3,645

Year.	Iron Ore.		Iron Pyrites.		Lead-Silver Ore.		Manganese Ore.		Mineral Fuel.	
									Coal.	
1892..	3,706,748	$2,516,168	230,480	$572,656	21,656	$673,164	32,406	$205,074	25,697,233	$64,038,766
1893..	3,517,438	2,339,169	231,025	561,284	24,599	530,742	38,080	290,078	25,172,792	58,105,463
1894..	3,772,101	2,459,461	283,439	684,908	29,055	473,920	32,751	200,275	26,964,125	60,747,709
1895..	3,679,767	2,341,711	253,416	639,414	21,503	448,835	30,871	184,143	27,582,819	60,930,220
1896..	4,069.390	2,565,205	282,064	712,986	19,042	450,709	31,318	185,717	28,750,452	62,516,853
1897..	4,582,236	3.008,028	303,488	752,658	21,212	556,261	37,212	208,035	30,337,207	66,020,171

Year.	Mineral Fuel—*Continued.*				Petroleum, Crude.		Salt.		Sulphur Ore. (*d*)		Zinc Ore.	
	Lignite.		Peat.									
1892..	481,468	$896,665	168,445	$394,251	8,008	$191,160	973,747	$3,353,390	7,231	$20,972	69,209	$1,481,111
1893..	478,189	852,720	174,290	460,974	8,710	189,580	1,114,327	3,317,471	3.733	13,022	77,466	1,061,964
1894..	452,780	804,321	131,717	356,606	8,762	192,578	890,607	2,787,624	851	2,747	80,065	825,867
1895..	437,074	779,554	131,547	366,439	9,446	215,002	871,312	2,406,155	4,213	14,077	72,989	873,765
1896..	439,448	776,938	130,207	379,047	9,292	209,184	1,042,614	2,478,568	9,720	25,660	81,346	1,160,021
1897..	460,422	781,517	98,067	253,851	10,233	234,811	948,003	2,239,824	10,723	31,252	83,044	1,305,003

METALLURGICAL PRODUCTION OF FRANCE. (*a*) (IN METRIC TONS; 5f. = $1.)

Year.	Aluminum.		Antimony.		Copper.		Iron, Pig.		Iron, Wrought.		Steel.	
1892....	75	$125,580	754	$148,362	2,163	$566,894	2,057,258	$24,973,706	828,519	$29,533.129	682,527	$36,823,618
1893....	137	194,900	868	157,300	6,587	1,507,700	2,003,096	23,346,911	808,171	28,047,478	664,032	33,366,862
1894....	270	274,400	1,012	136,024	6,415	1,353,574	2,069,714	23,600,457	785,781	25,794,576	674,190	33,895,860
1895....	360	360,000	779	98,784	8,245	1,872,740	2,003,868	22,264,312	756,798	23,443,660	714,523	36,645,671
1896....	370	259,000	969	130.217	6,544	1.641,876	2,339,537	26,476,584	828,758	25,343,590	916,817	41,898,067
1897....	470	285,500	1,033	141,857	7,376	1,845,243	2,484,191	29,121,027	584,540	24,406,664	994,391	45,276,858

Year.	Lead. (*f*)		Litharge.		Nickel.		Gold—Kg.		Silver—Kg.		Zinc.	
1892........	8,776	$510,478	49	$2,619	1,244	$1,236,400	210	$144,060	103,247	$2,994,163	20,609	$1,996,956
1893........	8,119	404,187	33	1,617	2,045	1,237,500	300	192,000	98,077	2,550,002	22,419	1,843,649
1894..... ..	8,696	362,882	62	2,994	1,545	1,237,500	376	240,640	96,955	2,133,010	23,387	1,815,439
1895........	7,627	385,422	(*b*)	....	1,545	1,087,500	380	261,744	71,073	1,563,606	24,230	1,781,515
1896........	8,232	444,371	(*b*)	....	1,545	921,500	327	225,637	70,479	1,592,825	35,585	2,922,028
1897........	9,916	630,928	(*b*)	....	1,245	741,500	276	190,109	80,351	1,639,160	38,067	3,263,490

(*a*) From *Statistique de l'Industrié Minérale.* (*b*) Not reported. (*c*) Includes pure bitumen, bituminous schist, bituminous sand, and asphaltic limestone. (*d*) Sulphur and limestone impregnated with sulphur. (*e*) Including briquettes. (*f*) Lead produced from native ores only, and does not include the metal produced from foreign ores and bullion.

MINERAL IMPORTS OF FRANCE. (*a*) (IN METRIC TONS; 5 f.—$1.)

Year.	Bituminous Substances. (*b*)		Coal. (*c*)		Copper.		Gold.	Silver.	Iron and Steel.		Cast Iron, Crude.	
1894..	59,980	$1,439,600	11,644,000	$34,335,400	30,300	$6,863,800	$92,308,712	$17,719,346	50,034	$1,867,800	47,118	714,600
1895..	43,975	879,600	10,261,069	33,368,400	38,196	8,859,400	50,775,039	28,230,595	66,240	1,619,400	36,247	581,200
1896..	30,954	495,200	10,180,449	34,733,600	46,830	12,692,400	60,167,745	45,951,627	48,423	1,712,000	18,323	368,800
1897..	29,931	419,000	10,457,255	37,894,400	54,460	14,375,400	58,143,077	42,468,793	60,804	1,954,800	35,633	944,400
1898..	20,385	285,400	10,445,090	37,696,200	52,976	13,974,800	39,743,044	33,600,322	......	1,891,600	......	986,000

Year.	Lead.		Nickel.		Nitrate of Soda.	Stone.	Sulphur.		Tin.		Zinc.	
1894..	84,167	$5,236,800	267	$269,400	$7,609,400	$4,989,000	112,536	$1,689,200	8,137	$2,969,400	33,652	$2,787,400
1895..	66,241	4,626,800	252	300,600	8,624,200	3,144,200	110,989	1,598,400	7,691	2,645,800	25,652	1,988,000
1896..	79,752	5,751,600	425	374,800	9,025,400	2,440,000	111,515	1,965,400	8,400	2,820,400	33,459	3,060,800
1897..	86,589	6,661,800	316	475,400	8,105,400	2,675,800	136,118	2,602,200	7,642	2,572,000	31,211	2,998,800
1898..	74,902	5,843,600	330	505,800	8,918,400	2,814,600	130,289	2,490,200	9,247	3,099,800	32,342	3,072,400

Year.	Alum.	Anti mony Ore.	Borax.	Bro-mides.	Carbo-nate of Lead.	Cement	Chlor-hydric Acid.	Chlo-ride of Lime.	Chlo-ride of Potash.	Chro-mate of Potash and of Soda.	Copper Ore.	Gold Ore Kg.
1894..	....	153	....	....	....	....	....	....	....	....	10,191	*e*24,824
1895..	109	....	442	12	1,077	13,441	2,777	1,047	3,524	2,875	10,450	*b*8,811
1896..	41	....	255	13	892	14,395	2,468	2,033	11,499	2,838	8,584	*e*101,499
1897..	54	(*d*)	264	18	1,327	15,141	2,119	1,713	11,630	2,852	11,960	(*d*)
1898..	27	(*d*)	139	30	1,376	11 290	1,994	1,288	10,929	2,890	8,779	(*d*)

Year.	Iron.		Kaolin.	Lead Ore.	Lime.	Manga-nese Ore.	Marble.	Mercury	Nitrate of Potash.	Nickel Ore.	Nitric Acid.
	Ore.	Pyrites.									
1894..	1,638,458	56,672	....	5,825	....	43·3	....	....	....	14,941	....
1895..	1,651,369	67,930	359	5,032	246,677	41·4	366	178	775	10,303	702
1896..	1,862,043	45,788	387	5,569	283,707	61·6	394	234	2,614	15,756	684
1897..	2,137,860	69,470	423	13,981	321,647	85·5	444	248	1,309	17,441	774
1898..	2,032,240	71,569	404	14,377	346,000	100·2	452	221	1,008	24,935	922

Year.	Oxides.						Petro-leum.	Phos-phates.	Plaster.	Plati-num. Kg.	Potash and Carbo-nate of.	Sal Ammo-niac.	Salt.
	Cobalt	Copper.	Iron.	Lead.	Ura-nium.	Zinc.							
1894..	....	....	....	....	....	....	....	....	....	....	....	....	....
1895..	5	24	855	1,931	2	802	258,700	139,600	2,412	926	796	9,923	17,598
1896..	5	22	897	1,984	4	938	272,693	256,888	1,774	2,117	1,526	15,256	17,191
1897..	9	29	1,125	1,759	5	1,114	288,671	313,608	1,869	1,069	1,769	27,454	32,917
1898..	9	52	1,021	1,874	15	1,256	291,961	336,842	2,040	505	2,418	30,426	35,863

Year.	Silver Ore. Kg.	Slates.	Soda. Caustic	Stone.		Sulphates.		Sulphide of Mercury.	Sulphuric Acid.	Superphosphate of Lime.	Tin Ore.	Zinc Ore.
				Building.	Paving.	Copper.	Iron.					
1894..	3,159,514	....	....	....	....	....	....	....	....	....	123	34,955
1895..	f524,394	8	1,021	95,461	49,762	24,404	3,882	23	3,461	150,758	104	41,622
1896..	f1,818,650	12	1,109	72,919	57,165	33,803	3,086	25	3,995	185,602	7	50,899
1897 .	(d)	11	1,378	93,004	66,741	30,132	1,353	24	3,147	195,853	149	58,074
1898..	(d)	11	1,772	80,600	61,110	30,897	896	19	4,666	178,569	357	60,481

(a) The figures for 1894 are from *Annales du Commerce Extérieur*, Paris, and for the remaining years from *L'Economiste francais*, and represent the *Commerce Spécial* of France. (b) Includes bitumen, bituminous schist and sand, and asphaltic limestone. (c) Including coke. (d) Not reported. (e) Gold and platinum, in ore, sheets, leaves, or threads. (f) Silver in ore, sheets, leaves, or threads.

MINERAL AND METALLURGICAL EXPORTS OF FRANCE. (IN METRIC TONS.)

Year.	Aluminum.	Antimony.		Copper.		Gold.		Iron.				Iron Pyrites	Lead.	
		Ore.	Metal	Ore.	Metal	Ore.	Metal	Ore.	Pig.	Cast.	Steel.		Ore.	Metal
1894...	31	1,173	20·0	4,536	6,598	966	4,448	247,519	140,718	156,688	87,361	48,627	11,868	8,133
1895....	110	832	68·2	1,772	8,829	......	1,353	236,923	15,054	10,755	29,074	37,908	8,670	8,037
1896....	793	736	73·8	1,261	10,494	......	2,193	238,430	25,900	19,521	63,543	44,232	8,597	10,856
1897....	224	623	61·4	2,000	12,657	......	3,335	299,589	40,877	10,865	64,909	54,307	12,007	10,864
1898....	192	616	100·6	1,783	14,350	......	1,812	236,160	28,221	16,143	71,772	60,406	10,216	8,663

Year.	Manganese Ore.	Nickel.		Quicksilver.	Silver—Kg.		Tin.		Zinc.	
		Ore.	Metal.		Ore.	Metal.	Ore.	Metal.	Ore.	Metal.
1894....	8,039	241	354	25	51,272	11,472	4	816	58,281	6,595
1895....	16,193	....	408	....	....	13,567	....	650	61,291	5,849
1896....	10,913	....	490	....	....	9,849	....	744	62,415	10,485
1897....	19,464	....	498	....	....	5,374	....	651	79,909	10,977
1898 ...	12,229	....	526	....	....	1,886	....	587	60,664	16,995

MINERAL PRODUCTION OF ALGERIA. (IN METRIC TONS AND DOLLARS; 5f —$1.)

Year.	Antimony Ore.		Cement.		Clays.		Copper.				Gypsum, for Fertilizing.		Iron.			
							Ore.		Matte.				Ore.		Cast, 2d Fusion.	
1893..	79	$4,740	...	.....	......	....	6,002	$42,808	619	$64,160	......	....	393,921	$702,388	517	$34,916
1894..	175	5,200	...	.....	......	....	......	....	702	64,584	23,900	$114,900	343,830	527,782	538	32,499
1895..	307	7,365	...	.....	......	....	......	....	364	30,800	2,800	1,248	318,416	508,946	537	35,894
1896..	658	18,957	60	$300	48,297	$42,526	......	....	427	39,028	300	138	374,476	583,946	577	35,430
1897..	781	21,621	...	.....	67,180	59,607	......	....	289	28,380	350	175	441,467	663,282	699	44,870

Year.	Lead-Silver Ore.		Lime.				Marble.		Onyx.		Phosphate Rock.		Plaster for Building.	
			Hydraulic.		White.									
1893..	212	$4,181	......	....	......	....	.....	......	.....	....	....	....	....	....
1894..	276	4,045	......	....	......	....	.....	......	.....	....	64,260	$230,888	....	....
1895..	178	2,959	20,000	$120,000	11,890	$39,840	1,112	$32,248	1,764	$132,300	157,880	634,323	34,136	$131,978
1896..	117	2,188	20,000	120,000	9,450	36,480	900	26,100	900	25,530	165,738	500,905	20,870	114,123
1897..	145	3,041	20,425	122,500	9,215	32,200	1,660	26,150	364	25,480	228,141	912,564	29,120	109,478

Year.	Quicksilver Ore.		Road-Making Material.		Salt.		Sand and Gravel.		Stone not Otherwise Specified.		Zinc Ore.	
1893..	757	$9,806	....	....	19,008	$93,416	....	....	....	....	24,390	$197,960
1894..	866	11,602	....	....	17,830	74,763	....	....	....	....	29,703	163,892
1895..	85	6,210	348,210	$136,137	25,758	102,990	20,400	$2,400	341,494	$146,070	13,967	93,486
1896..	8	500	376,000	147,000	19,658	85,122	41,400	8,280	453,810	167,000	17,587	168,768
1897..	......	....	468,040	156,470	23,222	78,058	80,860	20,280	527,490	351,431	32,269	301,538

Algeria exported in 1893, 6,762 of lead and copper ore, and 27,844 of zinc ore; in 1894, 2,678 of lead and copper ore, 28,788 of zinc ore, and 71 tons of antimony ore.

MINERAL PRODUCTION OF NEW CALEDONIA. (*a*) (IN METRIC TONS.)

Substances.	1889.	1890.	1891.	1892.	1893.	1894.	1895.	1896.	1897.
Chrome iron ore	2,254	3,826	4,343	512	242	2,927	8,079	20,186	3,949
Cobalt ore	2,185	3,740	1,343	2,200	2,200	4,112	4,277	*c*4,823	3,200
Copper ore	9,500	(*b*)	199	(*b*)	(*b*)	(*b*)	(*b*)	(*b*)	2,300
Copper regulus	(*b*)	259	152	(*b*)	(*b*)	(*b*)	(*b*)	(*b*)	....
Lead ore	2,500	881	236	(*b*)	(*b*)	(*b*)	(*b*)	(*b*)	300
Nickel ore	19,741	22,689	60,921	83,114	69,614	61,243	29,623	6,417	26,464

(*a*) From *Statistique de l'Industrie Minéralé*. (*b*) Not stated in the reports. (*c*) Exports.

MINERAL PRODUCTION OF TUNIS. (*a*) (IN METRIC TONS AND DOLLARS.)

Year.	Salt.		Silver-Lead.		Zinc Ore.			
					Picked.		Calcined.	
1892	......	....	......	....	....	....	....	....
1893	7,000	$21,000	10	....	1,100	$18.467	3,300	$43,560
1894	7,000	(*b*)	(*b*)	....	31,000	(*b*)	(*b*)	....
1895	8,000	7,200	(*b*)	....	14,800	*c*112,600	(*b*)	....
1896	5,500	22,000	(*b*)	....	12,100	118,600	(*b*)	....

(*a*) From *Annual General Reports on the Mineral Industry of the United Kingdom*, by C. Le Neve Foster. Additional products in 1896: Gypsum, 4,700 metric tons; marble, 1,000 metric tons; phosphate of lime, 1,000 metric tons; building stone (dressed), 18,000 cu. meters; stone (broken), 185,000 cu. meters. (*b*) Not reported. (*c*) Value corresponding to only 10,800 tons.

GERMANY.

THE mineral statistics of the German Empire, together with those of Baden, Bavaria, Prussia and Saxony, are summarized in the subjoined tables:

MINERAL PRODUCTION OF GERMANY. (*a*) (*b*) (IN METRIC TONS AND DOLLARS; 4 marks = $1.)

Year.	Antimony Ore.		Arsenic Ore		Asphaltum.		Boracite.		Coal.		Cobalt, Nickel & Bismuth Ores		Copper Ore.	
1894...	*Nil.*	......	2,906	$30,006	55,981	$112,762	176	$11,065	76,741,127	$127,275,053	4,524	$194,645	588,195	$4,060,039
1895...	24	$218	3,546	35,311	59,563	113,606	150	8,781	79,169,276	134,723,786	5,180	163,450	633,354	3,844,961
1896...	*Nil.*	......	3,691	46,960	61,552	113,348	184	10,620	85,690,233	148,244,000	4,087	129,688	717,306	4,239,362
1897...	*Nil.*	......	3,777	56,021	61,645	94,634	198	10,222	91,054,982	162,234,685	3,355	139,777	700,619	4,752,552
1898...	*Nil.*	......	(*c*)	(*c*)	(*c*)	(*c*)	(*c*)	(*c*)	96,279,992	177,564,243	3,157	138,545	702,781	4,921,152

Year.	Epsomite.		Graphite.		Iron Ore.		Iron Pyrites.		Other Vitriol and Alum Ores.		Kainite.	
1894...	8,252	$17,999	3,133	$45,732	12,392,065	$10,544,385	134,787	$244,864	465	$977	726,524	$2,578,208
1895...	7,328	14,856	3,751	50,612	12,349,600	10,268,935	127,036	243,878	351	525	680,174	2,402,339
1896...	2,350	5,250	5,248	72,108	14,162,315	12,849,650	124,950	231,579	369	512	856,290	2,989,736
1897...	2,601	5,637	3,861	66,126	15,465,979	15,021,922	133,302	241,117	225	338	992,389	3,486,007
1898...	(*c*)	(*c*)	(*c*)	(*c*)	15,893,246	15,202,159	136,849	242,586	(*c*)	(*c*)	1,103,643	3,835,856

Year.	Other Potash Salts.		Lead Ore.		Lignite.		Manganese Ore.		Petroleum.	
1894........	917,049	$2,992,149	162,675	$3,025,884	22,064,575	$13,287,909	43,702	$116,412	17,232	$243,112
1895........	841,748	2,776,504	161,614	3,234,902	24,788,363	14,502,821	41,327	124,534	17,051	240,614
1896........	902,707	2,964,750	157,504	3,249,000	26,797,880	15,233,273	45,062	120,155	20,395	297,128
1897........	953,798	3,080,143	150,179	3,253,884	29,419,503	16,562,642	46,427	115,356	23,303	349,111
1898........	1,105,212	3,576,628	151,601	3,412,128	31,648,498	18,339,869	(*c*)	....	25,789	394,552

Year.	Salt, Rock.		Silver and Gold Ore.		Tin Ore.		Uranium and Wolfram Ores.		Zinc Ore.	
1894........	734,937	$784,923	19,080	$629,672	211	$16,309	40	$6,194	728,616	$2,569,514
1895........	686,940	777,095	10,845	427,047	154	15,197	29	4,449	706,423	2,644,314
1896........	755,823	755,888	18,487	596,604	88	5,881	41	7,081	729,872	4,255,473
1897........	763,412	804,298	9,708	363,271	55	5,982	38	8,367	663,850	4,220,339
1898........	804,658	838,656	12,413	336,657	(*c*)	....	(*c*)	....	641,706	5,511,819

(*a*) Including Luxemburg. (*b*) From *Vierteljahrs- und Monats-hefte zur Statistik des Deutschen Reichs;* (*c*) Not reported.

METALLURGICAL PRODUCTION OF GERMANY. (*a*) (IN METRIC TONS AND DOLLARS; 4 marks = $1.)

Year.	Antimony and Manganese.		Arsenical Products.		Cadmium—Kg.		Copper Matte & Black Copper.		Copper, Ingot.		Gold—Kg.	
1893........	407	$64.486	1,794	$119,588	5,284	$5,461	842	$65,735	24,011	$5,860,517	2,547	$1,771,571
1894........	424	65,244	2,389	166,798	6,472	6,886	676	26,921	25,722	5,467,554	3,199	2,228,947
1895........	989	107,960	3,014	207,416	7,047	9,809	789	49,890	25,777	5,818,903	3,547	2,469,379
1896........	1,326	156,111	2,637	221,397	10,000	19,117	598	39,083	29,319	7,293,387	2,487	1,728,938
1897........	1,665	210,744	2,989	270,949	15,531	44,157	315	14,380	29,408	7,545,376	2,781	1,934,371
1898........	2,711	302,892	2,679	253,623	(*c*)	....	62	2,170	30,695	8,182,102	2,847	1,978,352

Year.	Iron, Pig.		Lead.		Litharge.		Mineral Paints.		Nickel & Various By-Products. (b)		Silver—Kg.		Sulphur.	
1893	4,986,003	$54,081,575	94,659	$4,609,353	3,551	$193,181	2,993	$84,008	1,402	$1,703,265	448,092	$11,736,868	2,161	$51,308
1894	5,380,039	57,892,412	100,751	4,762,392	3,646	192,043	2,834	72,802	997	1,154,757	442,822	9,625,876	2,168	50,298
1895	5,417,548	58,667,695	111,058	5,569,475	3,433	190,806	2,655	72,443	1,143	1,111,074	391,979	8,600,828	2,061	49,024
1896	6,295,272	73,930,749	113,792	6,258,097	3,930	235,698	2,668	75,313	1,391	1,365,330	428,429	9,718,000	2,263	47,181
1897	6,864,405	87,345,441	118,881	7,160,138	3,341	213,699	3,119	84,071	1,464	1,492,337	448,068	9,095,366	2,317	48,327
1898	7,215,927	93,575,641	132,742	8,555,606	3,857	265,496	(c)	....	1,691	1,465,254	480,578	9,539,162	(c)	....

Year.	Sulphuric Acid.		Sulphates. (d)								Tin		Zinc.	
			Copper.		Iron.		Zinc.		Mixed.					
1893	522,822	$3,940,784	4,773	$353,377	8,491	$42,064	4,527	$60,133	232	$5,535	951	$349,135	142,956	$11,821,589
1894	557,903	4,029,659	4,809	363,034	8,922	39,658	4,249	56,815	202	7,436	896	271,804	143,577	10,453,272
1895	537,928	3,713,673	4,638	341,600	9,820	40,579	4,018	55,950	170	6,492	884	266,292	150,286	10,409,160
1896	590,888	3,781,092	6,046	466,478	9,788	39,072	4,811	67,171	165	6,218	459	135,734	153,100	11,777,085
1897	623,130	3,739,472	5,549	470,016	10,351	44,684	5,488	77,495	266	10,543	929	287,829	150,739	12,619,131
1898	664,865	4,085,734	4,352	356,540	(c)	....	(c)	....	(c)	....	993	372,252	152,506	14,455,924

EXTRACTION OF SALTS FROM AQUEOUS SOLUTION. (a)

Year.	Alum.		Aluminum Sulphate.		Common Salt.		Glauber Salt.		Magnesium Chloride	
1893.....	4,102	$115,076	27,082	$505,166	504,522	$3,494,221	77,145	$498,539	12,764	$44,444
1894.....	3,914	107,092	26,804	480,174	522,590	3,574,655	71,929	423,218	17,422	50,913
1895.....	3,358	87,372	30,836	539,000	525,396	3,563,226	71,411	406,872	17,039	52,850
1896.....	3,430	88,250	34,370	595,000	547,486	3,661,034	71,958	448,915	17,525	57,386
1897.....	2,995	73,838	37,053	613,684	543,272	3,034,128	68,822	434,198	18,014	64,294
1898.....	(c)	....	35,366	514,952	565,683	3,116,087	69,111	452,509	(c)	....

Year.	Magnesium Sulphate.		Potassium Chloride.		Potassium Sulphate.		Sulphate of Potassium and Magnesium.	
1893..........	27,548	$79,240	137,216	$4,326,285	27,308	$1,119,835	14,199	$280,792
1894..........	28,628	88,814	149,775	4,722,049	23,281	958,736	14,156	274,694
1895..........	26,028	107,183	154,427	4,921,179	19,452	805,098	9,877	192,989
1896..........	27,161	107,835	174.515	5,718,559	19,682	813,381	4,623	85,977
1897..........	35,072	155,585	168,001	5,764,423	13,774	565,720	7,812	149,079
1898..........	(c)	....	191,347	6,380,220	18,853	763,397	13,982	259,485

(a) From *Vierteljahrs- und Monats-hefte zur Statistik des Deutschen Reichs.* (b) Including metallic bismuth, cobalt products, and uranium salts. (c) Not reported. (d) There was also produced nickel sulphate and tin chloride as follows: 1891, 143 metric tons, $43,401; 1892, 89 tons, $27,003; 1893, 81 tons, $24,172; 1894, 107 tons, $31,398; 1895, 131 tons, $34,511; 1896, 176 tons, $42,986; 1897, 209 tons, $51,333.

MINERAL IMPORTS OF GERMANY. (a) (IN METRIC TONS; unit of value, $1,000; $1 = 4 marks.)

Year.	Alabaster and Marble, Crude.		Aluminum, Nickel Wares, etc.		Ammonia, Sulphate of.		Cement.		Coal.		Cobalt and Nickel.		Coke.	
1893..	21,520	$650	294	$300	42,596	$2,550	26,854	$175	4,664,048	$14,925	1,006	$950	439,182	$1,725
1894..	24,784	750	291	300	36,635	2,375	24,863	175	4,805,971	15,075	431	350	404,179	1,425
1895..	23,785	725	299	300	29,203	1,600	27,351	200	5,117,356	15,825	621	425	461,779	1,675
1896..	26,655	800	335	350	32,061	1,450	32,394	275	5,476,753	15,275	b 951	600	393,881	1,625
1897..	29,633	900	417	425	33,113	1,325	42,364	350	6,072,029	16,625	b 1,390	875	435,161	1,950

Year.	Copper and Alloys.				Copper and Brass Manufactures.						Glass Manufactures, All Kinds.		Gold.		
	Crude. (c)		Bars and Sheets,		Fine.		Wire.		Cartridge Cases, Coarse Wares, etc.				Coin.	Bullion.	
1893..	38,455	8,850	448	$125	560	$450	151	$50	613	$300	3,008	$925	$16,050	27	$18,975
1894..	37,032	7,900	495	125	565	450	131	50	539	225	3,008	1,100	50,350	37	25,775
1895..	44,365	10,050	426	125	625	525	171	50	563	250	3,119	1,100	12,475	17	11,875
1896..	56,115	13,675	401	125	797	675	127	50	582	275	2,823	1,050	23,475	45	31,600
1897..	67,573	17,025	400	125	846	700	80	25	612	300	2,057	475	14,350	35	24,325

Year.	Gold and Silver.				Gold, Silver, and Platinum Ores.		Guano.		Iodide of Potassium, etc.		Iodine.		Iron.			
	Scrap, Pagament		Manufactures.										Ore.		Pig.	
1893.	41	$4,900	32	$1,300	20,516	$5,125	52,411	$2,125	2	(e)	324	$2,150	1,573,202	$5,250	218,998	$2,475
1894..	66	5,800	37	1,350	13,935	3,000	47,230	1,850	24	$150	431	2,850	2,093,007	6,850	203,948	2,375
1895..	45	4,125	35	975	6,134	1,350	39,638	1,325	22	125	113	750	2,017,136	6,825	188,217	2,250
1896..	39	3,700	34	825	6,024	3,950	38,120	1,150	29	150	94	625	2,586,706	8,700	322,502	4,000
1897..	37	3,075	35	750	8,927	4,475	40,956	1,125	18	25	164	950	3,185,644	11,675	423,127	5,375

Year.	Iron—*Continued.*																	
	Scrap.		Blooms, Bars, and Ingots.		Angle.		Wrought, in Bars.		Rails.		Wire.		Cast, Crude.		All Other Manufactures.		Lead, White.	
1893..	8,178	$100	511	(e)	146	(e)	17,054	$625	6,310	$125	4,946	$400	17,711	$1,850	7,119	$1,000	932	$75
1894..	7,900	100	719	$25	245	(e)	19,966	700	3,542	75	4,888	400	14,997	1,800	9,793	1,100	646	50
1895..	11,339	125	757	25	124	(e)	19,777	700	1,831	25	5,583	450	13,749	1,675	11,654	1,225	479	25
1896..	14,679	175	1,054	26	176	(e)	23,770	875	140	(e)	6,398	450	19,600	2,450	12,933	1,650	579	50
1897..	37,957	450	1,038	25	1,081	$25	29,467	1,225	774	25	5,609	375	24,627	2,950	18,135	1,975	696	50

Year.	Lignite.		Manganese Ore.		Mineral Oil.				Phosphate of Lime.		Porcelain.	
					Petroleum.		Lubricating Oil.					
1893..	6,705,672	$6,375	12,093	$250	765,100	$11,825	70,576	$4,400	226,923	$2,825	646	$175
1894..	6,868,162	6,175	14,255	225	785,102	11,375	65,703	2,800	286,812	3,575	693	200
1895..	7,181,050	9,875	22,576	275	811,058	15,400	75,041	3,275	278,046	3,475	748	200
1896..	7,637,503	11,275	63,870	750	853,642	14,950	81,256	3,425	216,950	2,700	806	325
1897..	8,111,076	12,775	86,911	925	894,611	11,550	83,957	3,250	289,234	3,625	812	350

Year.	Potash.		Other Potassium Compounds.						Precious Stones, Pearls and Corals, Crude and Set.		Pyrites. (*d*)	
			Chloride.		Cyanide.		Sulphate.					
1893..	2,168	$200	828	$25	1	(e)	1,665	$500	77	$1,325	274,766	$1,300
1894..	2,296	200	478	25	1	(e)	1,306	500	35	950	315,115	1,550
1895..	1,634	125	1,334	50	5	(e)	895	25	60	1.125	293,446	1,450
1896..	1,430	100	1,058	50	3	(e)	703	25	42	1,175	343,852	1,650
1897..	1,734	125	715	25	7	(e)	912	25	86	1,575	356,869	1,900

Year.	Roofing Tile and Brick.		Salt.		Saltpeter.		Silver.				Slag and Slag Wool.		Slate.	
							Bullion.		Coin.					
1893..	98,984	$350	23,645	$100	647	$ 75	78	$2,025	24	$575	477,183	$1,675	61,543	$1,075
1894..	118,010	400	23,440	100	1,135	125	190	4,075	20	375	632,878	2,200	61,401	1,050
1895..	114,020	375	24,286	100	1,005	100	98	2,150	18	350	537,542	1,825	52,418	950
1896..	140,333	475	22,908	100	1,380	125	128	2,900	21	425	680,251	2,525	53,583	1,050
1897..	151,455	575	(e)	....	2,889	275	147	2,975	20	375	670,224	2,575	48,380	1,075

Year.	Soda, Calcined.		Soda. Nitrate.		Stassfurt Salts.		Stone.				Superphosphate.		Thomas Slag, Ground.	
							Crude or Simply Hewn.		Grindstones, Polishing and Whetstones.					
1893..	420	$25	384,710	$16,350	1,061	(e)	697,448	$4,800	2,922	$225	110,876	$1,800	78,458	$1,050
1894..	753	25	404,561	17,200	770	$50	859,459	5,375	2,578	200	124,373	2,025	88,443	550
1895..	1,008	25	459,514	17,800	910	50	760,433	3,800	2,734	350	96,099	1,450	92,251	550
1896..	1,295	25	449,028	16,850	143	(e)	767,908	3,850	2,869	275	81,740	1,025	83,765	525
1897..	916	25	465,493	16,875	6	(e)	853,825	4,275	(e)	....	110,782	1,275	110,216	575

Year.	Tin.				Zinc.								Zinc-White, Zinc-Gray, and Lithophone.	
	Crude.		Manufactures		Ore.		Crude.		Drawn and Rolled.		Manufactures			
1893..	10,538	$4,475	69	$75	23,883	$425	13,211	$1,200	74	....	100	$50	2,633	$250
1894..	10,775	3,750	64	75	14,712	175	17,988	1,375	275	$25	112	75	2,513	200
1895..	10,581	3,375	79	75	25,818	300	17,542	1,250	128	....	114	75	2,906	225
1896..	13,798	4,275	75	75	21,493	325	16,343	1,350	180	25	139	75	2,899	250
1897..	12,395	3,875	75	75	24,735	400	19,734	1,850	130	25	146	75	3,532	325

(*a*) From *Statistisches Jahrbuch für das Deutsche Reich*, 1897. (*b*) Nickel only. (*c*) Previous to 1896 includes copper scrap. (*d*) Prior to 1896 includes alum ore. (*e*) Not reported.

MINERAL EXPORTS OF GERMANY. (a) (IN METRIC TONS; unit of value, $1,000; $1 = 4 marks.)

Year.	Alabaster and Marble, Crude.		Aluminum, Nickel Wares, etc.		Ammonia, Sulphate of.		Cement.		Coal.		Cobalt and Nickel.		Coke.	
1893..	538	$25	1,463	$1,475	422	$25	423,892	$2,450	9,677,305	$26,125	115	$125	1,902,424	$7,225
1894..	1,042	50	1,374	1,350	332	25	406,369	2,575	9,739,035	25,300	101	100	2,261,924	8,875
1895..	1,645	75	1,669	1,675	1,336	75	471,124	3,000	10,360,838	26,750	176	125	2,293,328	8,900
1896..	2,174	100	1,977	2,050	2,201	100	478,340	4,100	11,598,757	30,475	143	100	2,216,395	9,000
1897..	2,727	75	1,899	1,950	2,623	100	524,557	4,500	12,389,907	33,375	b 169	100	2,161,886	9,150

Year.	Copper and Alloys.				Copper and Brass Manufactures.						Glass Manufactures, All Kinds.		Gold.			
	Crude. (c)		Bars and Sheets, Unplated.		Fine.		Wire, Unplated.		Cartridge Cases, Coarse Wares, etc.				Coin.		Crude and in Bars.	
1893..	7,497	$1,775	4,890	$1,350	3,860	$2,950	3,053	$950	5,733	$3,550	107,509	$8,925	35	$21,725	5	$3,650
1894..	6,609	1,450	5,007	1,300	4,117	3,100	3,433	1,050	5,870	3,475	110,623	9,225	17	10,875	4	2,450
1895..	6,329	1,425	4,700	1,300	4,912	3,800	3,975	1,250	7,093	4,425	118,199	10,150	25	15,525	7	5,050
1896..	5,996	1,500	5,406	1,700	5,961	4,950	5,910	1,950	6,806	4,375	108,892	9,175	40	25,150	35	24,225
1897..	7,183	1,825	5,718	1,825	5,583	4,625	6,176	2,025	5,409	3,350	105,702	9,050	11	7,150	32	22,525

Year.	Gold and Silver Manufactures.		Gold, Silver, and Platinum Ores.		Guano.		Iodide of Potassium, etc.		Iodine.		Iron.					
											Ore.		Pig.		Scrap.	
1893..	87	$5,975	23	......	6,448	$275	105	$700	8	$50	2,353,398	$1,750	108,675	$1,475	62,954	$775
1894..	84	6,350	12	......	3,347	150	107	725	13	75	2,558,729	1,925	154,647	2,000	77,723	1,025
1895..	88	5,950	359	$50	2,207	75	128	875	23	150	2,480,136	1,850	135,289	1,775	84,814	1,150
1896..	97	7,525	14	25	1,600	50	142	950	26	175	2,642,294	1,975	140,449	1,925	52,466	750
1897..	101	9,975	53	25	2,019	50	124	775	26	150	3,230,391	2,400	90,885	1,250	38,102	550

Year.	Iron—*Continued.*														Lead, White.	
	Blooms, Bars and Ingots.		Angle.		Wrought, in Bars.		Rails.		Wire.		Cast, Crude.		Man'factures, All Other.			
1893	55,565	$1,050	107,383	$2,775	231,297	$5,900	87,360	$1,850	191,518	$6,525	113,867	$17,125	232,948	$18,050	13,412	$1,000
1894	41,992	850	130,458	3,225	300,559	7,375	119,410	2,475	209,818	6,575	125,057	17,450	379,862	20,450	13,990	950
1895	61,808	1,200	172,863	4,075	277,991	7,000	116,627	2,550	205,331	6,500	135,383	20,450	248,355	18,750	14,260	1,025
1896	49,529	1,125	178,887	4,675	259,461	7,175	129,413	3,100	207,116	6,875	153,652	24,325	309,882	23,200	16,350	1,275
1897	39,792	900	169,287	4,400	246,772	7,075	113,473	2,850	198,909	6,625	170,084	26,725	296,949	22,950	14,786	1,225

Year.	Lignite.		Lime.		Manganese Ore.		Porcelain, White and Variegated.		Potash		Potassium Compounds.						Precious Stones, Pearls and Corals, Crude and Set	
											Chloride.		Cyanide.		Sulphate.			
1893	22,757	$25	4,652	$75	2,688	$100	15,753	$3,375	10,865	$1,125	85,858	$3,050	38	$75	31,855	$1,025	5	$775
1894	20,444	25	3,761	75	2,787	75	15,532	3,325	12,105	1,175	98,913	3,375	36	125	25,357	850	5	725
1895	18,814	25	5,813	100	4,461	125	19,441	4,550	13,443	1,100	78,718	2,825	48	150	23,132	775	2	425
1896	15,703	25	5,548	100	7,178	150	21,624	8,975	12,673	1,050	85,862	3,075	55	125	16,019	575	6	475
1897	19,112	25	4,000	75	8,615	100	21,687	8,775	13,100	775	80,389	2,850	1,086	(e)	20,971	675	5	525

Year.	Pyrites. (d)		Roofing Tile, and Building Stone.		Salt.		Saltpeter.		Silver.				Slag and Slag Wool.		Slate.	
									Crude and in Bars.		Coin.					
1893..	15,889	$50	162,586	$1,125	196,095	$825	9,652	$925	451	$11,800	29	$700	17,413	$50	3,798	$50
1894..	17,510	100	169,110	1,125	237,058	650	13,213	1,325	406	8,725	8	150	21,504	75	4,056	55
1895..	14,228	75	188,046	1,350	196,344	600	12,928	1,250	259	5,750	4	75	20,432	50	4,489	75
1896..	16,833	75	287,811	1,775	214,060	625	11,323	1,750	306	6,975	32	650	17,214	50	5,796	100
1897..	15,387	75	233,229	1,550	(e)	......	8,986	850	371	7,550	28	525	27,723	100	4,948	100

Year.	Soda. Calcined.		Soda. Nitrate.		Stassfurt Salts.		Stone.				Superphosphate.		Thomas Slag, Ground.	
							Crude or Simply Hewn.		Grindstones, Polishing and Whetstones.					
1893..	30,433	$950	13,541	$600	212,289	$1,325	483,104	$2,425	11,516	$575	62,800	$1,175	80,922	$1,100
1894..	33,556	875	14,217	700	228,765	1,425	578,451	2,900	10,909	550	60,569	1,125	85,630	850
1895..	31,418	775	13,437	800	221,184	1,375	505,163	2,525	10,465	650	55,742	975	80,056	525
1896..	41,106	925	9,078	400	285,023	1,575	620,394	3,100	11,914	900	57,391	850	134,257	875
1897..	45,672	1,025	11,364	475	337,577	1,700	545,723	2,725	(e)	......	61,800	875	169,336	1,225

Year.	Tin.				Zinc.								Zinc-White, Zinc-Gray, and Lithophone.	
	Crude.		Manufactures		Ore.		Crude.		Drawn and Rolled.		Manufactures			
1893..	565	$250	731	$700	25,059	$450	62,592	$5,325	17,459	$1,675	893	$625	12,343	$1,000
1894..	613	225	819	725	35,682	425	61,800	4,525	16,038	1,350	842	575	14,437	1,750
1895..	700	225	975	825	31,031	350	56,933	4,025	15,921	1,275	1,032	700	15,539	1,125
1896..	868	275	1,010	850	37,959	575	58,082	4,750	16,227	1,500	1,206	825	16,969	1,325
1897..	861	275	967	800	30,047	500	51,341	4,400	17,453	1,650	1,163	800	17,631	1,325

(*a*) From *Statistisches Jahrbuch für das Deutsche Reich.* (*b*) Nickel only. (*c*) Previous to 1896 includes copper scrap. (*d*) Previous to 1896 includes alum ore. (*e*) Not reported.

MINERAL IMPORTS AND EXPORTS OF GERMANY FOR THE YEAR 1898. (IN METRIC TONS.)

The following figures are taken from *Chemiker Zeitung*, of Feb. 4, 1899, and while probably from official sources, are subject to correction.

Substances.	Imports.	Exports.
Alum, and aluminate of sodium and alumina, including hydrate.	152	22,765
Aluminum	1,104	125
Ammonium carbonate	1,578	2,795
Ammonium sulphate	30,254	4,083
Ammoniacal liquor	14,211	14,608
Antimony	1,729	90
Antimony and arsenic ore	570	238
Arsenic	....	22
Arsenic, white	179	966
Arsenical compounds	7·5	694
Asbestos and asbestic mastic	3,448	377
Asphalt, pitch, and wood cement	40,550	33,207
Barium chloride	2,377	3,059
Barium salts, N. E. S	82	2,336
Baryta-white	2	1,741
Barytes	1,889	41,092
Borax and boracic acid	2,510	2,009
Brass and tombac	1,733	5,637
Bromide of potash and other bromine preparations	7	310
Bromine	....	95
Calcium chloride	124	1,339
Carbon bisulphide	....	....
Carbonic acid	55	3,005
Cement	53,519	551,744
Chloride of lime	154	16,814
Chloride of magnesium	52	14,236
Chrome alum	0·4	1,013
Chrome ore	5,847	50
Coal	5,820,332	13,989,223
Coal-tar oil, light	5,915	1,612
Coal-tar oil, heavy	1,999	6,423
Cobalt and nickel ore	6,433	15
Coke	332,579	2,133,179
Copper, crude	73,291	6,972
Copper, in bars and sheets	136	1,669
Copper alloys, in bars, sheets, etc.	314	3,699
Cinnabar	11	209
Cryolite and bauxite	21,984	306
Explosives	36	3,135
Fluorspar	42	10,467
Gold, silver, and platinum ores	7,481	19
Gold preparations and salts	0·4	6
Graphite, amorphous	20,269	2,936
Gunpowder	4	2,492
Gypsum	....	....
Iodine	216	26
Iron, crude, all kinds	384.561	187,375
Iron, wrought, in bars	26,014	263,698
Iron alum and iron mordant	882	699
Iron ores	3,516,577	2,933,734
Iron oxide, red	2,042	1,181
Kaolin, feldspar, and refractory clay	208,186	129,082
Lead and copper ores	64,727	24,163
Lead	47,497	24,867

Substances.	Imports.	Exports
Lead, white	822	16,473
Lignite	8,450,149	22,155
Litharge	569	3,987
Magnesium, artificial carbonate	55	114
Magnesium, natural carbonate	9,209	2,130
Manganese ore	130,711	4,810
Manganese preparations	46	781
Mineral oil for use in the arts	3,155	....
Muriatic acid	3,421	13,393
Nickel	1,467	203
Nitric acid	595	1,131
Ozokerite, crude	13,475	1,413
Ozokerite, refined	215	1,734
Petroleum, crude	810 391	2
Petroleum, distilled	5,624	71
Petroleum, refined	89,675	53
Phosphorus	265	88
Pitch, except asphalt	52,700	4,887
Potash, all kinds	11,486	13,456
Potash, caustic	165	8,529
Potassium chlorate	421	96,236
Potassium chloride and sodium chloride	1,327	811
Potassium chromate	686	689
Potassium cyanide	2	1,907
Potassium iodide and other iodine preparations	16	135
Potassium sulphate	998	27,105
Quicksilver	560	97
Salt	21,957	225,548
Saltpeter, Chile	25,054	12,883
Saltpeter, potassium	1,895	10,969
Soda, calcined	7,824	32,249
Soda, caustic	579	5,118
Soda, crude and crystallized	....	....
Soda chromate	328	2,139
Soda, double carbonate	223	985
Sodium sulphide	124	5,020
Stone, refractory, from clay	41,597	117,851
Strontia	8,817	197
Strontia preparations	161	163
Sulphur	30,269	652
Sulphate of potassium and sulphate of sodium	3,169	1,680
Superphosphate	110,104	72,847
Thomas slag, ground	88,374	187,598
Tin	14,623	874
Tin salts	80	235
Ultramarine	49	4,120
Vitriol, all kinds	2,398	6,283
Waterglass	165	5,175
Witherite	2,825	431
Zinc, ingot	24,116	51,324
Zinc ores	48,050	30,408
Zinc, sheet	53	14,477
Zinc white, zinc gray, and zinc sulphide	3,652	18,615

MINERAL PRODUCTION OF BADEN. (a) (IN METRIC TONS AND DOLLARS; 4 marks = $1.)

Year.	Alumina Sulphate.		Coal.		Glauber Salt.		Gypsum.		Iron	
									Cast, Second Fusion.	
1893	1,498	$33,705	3,200	$9,200	3,718	$13,941	28,905	$16,217	24,940	$1,167,29
1894	1,863	31,671	3,768	10,726	3,975	9,937	40,652	21,407	31,195	1,302,84
1895	2,136	31,779	4,189	12,657	(b)	....	29,805	19,504	31,356	1,400,72
1896	1,824	27,133	4,001	11,003	(b)	....	32,801	24,651	36,235	1,730,77
1897	1,824	27,134	4,752	11,880	(b)	....	40,702	26,972	39,988	1,876,28

Year.	Iron—*Continued.*				Lead and Copper Ores.		Limestone. (d)		Manganese Ore.		Quartz Sand	
	Ingot.		Wrought.									
1893	189	$23,562	644	$38,376	5·0	$454	4,170	$1,122	5	$13	1,924	$1,84
1894	2,391	89,252	1,265	58,539	c2·0	415	107,139	16,218	(b)	....	2,211	1,32
1895	4,126	158,077	1,047	50,238	c2·5	105	40,188	6,332	(b)	....	3,319	1,9
1896	3,418	143,371	1,118	51,954	(b)	103	116,913	16,496	130	325	1,198	6
1897	3,875	147,718	1,167	58,401	(b)	....	137,670	19,502	(b)	....	1,648	1,17

Year.	Refractory Earths.		Salt.		Sandstone. (e)		Sulphuric Acid.		Tripoli.		Zinc Ore.	
1893	8,475	$6,647	28,473	$211,486	8	$200	12,573	$94,926	8·0	$763	484	$1,6
1894	8,772	8,346	27,867	211,226	1,592	1,775	11,955	76,208	12·0	976	723	1,44
1895	7,453	4,130	28,188	214,031	443	1,000	11,944	65,692	8·5	650	21	1
1896	6,819	4,267	29,227	191,252	(b)	....	14,226	78,243	9·0	1,125	(b)	....
1897	11,450	6,654	31,445	136,993	(b)	....	13,365	73,507	9·0	1,800	(b)	....

(a) Reported to THE MINERAL INDUSTRY by the *Grossherzogliche Badische Domänendirektion*, Carlsruhe. (b) Not reported (c) Lead ore only. (d) Including cement stone and bituminous shale. (e) Including millstones.

MINERAL PRODUCTION OF BAVARIA. (a) (IN METRIC TONS; 4 marks = $1.)

Year.	Barytes.		Building Stone.		Cement.		Coal.		Lignite.	
1893	2,607	$3,973	568,410	$837,190	79,779	$80,386	802,537	$2,000,357	17,167	$14,528
1894	4,550	5,425	627,116	844,769	81,723	83,426	806,389	1,972,941	20,687	17,059
1895	3,587	4,388	612,969	854,209	85,089	85,864	903,340	2,147,045	26,531	23,304
1896	3,397	3,404	658,581	856,753	94,481	99,929	900,080	2,129,069	35,934	33,580
1897	3,365	2,906	663,749	711,029	97,831	60,591	917,022	2,267,086	39,043	24,913

Year.	Copper Ore.		Emery.		Feldspar.		Fireclay.		Flagstones.		Fluorspar.		Graphite.	
1893	650	$2,762	172	$1,498	1,200	$2,400	110,970	$217,859	26,873	$110,493	3,988	$4,473	3,140	$52,600
1894	(b)	....	148	1,583	1,720	3,540	121,950	219,154	18,083	77,932	3,616	6,015	3,133	45,732
1895	(b)	....	229	2,353	1,115	3,075	106,925	180,188	18,909	84,685	3,940	5,986	3,751	50,612
1896	(b)	....	249	2,527	1,315	3,145	110,174	251,899	20,559	86,620	5,218	8,280	5,248	72,108
1897	(b)	....	217	2,307	1,689	4,678	144,425	300,495	14,647	57,343	4,904	5,744	3,861	86,128

Year.	Gypsum.		Iron Ore.		Iron Pyrites.		Kaolin (China Clay).		Limestone.		Lithographic Stone.		Manganese Ore.	
1893	25,540	$13,275	149,271	$147,110	2,108	$5,269	17,835	$22,117	283,488	$76,374	8,885	$234,637	180	$440
1894	25,267	13,219	138,977	140,944	1,928	4,748	15,944	22,121	229,784	60,943	9,286	185,730	80	270
1895	21,773	11,597	145,191	148,282	1,955	14,616	13,250	19,042	271,196	69,596	8,788	175,770	150	113
1896	28,799	15,334	161,279	164,262	1,997	14,930	19,080	26,268	238,434	74,982	10,868	163,020	70	35
1897	26,153	13,609	172,699	173,809	2,211	7,080	24,086	38,345	224,550	76,385	13,941	217,821	130	197

Year.	Ocher and Mineral Paints.		Quartz Sand.		Salt. Rock.		Sandstone.		Slate.		Soapstone.		Whetstone.	
1893	10,317	$31,168	30,400	$10,053	1,214	$7,564	238,405	$314,290	1,485	$12,886	1,911	$29,376	57	$1,300
1894	9,195	22,721	29,775	9,632	630	4,122	221,642	291,924	1,146	10,562	1,900	22,913	66	1,456
1895	8,579	25,945	33,457	10,513	1,321	8,443	210,450	281,893	1,278	11,766	2,421	37,979	215	9,000
1896	8,667	26,496	29,868	8,441	708	4,572	235,518	321,591	1,565	14,252	3,051	52,458	88	3,740
1897	8,673	25,872	31,678	8,812	1,161	5,582	242,112	315,117	1,496	14,001	2,464	35,360	95	3,800

Year.	Iron, Bar.		Iron, Cast, 1st Fusion.		Iron, Cast, 2d Fusion.		Iron, Pig.		Iron, Sheet.		Iron, Wire.	
1893..	49,908	$1,483,411	206	$6,960	49,614	$2,307,051	75,209	$856,182	1,482	$57,766	308	$8,211
1894..	46,860	1,351,169	196	6,606	53,342	2,490,670	75,669	865,543	268	10,087	279	7,426
1895..	48,596	1,368,013	293	9,898	62,131	2,868,070	77,114	884,615	150	5,250	100	2,551
1896..	53,573	1,587,338	114	3,993	71,006	3,378,247	79,621	907,306	(b)	....	243	6,232
1897..	58,200	1,884,224	138	4,829	78,008	3,735,979	83,418	968,932	(b)	....	252	6,895

Year.	Steel.		Salt. From Brine.		Sulphate of Soda.		Other Sulphates, Including Alum.		Sulphuric Acid.	
1893....................	60,824	$1,645,074	42,154	$459,596	1,230	$6,975	628	$29,623	7,355	$64,692
1894....................	86,594	2,022,324	42,183	459,146	568	3,250	668	34,526	6,979	59,693
1895....................	96,829	2,336,689	41,106	446,628	494	2,875	637	34,170	6,515	52,800
1896....................	101,954	2,497,814	40,400	438,379	663	3,067	601	32,209	7,064	61,214
1897....................	115,530	2,851,035	41,533	338,029	2,318	12,749	981	38,540	7,041	63,709

(a) From the *Uebersicht der Production des Bergwerks-, Hütten-, und Salinen- Betriebes in dem Bayerischen Staate.* (b) Not reported.

MINERAL PRODUCTION OF PRUSSIA. (a) (METRIC TONS; 4 marks = $1.)

Year.	Alum Shale		Antimony Ore.		Arsenic Ore		Asphalt.		Boracite.		Coal.		Lignite.	
1893..	302	$418	15	$75	1,634	$16,359	11,290	$34,730	139	$10,537	67,657,844	$110,084,144	17,553,482	$11,118,291
1894..	126	189	(b)	......	2,222	22,220	14,108	44,537	164	10,490	70,643,979	113,518,107	17,791,062	10,512,840
1895..	125	188	24	218	3,046	30,460	14,391	44,659	140	8,256	72,621,509	119,888,605	20,114,877	11,527,852
1896..	154	231	(b)	......	3,247	41,196	16,204	43,715	171	9,896	78,993,655	132,782,104	21,981,201	12,195,391
1897..	129	193	(b)	......	3,377	50,655	11,466	18,706	185	9,663	84,253,393	145,665,149	24,222,911	13,324,245

Year.	Cobalt Ore.		Copper Ore.		Epsom Salt.		Iron Ore.		Iron Pyrites.		Kainite.	
1893..	203	$8,492	573,721	$4,471,014	7,721	$14,656	4,007,898	$6,036,567	110,072	$192,937	531,560	$1,904,822
1894..	203	5,741	579,132	4,012,585	7,734	16,833	4,012,446	6,141,224	123,149	215,597	529,169	1,814,585
1895..	120	6,298	623,420	3,799,452	6,789	13,701	3,726,725	5,700,009	115,051	201,660	513,097	1,804,950
1896..	181	9,868	707,395	4,186,321	1,793	3,971	4,053,109	7,101,832	117,545	203,745	616,462	2,424,132
1897..	121	6,256	690,338	4,695,679	2,248	4,882	4,183,536	8,432,766	121,766	209,201	716,348	2,529,349

Year.	Other Potash Salts.		Lead Ore.		Manganese Ore		Nickel Ore.		Petroleum.		Quicksilver Ore.		Salt, Rock.	
1893..	596,063	$1,744,353	148,442	$3,364,366	39,132	$97,830	652	$8,095	1,365	$36,592	1.2	$25	260,727	$280,259
1894..	625,662	1,897,079	144,724	2,900,092	42,526	98,950	1,341	13,413	1,600	39,791	*Nil.*	......	305,810	329,326
1895..	558,846	1,707,929	140,991	2,911,948	39,882	105,916	2,058	4,190	1,612	46,446	*Nil.*	......	278,326	327,861
1896..	587,306	1,786,786	138,398	2,998,433	43,614	102,384	738	4,540	1,512	46,867	*Nil.*	......	305,227	346,952
1897..	640,236	1,898,647	133,158	3,122,880	45,254	103,137	204	1,580	2,600	73,038	*Nil.*	......	310,755	358,697

MINERAL PRODUCTION—*Cont.*

METALLURGICAL PRODUCTION OF PRUSSIA. (a)

Year.	Silver and Gold Ore.		Zinc Ore.		Antimony and Alloys.		Arsenic Products.		Bismuth.		Cadmium. Kg.		Cobalt Works' Products.	
1893..	12	$18,665	787,048	$3,570,105	362	$39,898	710	$31,939	0.048	$174	5,284	$5,461	44	$190,315
1894..	6	9,281	727,645	2,567,053	376	40,034	1,147	57,377	3.092	7,117	6,052	6,335	46	142,675
1895..	12	23,626	706,179	2,643,292	924	73,660	1,788	98,363	1.000	1,750	7,047	9,809	45	144,103
1896..	15	18,748	729,725	4,254,351	1,239	113,886	1,750	130,539	(b)	......	10,667	20,435	69	220,624
1897..	6	4,901	663,739	4,219,761	1,552	152,744	1,924	143,779	(b)	......	15,531	44,157	51	166,015

Year.	Copper Matte		Copper.		Gold—Kg.		Iron.		Lead.		Litharge.	
1893..	831	$65,666	20,707	$5,045,292	739·09	$513,707	3,539,702	$41,118,869	85,866	$4,166,992	2,548	$139,304
1894..	670	26,223	21,966	4,670,025	687·88	479,215	3,744,116	43,048,291	92,379	4,357,035	2,647	141,079
1895..	766	48,548	22,069	4,980,829	653·15	455,159	3,778,775	43,961,347	99,586	4,983,733	2,458	137,403
1896..	369	27,098	25,683	6,398,780	755·19	524,958	4,470,551	55,144,390	102,413	5,621,335	2,544	153,432
1897..	274	13,157	25,997	6,696,664	1,087·07	756,184	4,892,059	64,780,081	108,880	6,556,528	1,999	123,737

Year.	Manganese and Alloys.		Nickel.		Paint, Mineral.		Salt, Common.		Silver—Kg.		Sulphur.		Sulphuric Acid.	
1893..	45	$24,588	893	$828,750	2,415	$55,171	266,478	$1,707,265	276,645	$7,274,252	1,871	$47,659	387,306	$2,977,769
1894..	48	25,210	522	480,675	2,250	39,375	280,511	1,777,592	279,781	6,102,001	1,888	42,912	420,965	3,098,586
1895..	66	34,300	698	616,143	2,059	38,863	278,962	1,757,837	245,527	5,376,046	1,819	34,085	416,239	3,894,871
1896..	86	42,225	822	713,455	2,110	43,750	288,300	1,848,166	288,467	6,534,058	2,011	41,041	456,781	2,984,098
1897..	11	58,000	898	760,562	2,400	48,750	274,888	1,643,473	289,960	5,907,572	2,091	42,860	484,289	2,890,702

Year.	Sulphates.										Tin.		Zinc.	
	Copper.		Iron.		Nickel.		Zinc.		Mixed.					
1893..	1,873	$135,231	7,134	$30,765	37	$11,237	2,704	$37,111	187	$6,296	909	$329,790	142,773	$11,806,066
1894..	1,858	137,956	7,600	27,788	46	13,975	2,746	38,914	148	4,813	842	252,600	143,354	10,435,118
1895..	1,922	138,905	8,627	30,185	91	24,325	2,641	39,266	112	3,598	849	254,592	150,123	10,397,219
1896..	2,568	196,592	8,618	29,383	128	31,360	3,102	46,723	126	4,266	804	231,111	153,082	11,775,568
1897..	2,689	225,877	9,064	33,037	167	42,327	3,583	54,601	225	8,548	912	282,384	150,616	12,608,525

(*a*) From *Zeitschrift für das Berg-, Hütten-, und Salinenwesen.* (*b*) Not reported.

MINERAL PRODUCTION OF SAXONY. (*a*) (METRIC TONS AND DOLLARS; 4 marks = $1.)

Year.	Arsenical, Sulphur, and Copper Pyrites.		Barytes.		Bismuth, and Bismuth, Cobalt and Nickel Ores.		Coal. (Not Including Lignite.)		Fluorspar.		Galena.	
1893....	12,617·7	$31,521	488·9	$1,223	3,634·8	$174,917	4,274,064	$10,128,936	2,425	$4,547	2,917·3	$156,019
1894....	13,045·2	33,491	291·3	815	2,980·4	175,513	4,123,227	9,435,934	1,880	3,525	3,412·6	175,745
1895....	8,900·3	26,573	284·5	1,045	3,001·1	153,056	4,435,328	10,224,100	995	1,866	3,346·7	172,617
1896....	8,834·9	27,080	574·2	1,554	3,168·7	141,735	4,536,603	10,778,005	805	1,509	2,844·5	155,848
1897....	9,402·0	29,112	218·4	616	3,030·6	131,952	4,571,685	11,563,214	592	1,110	(*j*)	....

Year.	Iron Ocher, Swabiah and Colored Earths		Iron Ore.		Lignite.		Limestone and Various Products. (*b*)		Manganese Ore.		Quartz, Mica and Uranium Ore.	
1893....	113·4	$1,021	1,650·7	$2,972	940,988	$663,831	(*j*)	$5,994	2·6	$31	408·4	$3,308
1894....	213·9	776	1,043·0	1,833	918,589	641,836	379·9	5,917	15·0	187	735·0	2,353
1895....	415·0	628	20·2	107	1,018,486	670,274	(*j*)	6,051	8·4	105	26·5	366
1896....	333·4	1,054	3,499·4	4,314	1,035,825	666,590	10·0	6,967	10·4	130	20·1	327
1897....	92·7	643	13,181·1	12,837	1,073,239	666,358	65·1	7,682	260·0	637	39·7	629

Year.	Silver Ores. (c)		Tin and Tinstone		Wolfram.		Zinc Blende.		Specimens	Totals.	
1893....	15,657·5	$681,743	44·2	$18,908	42·3	$7,866	374·2	$1,200	$1,401	5,255,429·1	$11,885,340
1894....	14,628·2	499,778	211·4	23,697	38·7	5,747	155·1	621	512	5,080,845·7	11,008,582
1895....	14,431·6	493,431	154·4	14,942	29·1	4,289	134·6	567	775	5,485,561·4	11,770,917
1896....	13,315·3	455,183	88·2	8,555	40·3	7,149	72·2	373	706	5,606,044·5	12,257,580
1897....	11,428·5	449,625	54·7	5,981	36·7	7,035	111·6	579	540	5,683,437·0	12,878,352

Year.	Manufactured Products.							
	From a Part of the Coal.				From a Part of the Lignite.			
	Briquettes—*M.*		Coke.		Briquettes—*M.*		Lignite Bricks—*M.*	
1893............	1,478	$6,981	73,329	$290,677	31,210	$19,801	64,698	$115,687
1894............	1,166	5,711	66,742	237,555	64,500	40,590	61,084	107,466
1895............	1,412	6,917	70,449	252,420	99,228	68,245	65,353	115,255
1896............	1,981	7,997	77,086	299,248	123,571	81,933	58,374	104,506
1897............	(*k*) 3,547	13,517	77,507	331,863	(*k*) 53,460	107,451	60,166	105,476

METALLIC CONTENTS OF THE MIXED ORES INCLUDED IN THE PRECEDING TABLE WHICH WERE DELIVERED TO THE FISCAL SMELTING WORKS AT FREIBERG. (a)
(IN METRIC TONS; 4 marks — $1.)

Year.	Total Mixed Ores.		Arsenic. Tons.	Bismuth. Kg.	Copper. Tons.	Gold. Kg.	Lead. Tons.	Nickel and Cobalt. Kg.	Silver. Kg.	Sulphur. Tons.	Zinc. Tons.
1893....	31,335·6	$869,041	406·6	(j)	20·1	(j)	4,361·9	655·5	33,253·7	4,841·6	188·6
1894....	31,099·1	709,397	424·7	(j)	20·8	(j)	4,619·5	558·0	31,635·6	5,354·2	134·9
1895....	26,679·5	694,131	350·1	(j)	14·9	0·1314	4,730·8	60·3	29,383·3	3,905·0	81·9
1896....	24,923·3	638,237	303·2	(j)	20·0	0·1736	3,887·5	130·1	26,268·0	3,784·9	122·6
1897....	20,772·2	477,664	353·1	(j)	9·6	0·1396	5,016·2	(j)	21,974·5	7,398·3	124·6

PRODUCTS SOLD BY THE FISCAL SMELTING WORKS AT FREIBERG AND THE COBALT WORKS AT SCHNEEBERG, GERMANY. (a)
(IN METRIC TONS AND DOLLARS; 4 marks — $1.)

Year.	Arsenical Products. (d)		Bismuth. Kg.		Cobalt Products.		Copper Sulphate.		Fine Gold. Kg.		Lead.			
											Products. (e)		Sheet.	
1893....	1,084	$87,648	2,101	$8,446	445	$639,878	2,024	$154,222	954	$664,853	5,456	$291,292	643	$35,444
1894....	1,229	109,113	2,402	9,725	404	442,737	2,140	164,558	957	667,141	4,603	236,044	1,160	63,414
1895....	1,217	108,826	2,529	4,738	406	353,768	1,833	187,557	850	592,497	6,608	345,129	592	32,622
1896....	882	90,626	1,112	1,765	509	431,318	2,547	197,421	847	590,227	5,810	333,744	880	50,958
1897....	2,127	127,122	1,624	3,801	1,255	581,559	3,756	162,881	889	619,831	10,841	332,652	1,268	41,626

Year.	Lead—*Continued.*				Nickel, Speiss.		Fine Silver—Kg.		Sulphuric Acid. (g)		Other Chemicals. (h)	
	Shot.		Other Manufactures of. (f)									
1893....	194	$12,300	454	$26,830	54	$3,575	95,103	$2,467,555	14,400	$144,368	690	$7,821
1894....	161	10,055	198	11,981	75	3,656	81,323	1,745,831	12,679	132,171	699	6,995
1895....	238	14,613	218	13,563	26	1,052	54,770	1,211,118	10,205	113,419	624	6,744
1896....	149	10,001	213	14,874	83	7,195	46,577	1,063,337	9,238	97,812	499	5,788
1897....	284	9,850	448	15,771	150	4,166	72,862	1,465,076	21,565	73,450	943	5,672

Year.	Zinc and Zinc Dust.		Clay and Chamotte Manufactures. (i)	Total Value.
1893....	183	$15,364	$12,634	$4,571,743
1894....	222	18,056	13,507	3,634,982
1895....	163	11,886	11,729	2,950,263
1896....	16	1,347	12,593	2,909,003
1897....	241	10,483	16,855	2,907,602

(a) From *Jahrbücher für das Berg- und Hüttenwesen im Königreiche Sachsen.* (b) Including arsenic powder, slags, washing-sand, granular ore, refuse stones and chippings. (c) Including silver-bearing lead, copper, arsenic, zinc and sulphur ores. (d) Including arsenious acid, red, yellow and white glass, and metallic arsenic. (e) Including assay lead, soft lead, antimonial lead, litharge, lead fume and tin-lead. (f) Including lead pipes, lead wire and various lead apparatus. (g) Including sulphuric acid of all kinds. (h) Including copperas and glauber salt. (i) Including tiles, plates, figured stone, muffles, clay and graphite crucibles, and assaying utensils. (j) Not reported. (k) Quantity stated in tons.

GREECE.

The statistics of mineral production in Greece are summarized in the following tables:

MINERAL PRODUCTION OF GREECE. (*a*) (METRIC TONS AND DOLLARS; 1 drachma =20 cents.)

Year.	Blende.		Calamine, Calcined.		Chrome Ore.		Emery.		Gypsum.		Iron Ore.		Iron Ore, Manganiferous.	
1894..	1,030	$16,700	19,800	$348,480	1,477	$18,400	3,570	$46,410	85	$1,540	121,570	$142,100	159,080	$314,120
1895..	2,710	45,230	21,321	395,250	2,740	34,120	3,055	45,825	113	2,030	150,210	179,250	152,123	342,670
1896..	1,750	31,520	20,950	301,680	1,600	22,400	3,650	77,745	120	2,150	225,600	314,700	166,850	417,120
1897..	3,118	58,826	22,817	489,357	563	8,087	3,024	64,411	51	918	260,828	375,592	182,850	499,180
1898..	1,139	2,320	30,906	562,490	1,367	18,000	3,932	83,754	83	1,500	287,100	413,230	213,938	641,840

Year.	Lead, Soft.		Lead Ore, Argentiferous.		Lead, Argentiferous.		Lead Fume.		Lignite.		Magnesite, Crude.	
1894..	479	$24,910	6,250	$147,500	13,955	$1,073,500	1,680	$13,700	14,320	$28,640	13,262	$59,340
1895..	7	490	1,580	147,400	19,838	1,656,500	1,406	12,020	17,748	55,500	11,096	33,288
1896..	480	28,900	3,200	137,600	14,700	1,325,095	1,550	10,510	14,000	28,000	11,600	34,800
1897..	520	33,904	2,815	125,174	15,946	1,524,977	2,785	26,333	20,018	40,000	11,311	39,500
1898..	305	21,655	(*b*)	....	18,888	1,832,136	2,655	27,248	17,310	34,620	14,829	54,100

Year.	Magnesite, Bricks.		Magnesite, Calcined.		Manganese Ore.		Millstones. Number.		Puzzolan.		Sea Salt.		Sulphur.	
1894..	873	$17,460	602	$6,020	9,319	$52,190	11,448	$5,535	28,550	$25,700	21,310	$332,436	1,528	$35,755
1895..	(*b*)	....	(*b*)	....	7,250	42,750	(*b*)	10,020	29,810	26,830	22,238	335,800	1,480	31,970
1896..	(*b*)	....	(*b*)	....	15,500	93,600	(*b*)	8,200	31,300	30,100	22,800	344,280	1,540	35,500
1897..	826	18,172	686	7,550	11,868	73,580	6,975	3,785	42,600	38,340	20,421	308,357	358	7,518
1898..	516	11,352	129	1,160	14,097	90,220	18,500	9,500	70,700	72,114	25,250	363,600	135	2,880

(*a*) Statistics communicated by E. Grohmann, Seriphos. (*b*) Not stated. (*c*) In 1898 there was produced also ocher, 40 metric tons, value, $104.

INDIA.

THE official statistics of mineral production in British India, and the imports and exports, are summarized in the subjoined tables:

MINERAL PRODUCTION OF INDIA. (*a*) (IN METRIC TONS AND DOLLARS; 4 rupees = $1.)

Year.	Alum.		Antimony Ore		Asbestos. Kg.		Borax.		Clays. (*c*)	Coal.		Copper Ore.	
1893......	(*b*)	....	(*b*)	....	(*b*)	....	(*b*)	....	$862,834-	2,384,947	$1,621,572	(*b*)	....
1894......	418	(*b*)	418	....	203	$808	367	$25,009	1,150,366	2,818,488	2,062,076	(*b*)	....
1895......	418	(*b*)	(*b*)	....	205	910	400	27,880	814,945	4,441,681	2,904,167	(*b*)	....
1896......	418	(*b*)	(*b*)	....	304	12	340	30,034	830,922	3,909,764	3,065,067	22	$210
1897......	418	(*b*)	(*b*)	....	254	10	280	23,622	(*b*)	4,128,330	3,116,647	33	69

Year.	Fullers Earth.		Gold—Kg.		Granite. (*c*)		Gypsum.		Iron Ore.		Laterite.	
1893......	(*b*)	......	6,311	$4,203,770	962,202	(*b*)	(*b*)	....	37,695	$46,274	*c*916,884	(*b*)
1894......	(*b*)	......	6,587	*c*2,868,040	817,482	*d*$89,498	3,220	$407	39,004	50,079	4,038,852	$54,698
1895......	40,360	$3,066	7,741	*c*3,511,607	1,188,840	*d*108,216	6,816	776	46,552	55,688	5,678,001	56,906
1896......	(*b*)	......	10,662	5,314,617	835,697	*d*124,777	7,605	1,017	13,996	22,225	5,867,878	70,235
1897......	(*b*)	(*b*)	12,100	5,581,439	633,302	*d*129,881	8,187	1,082	44,009	38,075	5,399,794	94,074

Year.	Limestone.		Manganese Ore		Mica.		Ochers.		Petroleum—*Gallons*.		Plumbago.	
1893......	538,047	(*b*)	3,180	(*b*)	195	....	(*b*)	....	10,359,812	$159,453	(*b*)	.....
1894......	860,086	$102,726	11,583	(*b*)	183	$29,077	5,673	....	11,089,597	220,141	1,623	.....
1895......	1,109,272	80,797	16,070	(*b*)	382	63,322	5,772	....	13,018,990	308,518	(*b*)	.....
1896......	1,135,780	128,291	57,782	$113,738	538	85,617	(*b*)	....	15,057,094	448,339	(*b*)	.....
1897......	1,861,309	195,678	74,862	....	703	49,751	(*b*)	....	19,128,828	565,948	61	.....

Year.	Salt.		Saltpeter		Sandstone.		Slate. (*c*)		Soapstone. (*c*)		Tin Ore. (*c*)		Trap Rock. (*c*)	
1893......	853,266	$958,651	(*b*)	..	761,034	....	58,724	......	(*b*)	....	(*b*)	....	221,366	....
1894......	1,318,136	1,586,335	21,300	..	683,871	$40,720	11,111	$7,664	745	$3,378	100	$21,740	67,214	....
1895......	1,120,780	1,225,180	(*b*)	..	831,921	32,811	60,165	......	343	972	22	4,464	29,038	....
1896......	1,043,171	1,369,821	(*b*)	..	1,001,245	80,521	22,840	9,708	789	2,719	83	20,361	107,956	$5,447
1897......	937,932	1,219,074	(*b*)	..	956,532	48,328	26,589	10,102	1,008	3,915	62	9,888	140,764	19,629

(*a*) From the *Review of the Mineral Production in India*; and the *Annual General Report Upon the Mineral Industry of the United Kingdom of Great Britain*. (*b*) Not stated in the reports. (*c*) Incomplete. (*d*) Represents only a part of the product, about one-third in 1894 and 1895, and one-half in 1896 and 1897.

MINERAL IMPORTS OF BRITISH INDIA. (*a*) (IN METRIC TONS AND DOLLARS; 4 rupees = $1.)

Year. (*b*)	Alum.		Arsenic.		Asphalt.		Brass.		Bricks and Tiles. *Number*.		Cement.		Chalk and Lime.	
1891-92...	3,760	$76,250	57	$4,982	1,452	$17,925	559	$161,052	3,027,412	$57,270	27,906	$289,182	2,845	$12,002
1892-93...	3,234	72,110	65	5,707	581	7,202	634	178,440	2,425,744	42,567	25,896	251,045	1,869	7,293
1893-94...	3,040	69,745	78	7,612	832	8,847	614	155,695	2,032,919	37,522	25,959	265,885	1,709	6,718
1894-95...	3,632	91,855	76	6,980	1,055	13,937	445	119,238	2,337,410	41,275	28,048	279,702	2,688	12,975
1895-96...	4,050	99,550	123	12,705	1,387	19,502	543	150,182	2,328,741	45,270	36,408	356,302	1,607	6,055
1896-97...	2,738	62,372	88	7,870	883	12,003	499	144,675	3,915,291	72,475	29,196	278,617	1,745	4,875
1897-98...	5,248	115,980	98	9,520	1,082	13,307	670	187,542	4,507,161	69,637	37,354	371,690	1,040	5,177

GREECE.

The statistics of mineral production in Greece are summarized in the following tables:

MINERAL PRODUCTION OF GREECE. (*a*) (METRIC TONS AND DOLLARS; 1 drachma = 20 cents.)

Year.	Blende.		Calamine, Calcined.		Chrome Ore.		Emery.		Gypsum.		Iron Ore.		Iron Ore, Manganiferous.	
1894..	1,030	$16,700	19,800	$348,480	1,477	$18,400	3,570	$46,410	85	$1,540	121,570	$142,100	159,080	$314,120
1895..	2,710	45,230	21,321	395,250	2,740	34,120	3,055	45,825	113	2,030	150,210	179,250	152,123	342,670
1896..	1,750	31,520	20,950	301,680	1,600	22,400	3,650	77,745	120	2,150	225,600	314,700	166,850	417,120
1897..	3,118	58,826	22,817	489,357	563	8,087	3,024	64,411	51	918	260,828	375,592	182,850	499,180
1898..	1,139	2,320	30,906	562,490	1,367	18,000	3,932	83,754	83	1,500	287,100	413,230	213,938	641,840

Year.	Lead, Soft.		Lead Ore, Argentiferous.		Lead, Argentiferous.		Lead Fume.		Lignite.		Magnesite, Crude.	
1894..	479	$24,910	6,250	$147,500	13,955	$1,073,500	1,680	$13,700	14,320	$28,640	13,262	$59,140
1895..	7	490	1,580	147,400	19,838	1,656,500	1,406	12,020	17,748	55,500	11,096	33,288
1896..	480	28,900	3,200	137,600	14,700	1,325,095	1,550	10,510	14,000	28,000	11,600	34,800
1897..	520	33,904	2,815	125,174	15,946	1,524,977	2,785	26,333	20,018	40,000	11,311	29,500
1898..	305	21,655	(*b*)	....	18,888	1,832,136	2,655	27,248	17,310	34,620	14,829	54,100

Year.	Magnesite, Bricks.		Magnesite, Calcined.		Manganese Ore.		Millstones. Number.		Puzzolan.		Sea Salt.		Sulphur.	
1894..	878	$17,460	602	$6,020	9,319	$52,190	11,448	$5,535	28,550	$25,700	21,310	$332,436	1,528	$35,755
1895..	(*b*)	....	(*b*)	....	7,250	42,750	(*b*)	10,020	29,810	26,830	22,238	335,800	1,480	31,970
1896..	(*b*)	....	(*b*)	....	15,500	93,600	(*b*)	8,200	31,300	30,100	22,800	344,280	1,540	35,500
1897..	826	18,172	686	7,550	11,868	73,580	6,975	3,785	42,600	38,340	20,421	308,357	358	7,518
1898..	516	11,352	129	1,160	14,097	90,220	18,500	9,500	70,700	72,114	25,250	363,600	135	2,880

(*a*) Statistics communicated by E. Grohmann, Seriphos. (*b*) Not stated. (*c*) In 1898 there was produced also ocher, 40 metric tons, value, $104.

INDIA.

The official statistics of mineral production in British India, and the imports and exports, are summarized in the subjoined tables:

MINERAL PRODUCTION OF INDIA. (*a*) (IN METRIC TONS AND DOLLARS; 4 rupees = $1.)

Year.	Alum.		Antimony Ore		Asbestos. Kg.		Borax.		Clays. (*c*)	Coal.		Copper Ore.	
1893......	(*b*)	....	(*b*)	....	(*b*)	....	(*b*)	....	$862,834	2,384,947	$1,621,572	(*b*)	....
1894......	418	(*b*)	418	....	203	$808	367	$25,000	1,150,366	2,818,488	2,062,076	(*b*)	....
1895......	418	(*b*)	(*b*)	....	205	910	400	27,880	814,945	4,441,681	2,904,167	(*b*)	....
1896......	418	(*b*)	(*b*)	....	304	12	340	30,034	830,922	3,909,764	3,065,067	22	$210
1897......	418	(*b*)	(*b*)	....	254	10	280	23,622	(*b*)	4,128,330	3,116,647	33	69

Year.	Fullers Earth.		Gold—Kg.		Granite. (*c*)		Gypsum.		Iron Ore.		Laterite.	
1893......	(*b*)	......	6,311	$4,203,770	962,202	(*b*)	(*b*)	....	37,695	$46,274	*c*916,884	(*b*)
1894......	(*b*)	......	6,587	*c*2,868,040	817,482	*d*$89,498	3,220	$407	39,004	50,079	4,038,852	$54,698
1895......	40,360	$3,066	7,741	*c*3,511,607	1,188,840	*d*108,216	6,816	776	46,552	55,688	5,678,001	56,906
1896......	(*b*)	......	10,662	5,314,617	835,097	*d*124,777	7,605	1,017	13,996	22,225	5,867,878	70,235
1897......	(*b*)	(*b*)	12,100	5,581,439	633,302	*d*129,381	8,187	1,082	44,009	38,075	5,399,794	94,074

Year.	Limestone.		Manganese Ore		Mica.		Ochers.		Petroleum—*Gallons.*		Plumbago.	
1893......	538,047	(*b*)	3,180	(*b*)	195	....	(*b*)	....	10,359,812	$159,453	(*b*)	.....
1894......	860,086	$102,726	11,583	(*b*)	183	$29,077	5,673	....	11,089,597	220,141	1,623	.....
1895......	1,109,272	89,797	16,070	(*b*)	382	63,822	5,772	....	13,013,990	308,518	(*b*)	.....
1896......	1,135,730	128,291	57,782	$113,738	538	85,617	(*b*)	....	15,057,094	448,339	(*b*)	....
1897......	1,861,309	195,673	74,862	....	703	49,751	(*b*)	....	19,128,828	565,943	61	.....

Year.	Salt.		Saltpeter		Sandstone.		Slate. (*c*)		Soapstone. (*c*)		Tin Ore. (*c*)		Trap Rock. (*c*)	
1893......	853,266	$958,651	(*b*)	..	761,034	....	58,724	......	(*b*)	....	(*b*)	....	221,366	....
1894......	1,318,136	1,586,335	21,300	..	683,871	$40,720	11,111	$7,664	745	$3,378	100	$21,740	67,214	....
1895......	1,120,780	1,225,180	(*b*)	..	831,921	32,811	60,165	......	343	972	22	4,464	29,038	....
1896......	1,043,171	1,369,821	(*b*)	..	1,001,245	80,521	22,840	9,708	789	2,719	83	20,361	107,956	$5,447
1897......	937,932	1,219,074	(*b*)	..	956,532	48,328	26,589	10,102	1,003	3,915	62	9,888	140,764	19,629

(*a*) From the *Review of the Mineral Production in India*; and the *Annual General Report Upon the Mineral Industry of the United Kingdom of Great Britain.* (*b*) Not stated in the reports. (*c*) Incomplete. (*d*) Represents only a part of the product, about one-third in 1894 and 1895, and one-half in 1896 and 1897.

MINERAL IMPORTS OF BRITISH INDIA. (*a*) (IN METRIC TONS AND DOLLARS; 4 rupees = $1.)

Year. (*b*)	Alum.		Arsenic.		Asphalt.		Brass.		Bricks and Tiles. *Number.*		Cement.		Chalk and Lime.	
1891-92...	3,760	$76,250	57	$4,982	1,452	$17,925	559	$161,052	3,027,412	$57,270	27,906	$289,182	2,845	$12,002
1892-93...	3,234	72,110	65	5,767	581	7,292	634	178,440	2,425,744	42,567	25,896	251,045	1,889	7,293
1893-94...	3,040	69,745	78	7,612	832	8,847	614	155,695	2,032,919	37,522	25,959	265,885	1,709	6,718
1894-95...	3,632	91,855	76	6,980	1,055	13,937	445	119,238	2,337,410	41,275	28,048	279,702	2,688	12,975
1895-96...	4,050	99,550	122	12,705	1,387	19,502	543	150,182	2,328,741	45,270	36,408	356,302	1,607	6,055
1896-97...	2,738	62,372	88	7,870	883	12,002	499	144,675	3,915,291	72,475	29,196	278,617	1,745	4,575
1897-98...	5,248	115,980	98	9,520	1,082	13,307	670	187,542	4,507,161	69,687	37,854	371,690	1,040	5,177

Year. (b)	Clays.		Coal.		Coke.		Copper and Copper Manufactures.		Earthenware and Porcelain.	Earthenware (Piping).		Explosives.
1891-92...	1,979	$17,647	735,782	$3,033,502	12,970	$92,502	25,963	$5,222,560	$683,982	1,941	$36,142	$384,705
1892-93...	1,292	11,042	644,904	2,754,477	12,736	94,502	21,374	4,421,892	628,982	1,888	31,937	398,445
1893-94...	1,394	11,495	553,710	2,341,092	11,437	90,315	30,028	5,956,295	628,990	2,629	51,230	521,440
1894-95...	1,328	12,242	821,926	3,580,238	12,624	93,247	17,988	3,779,540	482,108	2,268	41,017	507,015
1895-96...	2,910	30,670	758,799	3,390,590	14,626	112,635	c23,968	5,224,837	578,555	1,011	21,163	561,505
1896-97...	2,326	22,720	488,732	2,326,890	13,588	117,352	12,237	2,688,535	537,752	1,391	26,400	504,130
1897-98...	2,788	26,950	251,574	1,213,220	15,027	130,160	16,383	3,554,325	512,605	785	15,370	511,850

Year. (b)	Glassware.	Iron, Pig. (d)		Iron and Steel, Manufactures of.		Jewelry and Plate.	Lead.					
							Ore.		Pig.		Manufactures	
1891-92...	$1,319,610	9,336	$113,052	216,902	$6,839,467	$222,750	156	$8,497	1,110	$54,157	4,959	$364,815
1892-93...	1,677,010	7,540	94,145	175,657	5,964,440	233,608	238	14,632	900	40,627	4,606	334,067
1893-94...	1,971,200	7,504	93,278	233,765	7,516,075	336,825	215	12,370	1,025	45,775	4,954	382,675
1894-95...	1,549,990	d8,163	108,995	203,046	7,077,455	237,855	176	10,392	712	33,860	4,828	360,577
1895-96...	1,852,695	10,213	142,490	279,472	9,599,703	299,377	137	8,712	1,050	54,400	4,908	372,178
1896-97...	1,760,460	13,278	190,152	184,499	6,742,000	278,432	309	17,810	709	36,865	5,252	368,720
1897-98...	1,441,680	11,045	155,355	189,379	7,043,877	302,665	160	9,855	547	28,302	5,098	351,545

Year. (b)	Mineral Oils. *Gallons.*		Paints and Colors.		Precious Stones.	Quicksilver.		Stone and Marble.		Tin and Tin Manufactures.		Zinc and Zinc Manufactures.	
1891-92...	58,109,283	$5,920,352	7,710	$564,367	$355,910	127	$109,112	22,664	$86,267	1,972	$660,827	6,038	$481,535
1892-93...	67,085,968	6,289,685	8,516	555,697	518,350	163	134,687	35,521	72,115	1,910	740,085	5,871	464,692
1893-94...	86,608,538	8,188,962	8,872	621,120	361,027	143	110,885	25,050	68,652	2,332	818,930	6,681	482,187
1894-95...	53,441,425	4,854,915	8,888	556,977	1,422,748	54	46,897	41,548	91,502	2,459	801,750	4,304	307,680
1895-96...	66,647,921	6,949,358	9,685	704,942	1,630,878	160	152,098	43,094	72,250	2,556	823,527	4,564	312,750
1896-97...	68,421,148	7,720,697	9,696	674,395	1,321,700	118	102,992	24,880	62,930	1,849	551,342	2,277	175,380
1897-98...	87,325,026	9,129,480	9,238	670,680	1,126,225	143	119,050	36,745	96,485	1,962	554,078	3,185	244,312

(a) From the *Tables Relating to the Trade of British India.* (b) Fiscal years ending March 31. (c) Also imported 2 tons copper ore, value $287. (d) Also imported, 1893-94, 5 cwt. iron ore, value $7; 1896-97, 2 metric tons, $82, and in 1897-98, 52 metric tons, $735.

MINERAL EXPORTS OF BRITISH INDIA. (a) (IN METRIC TONS AND DOLLARS; 4 rupees = $1.)

Year. (b)	Borax.		Brass.		Coal and Coke.		Copper.		Earthenware and Porcel'n.	Glassware.	Jadestone.		Jewelry and Plate.
1891-92...	470	$35,730	137	$66,255	4,587	$11,602	109	$38,500	$9,082	$9,722	145	$130,305	$63,327
1892-93...	389	32,525	149	77,582	15,976	39,472	144	48,482	14,338	13,747	178	167,562	73,782
1893-94...	367	31,368	139	75,005	52,853	126,627	107	32,073	11,460	11,745	115	108,662	146,245
1894-95...	400	34,852	179	89,629	54,523	120,557	112	35,700	11,532	13,642	129	113,265	98,552
1895-96...	340	30,035	204	127,325	82,217	216,450	123	38,817	12,842	14,997	215	136,538	92,760
1896-97...	280	23,622	188	106,012	138,906	352,810	111	35,930	11,432	12,897	219	158,780	85,632
1897-98...	184	16,277	162	76,755	216,261	211,195	84	26,825	10,157	11,717	196	150,025	89,530

Year. (b)	Lead.		Manganese Ore.		Mica.		Mineral Oils. *Gallons.*		Precious Stones.	Saltpeter.		Stone and Marble.		Tin.	
1891-92...	318	$13,135	(c)	(c)	118	$107,177	43,388	$5,002	$26,230	19,770	$914,045	618	$18,882	73	$19,087
1892-93...	157	6,002	1,016	$200	169	108,587	71,780	6,652	30,832	21,883	1,097,350	698	20,110	48	14,227
1893-94...	34	1,417	1,676	3,300	246	195,460	67,377	5,247	24,183	16,354	845,255	374	10,712	69	21,167
1894-95...	129	5,390	6,518	12,832	261	159,437	48,391	2,522	21,162	17,931	1,028,362	226	9,050	71	19,972
1895-96...	57	3,115	23,122	45,515	452	268,052	315,118	38,817	37,287	21,425	1,339,865	315	10,555	13	3,437
1896-97...	210	8,427	48,087	94,660	652	288,340	191,424	28,225	34,670	26,845	1,430,410	499	14,072	86	20,322
1897-98...	117	5,127	80,090	157,507	590	267,142	15,563	2,860	30,970	21,218	996,862	995	16,010	49	10,945

(a) From the *Tables Relating to the Trade of British India.* (b) Fiscal years ending March 31. (c) Not reported.

ITALY.

THE official statistics of mineral and metal production in Italy, together with the imports and exports as reported in the official statistics of the Kingdom, are summarized in the following tables:

MINERAL PRODUCTION OF ITALY. (*a*) (IN METRIC TONS AND DOLLARS; 5 lire = $1.)

Year.	Alum.		Alunite.		Antimony Ore.		Asphaltum.		Boracic Acid, Crude.		Copper Ore.		Gold Ore.		Graphite.	
1892	1,695	$36,050	4,000	$3,840	621	$45,672	34,580	$168,184	1,089	$130,680	102,427	$552,288	6,612	$94,696	1,645	$3,778
1893	1,545	32,485	4,200	4,032	1,193	40,402	25,980	113,160	2,847	313,170	96,299	509,509	7,393	132,746	1,465	3,080
1894	1,065	20,680	6,000	4,800	1,504	45,059	60,493	280,678	2,746	236,156	92,886	445,629	7,748	132,782	1,575	2,520
1895	*b*1,115	21,000	7,000	5,600	2,241	40,254	46,713	204,750	2,633	184,310	83,670	367,516	7,099	129,887	2,657	8,599
1896	850	16,260	6,000	6,000	5,086	60,590	45,456	177,728	2,616	167,424	90,408	424,719	7,659	170,602	3,148	10,193
1897	1,030	21,180	6,500	6,500	2,150	34,864	55,339	189,655	2,704	173,056	93,377	431,229	10,723	178,010	5,650	11,300

Year.	Iron Ore.		Iron and Copper Pyrites.		Lead Ore.		Mineral Fuel. (*c*)		Manganese Ore.		Manganiferous Iron Ore.		Mercury Ore.	
1892	214,487	$554,431	27,670	$71,987	$33,310	$1,139,098	295,713	$426,073	1,243	8,359	4,622	$8,320	(*d*)	....
1893	191,305	356,533	29,460	72,227	29,004	813,107	317,249	434,701	810	6,549	8,805	14,968	19,450	$221,069
1894	187,728	417,831	22,638	59,269	29,822	758,422	271,295	378,792	760	4,700	5,810	9,296	15,022	203,443
1895	183,371	405,711	38,586	85,741	30,632	813,324	305,321	433,555	1,569	14,128	5,860	9,376	10,504	166,680
1896	203,966	507,973	45,728	108,825	33,545	892,865	276,197	396,372	1,890	20,450	10,000	20,000	14,305	147,570
1897	200,709	572,102	58,320	156,028	36,200	1,008,525	314,222	467,111	1,634	15,008	21,262	34,019	20,659	157,782

Year.	Petroleum.		Salt, Rock.		Salt from Brine.		Silver Ore.		Sulphur, Crude.				Zinc Ore.	
									Ground.		Fused.			
1892	2,548	$150,900	15,504	$49,619	8,217	$38,455	1,680	$345,898	15,500	$164,300	403,055	$7,680,033	129,731	$2,722,099
1893	2,652	159,010	16,790	56,910	8,602	39,811	1,236	235,595	16,300	136,920	401,371	5,786,415	132,767	2,093,415
1894	2,854	169,452	19,467	56,974	11,326	57,861	1,108	169,444	13,850	96,950	391,931	4,956,641	132,777	1,839,791
1895	3,594	186,099	18,710	56,019	10,605	54,171	870	128,273	(*d*)	....	370,766	4,134,588	121,197	1,542,905
1896	2,524	128,894	17,300	52,024	11,974	61,298	640	107,251	(*d*)	....	426,353	6,134,253	118,171	1,498,129
1897	1,932	98,456	19,801	54,404	11,725	63,100	405	85,625	(*d*)	....	496,658	8,995,647	122,214	1,656,065

REFINED PRODUCTS OF ITALY. (*a*) (IN METRIC TONS AND DOLLARS; 5 lire = $1.)

Year.	Aluminum. Sulphate.		Antimony.		Asphalt, Mastic and Bitumen.		Boracic Acid, Refined.		Borax, Refined.		Briquettes.	
1892....	1,890	$40,450	315	$53,822	9,270	$78,570	....	....	1,471	$191,230	612,069	$3,883,631
1893....	2,050	43,120	376	59,560	8,905	65,712	87	$11,745	1,099	131,916	559,990	3,614,530
1894....	2,480	45,960	351	57,626	7,820	49,826	170	22,100	1,141	136,872	527,904	3,207,265
1895....	2,950	53,760	423	56,986	14,491	75,920	253	29,807	944	101,930	451,470	2,527,564
1896....	2,390	41,020	538	72,440	12,490	75,280	253	32,838	943	79,279	422,409	2,081,645
1897....	2,310	40,160	404	57,072	18,644	89,125	260	31,200	990	83,160	549,050	2,768,350

Year.	Copper and Alloys.		Gold—Kg.		Iron, Pig.		Iron: Bar, Sheet, Pipe, Wire, etc.		Iron, Sheet, Tinned.		Lead.		Mercury.	
1892	6,039	$2,159,469	330·0	$204,579	12,729	$411,246	124,273	$6,278,691	320	$32,000	22,000	$1,144,000	325	$335,5
1893	6,911	2,386,247	362·0	224,203	8,088	230,921	138,046	6,826,843	2,500	250,000	19,898	994,900	273	264,7
1894	7,048	2,466,927	349·0	252,057	10,329	301,339	141,729	6,643,615	5,750	563,000	19,605	968,446	258	227,0
1895	8,508	2,675,064	280·0	199,453	9,213	207,948	163,824	6,962,837	5,860	555,360	20,353	1,078,920	199	180,3
1896	10,319	3,423,050	274·7	189,024	6,987	192,681	139,991	6,606,951	2,918	256,784	20,786	1,162,469	186	174,8
1897	8,545	3,059,546	316·0	218,862	8,393	181,763	149,944	7,176,951	6,500	546,000	22,407	1,462,909	192	192,0

Year.	Petroleum, Benzine, etc.		Salt, Sea.		Silver—Kg.		Steel.		Sulphur.			
									Refined.		Ground.	
1892....	1,573	$154,495	395,269	$800,831	43,000	$1,281,400	56,543	$2,756,499	59,370	$1,521,265	123,260	$3,074,019
1893....	2,613	258,676	397,506	929,522	40,095	1,130,679	71,380	3,397,426	70,707	1,306,120	94,624	1,912,678
1894....	1,640	193,520	402,515	830,382	58,626	1,289,772	54,614	3,353,852	71,295	1,152,576	90,561	1,654,465
1895 ...	4,191	440,953	448,335	957,529	44,189	1,007,460	50,314	3,794,256	75,329	1,125,494	91,517	1,473,093
1896....	2,734	296,147	422,555	856,072	38,075	882,228	65,955	4,874,539	71,072	1,198,447	89,292	1,602,797
1897....	3,392	279,533	429,253	885,637	45,313	917,670	63,940	4,156,850	85,872	1,874,642	69,178	1,541,510

(*a*) From *Rivista del Servizio Minerario.* (*b*) Including 120 tons of soda alum. (*c*) Includes anthracite lignite, fossil-wood, and bituminous schist. (*d*) Not reported.

MINERAL IMPORTS OF ITALY. (*a*) (IN METRIC TONS AND DOLLARS; 5 lire=$1.)

Year.	Antimony.		Arsenic—Kg.		Asbestos.		Baryta.		Bitumen.		Brick, Tile, etc.	
1893........	39·2	$8,389	1,000	300	193·0	$27,020	734·4	$8,813	4,023	$64,371	14,728	$103,856
1894........	30·4	5,715	1,200	348	856·7	102,804	629·1	16,357	9,380	150,080	15,336	107,649
1895........	41·6	7,238	1,200	348	530·8	47,772	507·9	13,205	6,075	97,205	14,623	102,193
1896........	38·2	6,112	(*h*)	....	850·8	76,572	549·4	14,284	11,892	190,277	18,504	123,228
1897........	65·7	10,512	3,200	928	618·9	61,890	577·9	15,025	1,632	26,120	19,086	133,608

Year.	Building Stone.		Cement and Hydraulic Lime.		Chalk.		Coal.		Colored Earths		Copper and Brass Tissue.	
1893........	*b*114,097	$474,316	19,555	$175,995	24,843	(*c*)	3,724,401	$18,622,005	907·2	$21,773	39·5	$40,665
1894........	58,375	250,940	14,100	126,900	14,866	(*c*)	4,696,258	22,072,413	1,149·6	27,590	43·1	43,100
1895........	49,775	213,764	15,120	136,080	17,102	$119,714	4,304,787	17,219,148	933·6	22,406	45·5	45,500
1896........	73,345	310,344	12,810	115,290	15,716	110,012	4,081,218	17,141,116	851·7	20,441	66·4	66,400
1897........	32,305	140,852	16,680	150,120	28,937	202,559	4,259,643	19,594,358	887·7	21,305	46·5	46,500

Year.	Copper Ore.		Glass and Manufactures.		Gold, Manufactures. Kg.		Graphite.		Iron Ore.		Iron, Cast.			
											Pig.		Wrought.	
1893........	3,166	$50,656	15,063	$1,168,546	1,066	$514,249	153·6	$9,216	1,809	$3,437	114,343	$1,829,486	9,867	$511,347
1894........	903	72,240	12,274	914,911	1,371	628,050	152·6	9,156	1,003	1,906	119,267	1,908,272	6,263	368,330
1895........	1,600	160,000	11,237	924,776	1,851	775,990	216·3	12,978	1,742	3,484	131,870	2,109,920	5,393	326,064
1896........	484	67,760	9,322	813,134	1,515	570,446	204·4	12,264	594	1,425	119,491	1,911,854	4,820	327,413
1897........	1,611	225,540	11,182	937,282	1,375	459,240	314·7	18,882	5,831	15,161	156,019	2,496,310	3,801	269,378

Year.	Iron and Steel. Plates, Rods and Manufactures.		Kaolin.		Lead.						Limestone. (*e*)	
					Ore. (*d*)		Pig and Scrap.		Manufactures.			
1893........	264,230	$7,139,218	1,914	(c)	10,047	$321,504	1,657	$89,489	105	$44,067	58,002	*f* $598,313
1894........	244,903	6,704,733	2,417	(c)	13,379	387,991	1,499	80,924	131	34,463	85,975	*f* 722,886
1895........	263,672	6,880,541	2,924	$23,392	8,901	284,832	947	53,043	233	42,639	73,220	512,540
1896........	252,202	6,919,787	3,775	30,200	9,730	408,660	1,166	69,978	192	39,524	100,522	708,624
1897........	88,895	5,179,932	5,719	45,752	14,854	653,576	1,178	75,424	247	54,481	82,997	560,758

Year.	Lithographic Stone.		Marble, Crude.		Marble and Alabaster.		Majolica Wares.		Mercury.		Millstones.	
1893........	186·5	$13,055	550	$7,150	244·6	$31,260	340·6	$16,916	13·1	$11,345	(*g*)	....
1894........	204·8	14,336	350	4,200	294·7	50,799	359·7	16,532	11·4	9,576	320	$20,480
1895........	158·4	10,888	344	4,128	202·6	28,434	416·3	18,109	10·3	9,373	275	17,600
1896........	208·9	14,623	710	8,520	252·8	29,262	438·7	20,293	30·2	27,180	329	21,056
1897........	148·0	10,360	2,114	25,368	244·4	23,997	476·1	24,637	30·3	27,270	773	49,472

Year.	Nickel and Manufactures of.		Nitrate of Soda.				Porcelain.		Potash, Ammonia and Caustic Soda.		Potters' Clay and Manufactures.	
			Crude.		Refined, and Potassium Nitrate.							
1893........	296·8	$430,737	11,281	$609,152	613	$46,580	499·1	$155,098	8,496·7	$476,064	865·4	$129,080
1894........	672·1	742,215	6,128	306,425	601	45,653	496·1	148,630	9,665·7	541,499	715·9	106,540
1895........	248·6	304,826	12,670	582,825	569	39,865	484·4	132,181	9,906·7	439,140	748·5	117,060
1896........	411·3	473,139	11,685	537,501	541	40,320	441·0	123,128	9,840·7	458,087	786·6	120,820
1897........	432·4	473,386	16,400	705,204	917	69,946	471·7	126,548	11,012·0	528,537	743·4	116,980

Year.	Precious Stones, Manufactures. Kg.		Slag.		Silver.				Sodium Carbonate.		Sulphates of Copper and of Zinc.	
					Bars, etc. Kg.		Manufactures. Kg.					
1893........	392·6	$1,833,771	8,700	$87,004	1,123	$26,952	4,676	$82,290	18,602	$483,655	9,026	$722,072
1894........	467·5	1,434,655	9,454	94,539	3,529	70,580	4,128	66,081	19,122	458,830	17,467	1,397,392
1895........	454·1	1,897,948	11,316	113,156	2,845	56,900	6,070	102,887	19,949	359,078	14,112	1,128,992
1896........	478·9	1,505,464	30,275	363,294	2,291	48,111	6,533	125,212	18,927	329,335	24,257	2,037,550
1897........	490·8	2,033,714	37,201	409,208	2,434	43,812	5,286	91,453	20,721	393,699	28,896	2,658,074

Year.	Sulphate of Potassium.		Sulphur, Crude and Refined.		Tin.				Terra Cotta.	
					Bars.		Manufactures.			
1893........	158·0	$3,480	24·8	$397	1,156·2	$531,852	88·3	$45,878	3,762	$116,907
1894........	346·4	19,398	267·4	3,851	1,297·6	503,469	77·3	34,558	2,274	71,922
1895........	262·2	14,683	255·2	3,318	1,484·0	516,432	87·3	35,520	2,533	79,194
1896........	431·1	24,142	4·7	72	1,763·5	581,955	90·6	35,156	2,675	84,622
1897........	562·5	30,375	236·4	4,492	1,520·4	501,732	81·4	31,560	2,167	69,621

Year.	Zinc.								All Other Ores.		Metals Not Specified.			
	Ore.		Oxide.		Bars.		Manufactures.				Crude.		Manufactures.	
1893........	(*h*)	......	509·2	$45,828	1,681·8	$164,816	3,029·5	$399,957	2,204	$52,896	876·2	$64,839	3·5	$1,820
1894........	13	221	593·2	59,320	2,229·9	196,231	3,248·9	385,087	108	2,160	947·5	70,115	4·2	2,184
1895........	22	330	576·4	57,640	2,378·0	190,240	3,136·4	344,390	297	5,940	933·3	69,064	17·9	9,308
1896........	(*h*)	......	540·2	54,020	2,596·0	233,640	3,482·4	421,918	1,894	75,760	1,134·6	90,768	21·1	10,972
1897........	(*h*)	......	570·0	57,000	3,278·2	295,038	3,556·5	421,153	380	15,200	1,508·7	114,661	30·3	15,756

(*a*) From the *Movimento Commerciale del Regno d'Italia.* (*b*) Including millstones. (*c*) Value included under limestone. (*d*) Includes argentiferous lead ore. (*e*) Including other stones and non-metallic minerals. (*f*) Representing the values of chalk, kaolin, limestone, and other stones and non-metallic minerals. (*g*) Included under building stones. (*h*) Not reported.

MINERAL EXPORTS OF ITALY. (*a*) (IN METRIC TONS AND DOLLARS; 5 lire = $1.)

Year.	Alabaster, Crude.		Ammonia, Potash and Caustic Soda.		Antimony.		Asbestos.		Baryta.		Bitumen.	
1893........	42·0	$1,092	74·2	$4,649	258·1	$55,233	3·6	504	24·8	$298	5,059·6	$30,358
1894........	114·0	2,964	72·4	4,408	278·6	52,378	70·3	8,436	24·6	639	6,091·3	36,548
1895........	176·1	4,579	42·7	3,125	254·1	44,213	29·3	2,637	1,053·8	27,399	9,535·5	57,213
1896........	289·0	7,514	88·2	7,453	361·0	57,760	129·7	11,673	65·6	1,705	13,729·2	82,375
1897........	268·8	6,989	65·6	5,492	270·9	43,344	169·9	11,898	143·2	3,723	15,310·0	244,960

Year.	Brick, Tile, etc.		Building Stone.		Cement and Hydraulic Lime.		Chalk.		Coal.		Colored Earths.	
1893........	89,826	$547,824	*b*29,175	$100,200	1,518	$12,114	4,183	(*c*)	12,655	$63,275	2,566·6	$56,465
1894........	107,197	610,273	39,288	140,427	2,416	19,328	5,002	(*c*)	13,692	64,352	1,704·5	37,499
1895........	127,177	727,210	26,722	89,905	4,534	36,272	3,719	$18,595	16,497	65,988	1,923·3	42,313
1896........	143,648	809,823	23,580	93,699	3,871	30,968	5,593	27,965	18,924	71,481	2,411·6	53,055
1897........	125,925	710,155	36,229	117,705	5,330	42,640	7,556	37,780	23,191	106,679	2,318·2	46,384

Year.	Copper Ore.		Glass and Manufactures of.		Graphite.		Iron Ore.		Iron, Cast.			
									Pig.		Wrought.	
1893........	12,697	$203,152	5,341	$1,026,899	2,158·6	$21,586	156,273	$296,919	456·7	$ 7,307	163·7	$8,157
1894........	7,789	112,162	5,551	1,069,004	2,285·1	18,281	159,205	302,489	1,052·9	16,846	264·2	13,241
1895........	5,859	87,885	6,327	1,198,147	3,153·3	25,226	164,367	328,734	965·9	15,454	430·1	27,519
1896........	3,603	61,251	5,867	1,066,612	3,727·1	29,817	187,059	448,941	1,378·5	22,056	427·3	26,006
1897........	2,408	40,935	6,022	1,015,225	4,164·2	45,806	207,619	539,809	498·3	7,973	1,434·3	67,100

Year.	Iron and Steel. Plates, Rods and Manufactures.		Kaolin.		Lead Ore, Including Argentiferous.		Lead and Lead Alloys, in Pigs.		Lead Manufactures.		Limestone. (*d*)	
1893........	1,704·1	$215,066	20	(*c*)	5,561	$177,952	1,570	$84,796	1,831	$143,303	40,203	$222,000
1894........	1,345·9	164,411	18	(*c*)	6,354	184,266	1,157	62,505	1,762	135,687	55,677	303,485
1895........	2,467·6	266,465	23	$184	6,622	211,904	2,153	120,596	1,492	125,716	45,139	225,665
1896........	1,819·5	227,305	49	392	4,731	151,392	1,419	85,140	1,441	141,019	37,676	188,280
1897........	7,148·1	662,990	93	744	4,747	161,398	2,790	178,586	1,410	141,672	48,972	244,850

Year.	Marble, Crude.		Marble and Alabaster.		Mercury.		Millstones.		Nickel, Alloys, and Manufactures of.		Nitrate of Soda, Crude	
1893........	72,845	$946,985	56,448	$2,084,587	235·0	$203,510	(*e*)	....	5·6	$15,264	32·5	$1,735
1894........	72,812	873,744	51,211	1,827,062	337·0	283,080	56	$3,584	14·0	20,665	24·6	1,230
1895........	75,462	905,544	58,271	1,796,680	213·1	193,921	39	2,496	1·7	3,422	26·9	1,237
1896........	80,750	969,000	68,639	1,937.828	155·1	139,590	156	9,984	34·7	34,229	51·1	2,351
1897........	83,081	996,972	62,750	1,864,862	236·3	212,670	51	3,264	26·9	25,270	151·3	6,536

Year.	Nitrate of Soda and Potash, Refined.		Porcelain.		Potters' Clay and Manufactures.		Salt, Sea and Rock.		Slag.		Silver—Kg.	
1893........	187·2	$14,227	90·8	$22,792	556·1	$79,800	157,362	$346,196	3,599	$3,599	51,856	$1,400,112
1894........	292·7	22,245	102·0	26,024	471·4	72,700	172,319	379,102	2,454	2,454	134,305	3,035,258
1895........	378·4	26,488	161·1	35,566	590·7	90,010	209,395	460,669	2,469	2,469	37,013	836,494
1896........	306·5	23,751	98·8	24,912	1,142·5	157,100	171,741	360,656	4,753	4,753	26,854	644,436
1897........	343·7	26,482	137·7	38,164	1,292·1	174,910	176,522	247,131	8,847	8,847	50,503	1,050,462

Year.	Sodium Carbonate.		Sulphates of Copper and of Zinc.		Sulphur, Crude and Refined.		Tin.				Terra Cotta.	
							Pigs, Bars, etc.		Manufactures.			
1893........	241·5	$6,279	17·4	$1,392	310,867	$4,973,878	13·5	$6,210	31·7	$16,307	2,202	$72,252
1894........	232·7	5,585	18·3	1,464	299,090	4,306,890	8·5	3,298	62·6	27,592	1,857	61,598
1895........	289·2	5,206	12·0	960	317,566	4,128,359	7·2	2,506	74·4	29,822	2,509	79,737
1896........	279·0	4,855	71·4	5,998	356,370	5,452,461	9·9	3,267	89·1	33,994	2,852	97,548
1897........	275·0	5,225	17·8	1,638	358,932	6,819,713	29·3	9,699	108·8	41,452	2,472	85,653

Year.	Zinc.								Other Ores.	
	Ore.		Oxide.		Pigs, Bars, etc.		Manufactures.			
1893........	113,218	$2,264,360	36·7	$3,303	(*f*)	....	7·2	$2,223	7,334	$ 88,006
1894........	123 268	2.095,556	34·3	3,430	(*f*)	....	5·9	1,561	6,939	69,330
1895........	111,227	1,668,405	19·3	1,930	3·9	$312	9·4	2,165	2,072	20,720
1896........	115,454	1,847,264	48·4	4,840	33·1	2,979	8·4	2,644	15,296	152,960
1897........	133,125	2,130,000	189·1	18,910	308·7	27,783	63·0	13,142	32,494	194,964

(*a*) From the *Movimento Commerciale del Regno d'Italia*. (*b*) Includes millstones. (*c*) Values included under limestone. (*d*) Includes chalk, kaolin, and certain other non-metallic minerals. (*e*) Included under building stone. (*f*) Not reported.

JAPAN.

THE mineral production of Japan is reported in the subjoined tables. The Japanese are rather backward in publishing their mineral statistics, those for 1896 being the latest that have been issued.

MINERAL PRODUCTION OF MINES IN JAPAN WORKED BY THE GOVERNMENT. (*a*)
(In metric tons.)

Fiscal Year.	Coal.	Copper.	Copper Sulphate.	Gold—Kg.	Lead.	Pig Iron.	Silver.
1890-91	6,699	24	347	260·49	(*f*)	2,192	7,026·1
1891-92	14,925	17	853	217·72	(*f*)	1,969	5,980·1
1892-93	21,945	275	63	273·77	70	1,417	8,584·1
1893-94	20,123	149	(*f*)	306·12	162	1,186	10,422·9
1894-95	22,506	277	(*f*)	372·15	50	1,210	10,991·1
1895-96	24,947	321	(*f*)	334·11	20	1,189	8,528·1

MINERAL PRODUCTION OF JAPANESE PRIVATE MINES. (*a*) (IN METRIC TONS). (*b*)

Year.	Alum, Refined	Antimony.		Arsenic Kg.	Asphalt, Refined.	Coal.	Copper.	Copperas	Gold.	Graphite.	Iron.
		Ore.	Metal.								
1892	(*f*)	1,343·5	42·2	1,025·0	(*f*)	3,185,860	20,521	(*f*)	396·19	601	17,583
1893	(*f*)	1,526·2	122·2	3,324·0	(*f*)	3,329,313	17,874	766·5	445·03	26	15,851
1894	(*f*)	1,170·6	403·3	5,387·0	(*f*)	4,280,247	19,665	904·4	499·83	1,091	18,258
1895	(*f*)	1,060·7	640·7	7,343·0	(*f*)	4,747,707	18,826	835·2	563·31	77	24,653
1896 (*g*)	(*f*)	827·6	517·3	6,043·4	(*f*)	5,100,005	20,114	868·6	963·69	215	27,420

Year.	Lead.	Lignite	Manganese Ore.	Mercury Ore. Kg.	Ocher, Red.	Petroleum, Refined.	Salt—*Hektaliters.*		Silver. Kg.	Sulphate of Copper	Sulphur, Refined	Tin.
1892	819·7	(*f*)	5,027	14,855	(*f*)	2,018	10,202,488	$3,584,093	51,026·8	(*f*)	20,511	41·4
1893	940·7	(*f*)	14,169	2,141	1·6	1,644	12,006,374	3,666,349	58,984·3	(*f*)	23,930	38·1
1894	1,414·8	(*f*)	13,368	*e*1,547	(*f*)	*c*22,837	11,411,275	3,483,078	68,798·9	(*f*)	18,787	38·7
1895	1,927·9	(*f*)	17,142	*e*481	(*f*)	*h*26,965,764	(*f*)	....	63,824·1	(*f*)	13,557	48·3
1896	1,957·7	(*f*)	17,967	*e*1,761·6	9·7	*c*31,314	(*f*)	....	64,302·7	(*f*)	12,540	50·0

(*a*) From *Résumé Statistique de l'Empire du Japon*, Tokio, and special reports from the Japanese Government. The data contained in the table of production of the Government mines are nearly exact, but those contained in the table of private mines, being the figures furnished by the mine owners themselves, are only an approximation and considerably less than the actual figures. (*b*) In making the conversions from the Japanese units to metric tons, hektoliters and dollars, the following relations. were employed: 1 kwan = [illegible] metric tons; 1 koku = 1·8039 hektoliters; 1 yen = $1; 1 kg. = 32·151 troy ounces. (c) Crude product. (d) Number of liters. (*e*) Metal. (*f*) Not reported. (*g*) Includes production of Government mines. (h) Number of liters.

MINERAL IMPORTS OF JAPAN. (*a*) (IN METRIC TONS AND DOLLARS.)

(1 kin = 0·00060104 metric tons; 1 yen = $1.)

Year.	Brass				Coal.		Coke.		Copper.		
	Crude and Old.		Tubing.	Other M'f'res.					Sheet, Rod, and Old.		Tubes, and oth'r M'f'res.
1893........	40	$23,071	$21,373	2,420	7,564	$81,707	5,106	$29,325	3	$1,479	$20,566
1894........	46	33,872	34,990	6,729	37,843	472,757	2,876	22,880	24	11,819	28,373
1895........	68	42,968	107,569	30,811	70,034	853,080	10,065	90,669	72	37,457	118,909
1896........	47	20,188	92,300	41,243	50,315	519,380	6,179	42,537	91	46,507	94,306
1897........	20	10,496	89,170	41,092	70,229	578,570	7,600	56,154	76	43,899	201,833

Year.	Iron.									
	Pig.		Bar and Rod.		Plate and Sheet.		Various Manufactures. (*b*)		Railway Materials	Pipe and Tubes, and other M'f'res. (*c*)
1893........	23,281	$446,477	22,630	975,787	8,160	456,644	17,654	$1,255,639	$147,641	$639,516
1894........	36,642	743,553	28,098	1,339,034	14,778	889,185	21,334	1,765,302	881,805	1,237,516
1895........	35,309	673,796	44,175	2,085,684	17,580	1,038,794	23,561	1,853,160	1,253,343	1,322,730
1896........	39,028	739,556	50,464	2,359,705	27,642	1,659,417	36,306	2,641,885	1,280,480	1,534,259
1897........	43,633	934,010	55,236	3,046,132	23,883	1,724,913	33,944	2,663,807	2,001,119	1,734,987

Year.	Iron and Steel for Rails.		Steel, including Wire and Wire Rope.		Steel Manu-fact'res	German Silver.		Kerosene. *Hektoliters.*		Lead.	
										Pig.	
1893........	22,517	677,108	4,524	499,746	$202,301	45·6	38,290	2,055,788	$4,401,041	1,768	$149,228
1894........	34,657	1,209,205	3,383	462,619	297,876	42·8	41,909	2,257,067	5,135,332	1,698	177,628
1895........	25,977	925,531	4,619	639,419	377,396	34·2	33,476	1,826,844	4,303,929	3,189	313,632
1896........	65,387	2,595,450	6,810	1,022,108	446,275	44·4	39,830	2,213,073	6,331,036	2,411	257,383
1897........	87.076	3,325,005	4,227	577,260	358,255	44·4	41,790	2,472,669	7,667,350	2,298	257,805

Year.	Lead—*Continued.*			Mercury.		Nickel.		Plumbago. Kg.		Tin.	
	Sheet and Tea.		Pipe.								
1893........	1,779	182,541	$5,428	76	$115,082	42	$60,761	1,615	$232	142	$90,578
1894........	2,134	255,755	9,986	78	127,541	25	39,451	7,539	1,889	281	179,068
1895........	1,664	224,546	40,310	75	141,302	29	37,731	10,718	2,160	323	191,306
1896........	2,147	259,593	144,504	78	130,303	28	30,933	36,620	8,075	270	146,412
1897........	1,972	269,638	76,910	80	156,527	44	50,321	155,595	15,413	264	145,174

Year.	Tin—Plate.		Yellow Metal.					Zinc.				All Other Metals and M'f'res
			Plates.		Rods.		M'nfac-tures.	Ingot.		Sheet and Old.		
1893......	579	$56,267	120	$44,204	3	$1,196	$287	715	$95,106	2,350	$367,584	$188,147
1894......	3,211	352,675	112	45,755	6	2,330	146	480	66,970	2,681	465,768	200,844
1895......	3,009	314,044	303	124,461	14	5,851	1,494	993	134,614	3,441	555,695	441,198
1896......	2,626	251,344	172	74,940	24	9,403	375	1,078	144,240	2,991	512,241	1,772,614
1897......	5,677	559,654	116	51,419	13	6,334	4,703	836	127,708	4,164	799,980	1,053,982

(*a*) From data specially furnished THE MINERAL INDUSTRY by the Japanese Government. (*b*) Includes old iron, hoop and band, roofing or corrugated and galvanized, nails, and galvanized telegraph wire, iron wire and wire nails (old), and other manufactures. (*c*) Includes also anchors and chain cables, screws, bolts, nuts, *etc.*, and other iron ware.

MINERAL EXPORTS OF JAPAN. (*a*) (IN METRIC TONS AND DOLLARS.)

(1 kin = 0·00060104 metric tons; 1 yen = $1.)

Year.	Antimony.			Brass.			Bronze.			Coal.			
	Metal.		M'factures.	Wire.		M'factures.	Ingot.		M'factures.	For Ships' Use.		Dust.	
1893..	1,316	$183,854	$75,678	151	$52,029	$22,819	416	$91,115	$190,501	417,229	$1,529,070	269,838	$641,758
1894..	1,612	254,261	72,366	232	85,850	43,690	306	70,609	183,688	442,596	1,904,156	238,100	757,039
1895..	1,450	289,935	180,277	177	79,381	44,533	104	27,602	229,291	476,247	2,195,078	283,902	982,710
1896..	624	83,806	104,010	222	108,612	35,526	54	14,570	180,318	588,963	2,636,324	299,102	980,704
1897..	1,577	267,548	63,331	147	72,414	66,388	27	8,373	183,022	582,031	3,229,024	188,132	859,514

Year.	Coal—*Continued.*		Coke.		Copper.							
	Other Kinds.				Ingot.		Old.		Refined.		Slab.	
1893....	842,942	$2,647,084	1,143	$7,219	7,102	$2,033,820	22·0	$5,480	5,115	$1,597,926	3,110	$981,802
1894....	1,047,651	3,917,266	1,197	7,763	5,707	1,799,435	7·0	1,868	5,820	1,945,456	3,651	1,155,863
1895....	1,114,123	4,426,400	4,382	37,487	3,942	1,340,584	0·09	82	5,527	2,123,707	5,067	1,698,877
1896....	1,350,458	5,312,227	3,399	29,114	6,756	2,423,116	0·82	113	6,134	2,461,039	1,644	594,448
1897....	1,366,498	7,457,262	2,950	45,884	2,741	1,075,945	6·81	2,077	6,172	2,680,807	5,045	2,017,947

Year.	Copper—*Continued.*			Iron Wares.	Other Metal Wares.	Manganese Ore.		Sulphur.		Other Mineral Products.	Total.
	Wire.		M'factures.								
1893....	71	$26,169	$71,984	42,555	176,254	18,089	$212,884	10,441	$238,832	$4,202	10,804,023
1894....	69	26,975	134,970	62,142	170,894	17,634	198,810	12,683	244,542	2,245	13,059,614
1895....	60	31,145	101,342	96,875	314,530	16,496	196,509	15,894	296,136	660	14,651,701
1896....	72	33,717	185,079	105,481	358,126	20,986	274,434	12,825	308,588	1,920	16,161,052
1897....	93	46,947	223,234	178,037	485,862	14,665	205,318	9,209	321,341	1,396	19,525,094

(*a*) From data furnished to The Mineral Industry by the Japanese Government.

MEXICO.

The Mexican government collects no statistics of production; those compiled by The Mineral Industry, which represent all the important substances, will be found under the respective captions "Lead," "Copper," "Coal," etc.

MINERAL EXPORTS OF MEXICO. (*a*) (IN METRIC TONS AND MEXICAN DOLLARS.) (*e*)

Year.	Antimony Ore.		Asphalt.		Building Material.		Coal.		Copper.				
									Ore.		Ingot.		Argentiferous.
1894......	80	$2,836	17	$434	263	$3,605	45,911	$189,824	1,904	$63,921	8,454	$1,288,699	$102,862
1895......	600	14,281	(*b*)	....	1,355	5,774	61,686	255,355	3,006	523,650	20,429	3,387,586	(*b*)
1896......	3,231	32,093	17	2,134	9,934	45,418	75,541	312,738	144	5,210	20,659	4,008,404	(*b*)
1897......	5,873	71,835	10	282	1,808	7,858	105,298	434,624	1,094	176,391	16,858	3,329,331	(*b*)
1898......	5,932	98,815	(*b*)	....	597	3,561	118,553	486,596	13,146	3,738,738	10,362	2,314,790	(*b*)

Year.	Gold.						Guano.		Gypsum.		Iron Ore.	
	Coin.	Ore.	Bullion.	Mixed.	Cyanide	Sulphide.						
1894......	$243,532	$40,460	$1,857,563	$233,728	(*b*)	(*b*)	(*b*)	....	1,488	$13,762	27	$750
1895......	175,098	103,773	4,920,504	(*b*)	$31,231	$3,026	1,179	$34,037	1,340	4,270	1,012	2,572
1896......	261,078	206,874	5,533,789	(*b*)	161,784	44,890	1,001	15,663	2,050	10,250	1	30
1897......	202,223	365,226	6,220,765	(*b*)	226,986	33,916	1,098	14,615	2,095	7,775	(*b*)	......
1898......	(*b*)	1,037,202	6,493,735	(*b*)	294,730	64,061	2,267	28,983	1,650	8,250	21	208

Year.	Jewels and Precious Stones. Grams.		Lead.					Marble.		Pearls—*Carats.*		Plumbago.	
			Ore.		Base Bullion.		Argentiferous						
1894......	18,489	$36,173	294·0	$9,382	16,606	$773,133	$5,247	1,220	$110,202	846	$19,230	663	$9,171
1895......	81,396	54,891	528·0	30,165	50,122	2,371,574	(*b*)	1,810	229,495	304	9,500	794	9,121
1896......	1,357	1,113	167·0	4,820	48,663	1,340,785	(*b*)	2,258	210,836	225	5,500	795	7,290
1897......	3,880	5,628	1·9	87	60,029	3,006,821	(*b*)	2,176	185,004	1,420	6,000	759	6,698
1898......	11,241	27,867	(*b*)	....	60,918	3,291,014	(*b*)	469	46,724	*d*327	3,500	(*b*)	....

Year.	Quicksilver. Kg.		Silver.							Tin.	
			Coin.	Ore.	Slag.	Bullion.	Sulphide.	Cyanide	Mixed.		
1894......	410	$600	$16,251,832	$10,265,625	$65,008	$10,431,573	$749,512	(*b*)	$2,169,664	42·0	$10,015
1895......	2,378	3,910	18,300,553	10,977,079	72,590	22,178,294	555,475	$14,649	(*b*)	39·0	22,911
1896......	450	770	18,737,331	9,971,053	64,121	28,565,843	1,495,306	38,049	(*b*)	5·0	2,265
1897......	1,845	3,150	21,925,347	11,401,176	39,800	35,775,125	1,663,581	123,246	(*b*)	0·6	40
1898......	(*b*)	......	(*b*)	11,048,358	46,488	37,137,599	1,663,501	257,342	(*b*)	(*b*)	......

(*a*) From the *Estadistica Fiscal.* The figures for the calendar years were arrived at by combining those of the successive semesters of the different fiscal years. Additional exports in 1898: salt, 1,533 metric tons, $13,449; zinc ore, 1,001 metric tons, $10,016. (*b*) Not reported. (*d*) There was also exported pearls, 42 g., $1,000. (*e*) The average value of the Mexican dollar in New York in 1893 was $0·5000; in 1894, $0·5000; in 1895, $0·5218; in 1896, $0·5228; in 1897, $0·4671: and in 1898, $0·4641.

NORWAY.

The official statistics of mineral production, imports and exports, are summarized in the following tables:

MINERAL PRODUCTION OF NORWAY. (*a*) (IN METRIC TONS AND DOLLARS; 1 krone—27 cents.)

Year.	Apatite. (*b*)		Cobalt Ore.		Copper Ore.		Feldspar. (*b*)		Iron Ore.	
1891........	4,258	$96,713	187	$13,500	20,939	$192,240	12,257	$57,915	1,464	$2,700
1892........	2,427	45,873	123	8,910	18,888	140,130	5,936	28,836	860	1,485
1893........	1,513	26,541	123	12,150	21,907	175,203	3,506	17,037	800	1,485
1894........	2,086	36,720	89	8,100	20,226	192,240	7,836	35,910	(*c*)	....
1895........	1,601	23,760	45	4,050	21,869	218,835	9,780	44,901	1,250	2,349

Year.	Nickel Ore.		Pyrites, Iron and Copper.		Silver Ore and Native Silver.		Zinc and Lead Ore.		Cobalt—Kg.	
1891........	12,839	$33,885	49,048	$217,755	1,277	$155,655	498	$4,725	7,000	$18,900
1892........	6,959	19,575	58,570	208,440	1,121	132,300	576	5,400	4,500	13,500
1893........	2,397	6,480	53,574	190,080	890	117,045	(*c*)	....	5,000	16,200
1894........	2,355	5,400	70,859	268,110	750	100,710	200	1,890	(*c*)	....
1895........	494	1,080	49,005	194,400	490	95,850	(*c*)	....	2,500	8,100

Year.	Copper and Copper in Mattes.		Gold.	Iron, Pig and Cast.		Iron, Bar and Steel.		Nickel.		Rutile. Kg.		Silver—Kg.	
1891........	677	$181,710	$6,480	421	$5,454	593	$37,152	136	$100,710	(*c*)	....	4,677	$151,200
1892........	631	143,370	9,855	610	7,020	488	25,110	97	54,810	(*c*)	....	4,811	130,680
1893........	786	168,750	5,940	336	3,510	439	23,760	113	70,605	7,000	$1,620	4,773	117,855
1894........	907	178,200	1,080	287	3,240	433	19,980	103	63,450	19,000	4,050	4,760	102,600
1895........	958	184,950	2,160	348	3,942	379	19,440	17	10,530	28,000	8,100	5,000	105,300

(*a*) From *Tabeller vedkommende Norges Bergvaerksdrift*, and from *Statistisk Aarbog for Kongeriget Norge*, 1896, published by Det Statistike Centralbureau, Christiania. There were also produced in 1894, 7,000 kg. of molybdenite, value $3,780, and in 1895, molybdenite 4,000 kg., $2,160, and chrome ore 190 metric tons, $2,295. (*b*) Export returns, which approximately represent the production. (*c*) Not reported.

MINERAL IMPORTS OF NORWAY. (*a*) (IN METRIC TONS AND DOLLARS; 1 krone = 27 cents.)

Year.	Borax and Boracic Acid. Kg.		Bricks. *Thousands.*		Cement and Hydraulic Lime. Hektoliters.		Coal, Coke, and Cinders—*Hektoliters.*		Copper and Brass.			
									Plates, Bolts, and Bars.		Wares.	
1893....	40,372	$6,534	1,178	$37,854	91,838	$130,167	11,251,929	$3,402,594	1,331	$348,597	342	$252,936
1894....	32,814	4,428	887	41,148	81.013	112,644	13,259,415	3,938,058	1,362	338,337	358	257,877
1895 ...	40,154	5,427	644	38.988	80,861	109,161	14,012,194	3,783,294	1,262	323,595	417	293,841
1896....	38,305	4,671	3,691	54,810	16,028	137,052	14,201,090	3,642,570	1,074	290,061	479	330,696
1897....	44,495	4,941	4,009	56,295	18,734	174,258	15,374,572	4,234,167	1,140	307,827	591	775,089

Year.	Earthenware.		Porcelain, Glazed and Unglazed.		Tiles. *Thousands.*		Glass and Glassware.		Iron.					
									Pig and Cast.		Bars, Hoops, Ships' Knees, etc.		Ships' Anchors and Cables.	
1893....	797	$120,447	106	$44,280	1,178	$16,956	3,910	$389,799	17,070	$218,916	22,974	$765,045	1,271	$82,323
1894....	1,043	157,653	116	48,627	887	12,852	3,729	407,268	20,876	270,540	23,559	742,500	1,246	77,409
1895....	1,037	156,816	108	45,036	644	9,261	3,343	407,592	19,654	254,718	24,985	790,695	1,152	71,523
1896....	1,097	207,279	119	49,923	881	11,880	3,729	449,577	20,201	252,801	26,552	890,433	1,090	67,889
1897....	1,348	254,799	199	68,877	521	12,760	4,262	548,802	21,606	289,791	29,038	1,013,364	1,367	87,225

Year.	Iron—*Continued.*						Steel.		Locomotives and Machines.	Lead, in Pigs and Sheets.		Lead White and Zinc Oxide.	
	Rails.		Nails, Spikes, Bolts, Screws, etc.		Other Manufactures of Iron.								
1893....	8,482	$251,910	1,144	$71,087	11,809	$1,496,178	1,897	$107,595	$1,207,035	729	$43,281	808	$73,994
1894....	10,878	293,706	1,482	80,028	14,486	1,554,768	3,058	165,105	1,288,907	797	45,225	931	80,406
1895....	10,337	279,099	1,796	96,984	15,588	1,805,652	3,654	197,289	1,263,438	657	38,988	1,068	89,424
1896....	4,315	116,505	1,760	104,544	19,358	1,880,415	2,754	156,141	1,895,994	653	40,527	1,192	99,738
1897....	7,637	247,428	2,097	153,738	27,422	3,367,359	4,350	246,645	2,058,885	848	54,972	1,119	93,695

Year.	Paraffin Oil, Petroleum, etc.		Potash.		Salt. *Thousands.*		Saltpeter.		Soda.		Sulphur. (*b*)		Tin, in Blocks, etc		Zinc, in Plates, Bars, etc.	
1893...	24,232	$654,264	662	$53,649	1,680,184	$635,121	290	$23,490	5,878	$87,291	5,415	$146,205	87	$40,095	1,027	108,159
1894...	29,324	791,748	780	63,207	1,611,769	609,255	404	32,697	5,002	67,527	5,991	145,584	85	32,130	864	81,673
1895...	27,978	982,044	562	47,034	1,239,334	468,477	227	17,199	5,234	56,511	7,271	176,688	99	34,884	978	87,129
1896...	35,823	1,160,676	945	79,083	1,179,198	429,813	308	23,247	5,156	55,674	9,347	227,124	142	49,788	1,101	107,055
1897...	39,810	1,021,113	919	76,928	1,645,716	630,963	277	15,708	5,492	59,313	10,701	260,037	236	82,917	1,102	107,088

MINERAL EXPORTS OF NORWAY. (*a*) (IN METRIC TONS AND DOLLARS; 1 krone = 27 cents.)

Year.	Apatite.		Bricks—*Number.*		Cobalt Ore—Kg.		Copper.					
							Ore.		Ingot.		Old Metal.	
1892........	2,427	$45,873	446,472	$1,917	98,670	$26,649	1,845	$24,921	350	$80,352	729	$112,212
1893........	1,513	26,541	3,510,772	14,229	35,200	9,504	11,874	144,261	378	81,702	961	137,531
1894........	2,086	36,612	3,650,552	12,825	68,700	18,549	14,811	151,956	397	80,325	814	98,925
1895........	1,601	23,760	5,016,158	20,304	30,000	8,910	20,283	219,051	705	142,722	810	109,464
1896........	1,160	17,226	10,008,823	48,654	33,800	8,208	30,367	409,941	565	129,573	712	86,454
1897........	872	12,960	11,711,511	59,454	45,000	9,720	15,111	204,012	552	131,031	670	81,429

Year.	Feldspar.		Glassware.		Iodine—Kg.		Iron.					
							Ore.		Pig and Old.		Bars and Hoops.	
1892........	5,936	$28,836	1,858	$119,664	6,298	$35,721	69	$135	2,980	$34,614	155	$6,264
1893........	3,506	17,037	1,495	101,115	4,015	22,761	510	972	4,767	54,054	62	2,430
1894........	7,836	35,964	1,159	107,892	10,074	57,132	1,607	3,024	4,679	53,055	462	17,469
1895........	9,780	44,901	1,153	89,721	2,683	15,201	1,545	2,916	8,188	92,826	19	702
1896........	12,223	54,459	1,231	94,581	1,959	11,097	2,051	3,888	5,493	62,289	12	459
1897........	17,392	79,839	1,432	78,894	2,395	11,151	4,242	8,019	4,631	53,136	56	2,376

Year.	Iron—*Continued.*		Steel.		Machinery.		Nickel.			
	Nails and Spikes.						Ore—Kg.		Metal—Kg.	
1892........	8,916	$741,123	119	$ 8,046	649	$175,311	(c)	....	56,444	$16,787
1893........	8,559	692,307	104	6,696	402	108,621	340,060	$2,295	177,253	50,247
1894........	8,081	625,509	160	9,504	1,578	426,087	160	(c)	146,384	40,311
1895........	10,408	781,758	133	7,884	1,100	297,055	(c)	....	(c)	....
1896........	10,664	747,819	132	7,830	714	192,915	(c)	....	(c)	....
1897........	9,097	636,552	167	9,936	992	267,840	(c)	....	(c)	....

Year.	Porcelain, Glazed and Unglazed.		Pyrites.		Silver Ore—Kg.		Stone, Ashlar.		Whetstones.	
1892........	192	$20,682	47,748	$167,589	474,000	$33,264	45,598	$221,616	129	$8,748
1893........	206	22,221	52,757	170,937	154,200	9,369	48,691	243,216	109	7,344
1894........	225	24,273	40,771	132,111	202,200	10,908	52,344	247,320	189	12,771
1895........	286	30,888	39,710	128,655	136,800	5,535	54,888	237,114	169	11,427
1896........	365	39,420	41,062	140,265	173,500	7,261	66,233	312,957	205	13,851
1897........	260	27,378	70,552	238,113	118,900	3,861	74,492	367,065	112	7,587

(*a*) From *Statistisk Aarbog for Kongeriget Norge.* (*b*) Inclusive of flowers of sulphur. (*c*) Not reported.

PORTUGAL.

The mineral statistics of Portugal are summarized in the subjoined table, for which we are indebted to the courtesy of Sehnor Severiano Augusto da Fonseca Monteiro, chief of the Reparticao de Minas, Ministerio das Obras Publicas of Portugal.

The statistics of mineral production in Portugal in years prior to 1893 may be found in THE MINERAL INDUSTRY, Vol. II. and Vol. VI. It will be observed that the mineral Industry of Portugal has not yet attained much importance, the output of copper and copper ore forming the chief part of the total. This is principally the production of the Mason & Barry Co., Ltd., an English company, whose report for 1898 is to be found under the caption "Copper," elsewhere in this volume.

MINERAL PRODUCTION OF PORTUGAL. (*a*) (IN METRIC TONS AND DOLLARS; 1 milreis — $1·08.)

Year.	Anthracite. (*b*)		Antimony Ore.		Copper Ore.		Copper (Cement).		Copper Pyrites.	
1893......	18.213	$37,705	760	$55,826	(*c*)	....	6,886	$603,143	38,710	$59,743
1894......	10,017	32,839	803	48,952	321	$3,954	6,924	498,675	44,005	132,707
1895......	8,787	35,820	753	36,789	202	3,822	5,055	498,783	32,814	99,020
1896......	8,743	17.944	595	28,917	436	10,535	3,453	467,897	43,665	139,525
1897......	7,996	32,816	418	28,063	241	4,496	3,304	408,262	60,473	178,014

Year.	Copper Pyrites (Dressed).		Galena.		Galena (Argentiferous).		Gold Ore.		Iron Ore.		Lignite. (*b*)	
1893......	143,942	$124,055	1,261	$22,743	(*c*)	....	(*c*)	....	8,383	$1,811	4,632	$13,247
1894......	203,241	245,621	1,713	27,445	(*c*)	....	(*c*)	....	620	335	9,649	25,002
1895......	162,499	227,025	1,346	27,620	25	$513	222	$3,159	(*c*)	....	10,309	22,090
1896......	163,775	224,634	1,333	30,236	(*c*)	....	(*c*)	....	(*c*)	....	8,000	18,306
1897......	210,265	287,254	2,180	56,300	(*c*)	....	17	12,085	(*c*)	....	9,342	24,215

Year.	Manganese Ore.		Silver—Kg.		Tin Ore.		Wolfram Ore		Zinc Ore.	
1893......	4,080	$34,520	(*c*)	......	45	$11,066	(*c*)	.. ....	28	$302
1894......	5,246	38,866	76	$741	26	6,270	18·9	$2,284	(*c*)	......
1895......	1,240	8,250	(*c*)	......	3	528	12·4	2,786	(*c*)	......
1896......	1,494	14,079	(*c*)	......	6	756	14·0	3,640	(*c*)	......
1897......	1,652	11,596	793	2,042	*d* 9	2,550	29·4	7,263	(*c*)	......

(*a*) From a report specially furnished THE MINERAL INDUSTRY by Senhor Severiano Augusto da Fonseca Monteiro, Chief of the Department of Mines of the Ministerio das Obras Publicas. (*b*) Consumed in the country. (*c*) No report, probably none. (*d*) Includes metallic tin and cassiterite.

RUSSIA.

The official statistics of mineral production, imports and exports, are given in the subjoined tables. The Russian official statistics of production are somewhat tardy in appearance, the latest being those for 1896. Statistics of the production of gold, silver, copper, zinc, petroleum, quicksilver and pig iron for 1897 and 1898 have been reported, however, especially to The Mineral Industry, and will be found under their respective captions elsewhere in this volume.

It will be observed that the statistics of gold production in 1896 in the following table differs from those used in the summary of the world's gold production on a previous page of this volume. Reference should be made to earlier volumes of The Mineral Industry for an explanation of this discrepancy.

MINERAL PRODUCTION OF RUSSIA. (*a*) (IN METRIC TONS AND DOLLARS.)

Year.	Asbestos.		Asphaltic Mastic		Chrome Iron Ore.		Cobalt Glance.		Coal.		Copper.		Gle:uber Salt.		Graphite.	
1892..	1,239	$8,000	15,442	$118,760	3,007	$7,509	20	......	6,945,994	$8,416,720	5,315	$1,476,427	2,481	$18,099	82	$4,000
1893..	1,059	6,836	14,925	114,800	14.585	35,101	3	......	7,613,724	9,225,200	5,463	1,586,526	5,037	36,754	311	15,200
1894..	570	3,400	16,054	176,422	7,537	18,760	9	......	8,762,340	11,473,600	5,419	1,320,800	4,014	28,800	...	......
1895..	1,131	6,800	18.783	143,542	21,013	51,316	3	......	9,099,476	12,150,400	5,854	1,429,516	4,207	15,612	...	......
1896..	1,274	7,666	18,174	135,783	6,682	16,316	4	......	9,377,551	13,198,424	5,832	1,424,076	5,198	9,600	...	......

Year.	Gold—Kg.		Iron, Cast.		Iron Pyrites.		Lead.		Manganese Ore.		Naphtha.		Phosphorites.	
1892..	42,997	$20,921,600	1,071.772	$18,357,586	14,109	$45,200	883	$49,160	203,237	$370.800	4.896,179	$1,990,686	23,228	40,445
1893..	44,865	21,869,708	1,148,902	19,100,000	16,271	51,200	844	46,922	268,621	669.100	5,790,692	2,781,851	13,706	35,994
1894..	42,932	20,985,200	1,332,465	22,200,000	19,801	48,000	743	36,000	243,469	490,800	5,161,552	3,916,800	14,072	31,680
1895..	41,097	18,812,993	1,452,337	22,312,800	11,042	31,200	412	20,000	203,081	388,522	7,056,330	11,490,800	6,327	18,682
1896..	37,199	16,845,122	1,620,812	25,315,436	11,550	32,435	262	12,775	208,025	398,000	7,105,868	12,588,860	3,776	11,035

Year.	Platinum—Kg		Porcelain Clay.		Quicksilver		Salt.		Silver—Kg.		Sulphur.		Tin.		Zinc.	
1892..	4,570	$558,400	4,149	$16,874	343	$376,000	1,458,508	$2,404,000	11,187	$304,000	408	$4,920	9	$3,520	4,369	$416,000
1893..	5,094	917,231	3,197	12,848	201	220,484	1,351,056	2,224,234	9,500	276,696	589	7,200	7	2,501	4,501	428,604
1894..	5,202	890,400	7,742	30,400	196	153,200	1,354,218	1,723,200	7,813	227,200	1	18	4	1,440	5,014	367,235
1895..	4,406	883,960	25,330	101,200	434	371,000	1,540,194	2,383,743	7,878	239,088	85	2,816	8	5,068	5,029	360,000
1896..	4,930	1,083,600	6,099	24,575	491	432,058	1,346,247	2,079,388	7,813	235,172	437	5,339	2	476	6,257	458,369

(*a*) From the Russian official mining report "Sbornik Statisticheskekh Svedenie o Gornozavodskoi Promyshlennostye Rossie v Zavodskom Godu," St. Petersburg. (*b*) In the Russian reports the quantities are stated *in poods* and the values in silver rubles. In making the reductions the following relations were employed: *1 pood* = ·01638 metric ton; 1 silver ruble = 40 cents.

MINERAL IMPORTS OF RUSSIA. (*a*) (IN METRIC TONS AND DOLLARS.)

(1 poud = .01638 metric ton; 1 ruble = 40 cents.)

Year.	Asbestos.		Asbestos Manufactures		Asphalt Rock.		Baryta and Witherite.		Clay, Bauxite and Talc.		Coal.		Coke.	
1892..	52	$5,938	101	$23,093	1,115	$20,657	3,524	$31,719	38,615	$267,932	1,438,158	$4,144,014	230,109	$742,420
1893..	38	1,060	108	19,924	349	2,600	4,083	33,289	52,593	322,717	1,715,258	4,776,731	289,978	968,914
1894..	49	5,819	197	34,192	458	5,654	6,355	40,797	64,428	418,048	1,973,516	5,361,537	290,822	925,762
1895..	107	8,186	241	58,888	566	3,465	7,703	65,297	55,212	327,838	1,934,289	5,293,392	310,113	961,475
1896..	137	5,510	378	85,432	947	8,183	7,053	56,528	*e*64,135	419,564	1,979,649	5,439,378	364,427	1,146,668

Year.	Copper and Alloys.		Copper and Brass Manufactures.		Glauber Salt.		Gold Bars and Coin—Kg.		Iron.			
									Castings.		Pig.	
1892..	10,485	$3,085,878	1,318	$801,724	1,819	$43,009	134,670	$41,745,312	2,276	$250,216	90,099	$1,500,818
1893..	13,814	4,047,212	*d*1,480	882,232	2,649	88,728	17,019	5,278,049	3,081	326,122	160,507	2,757,083
1894..	8,033	3,352,393	*d*1,558	935,503	3,644	99,052	(*b*)	....	4,669	463,440	144,641	2,299,816
1895..	11,679	2,725,159	*d*1,598	1,045,372	3,303	79,883	(*b*)	....	4,799	602,644	127,202	2,227,129
1896..	15,726	3,997,911	*d*1,601	1,100,247	3,619	78,067	(*b*)	....	7,810	759,814	75,317	1,454,885

Year.	Iron—*Continued.*				Iron and Steel Manufactures.		Lead.		Magnesite.		Mica—Kg.	
	Tin Plate.		All Other.									
1892....	28,233	$2,080,870	40,337	$2,549,708	17,435	$3,804,331	26,075	$1,800,277	935	$11,559	409	$129
1893....	22,132	2,309,751	87,449	4,363,469	(*b*)	2,121,317	18,712	1,481,455	720	17,056	377	348
1894....	20,790	1,491,923	188,418	8,689,817	19,171	3,143,306	11,979	1,620,034	276	5,057	2,162	529
1895....	30,489	2,194,164	218,985	10,182,897	20,214	3,205,413	29,556	1,820,843	1,127	20,094	2,407	917
1896....	23,144	1,707,033	274,288	12,787,963	26,989	3,771,949	31,388	1,720,550	1,434	5,510	3,047	1,301

Year.	Naphtha, Crude.	Oil, Kerosene, Benzine, etc.		Ores, Except Graphite.		Other Metals, Manufactures of.		Phosphorites.		Pyrites.		Quicksilver.	
1892..	$6	317	$28,049	3,140	$104,339	489	$250,165	10,785	$107,790	14,686	$110,498	6	$4,489
1893..	11	362	28,664	1,022	2,261	479	106,241	14,815	156,032	11,594	215,734	10	7,312
1894..	10	465	31,249	(*b*)	....	1,573	982,444	27,006	356,372	36,950	279,076	11	5,993
1895..	(*b*)	537	44,494	1,534	19,800	12,370	3,374,097	(*b*)	....	22,410	241,393	7	6,986
1896..	8	120	7,084	21,468	58,003	4,853	1,987,570	18,156	176,868	20.631	149,582	15	10,499

Year.	Salt.		Saltpeter, Chile.		Salts, Stassfurt.		Silver—Kg.		Steel, All Kinds.		Strontianite and Celestine.		Sulphur.	
1892..	11,000	$68,264	10,982	$394,856	2,770	$21,219	223,161	$3,760,715	16,396	$1,391,168	312	$1,670	16,124	$327,872
1893..	6,914	40,946	8,487	183,537	4,746	37,528	428,107	7,249,674	35,130	2,151,926	404	18,814	23,569	407,818
1894..	10,433	65,692	14,083	357,504	5,190	56,308	(*b*)	....	38,805	3,782,477	575	24,986	19,893	384,752
1895..	10,332	57,400	12,315	294,788	4,463	41,038	(*b*)	....	52,523	3,003,229	34	1,118	19,525	233,233
1896..	10,213	62,492	18,337	414,770	5,553	51,250	(*b*)	....	72,509	3,931,378	216	12,690	19,532	227,428

Year.	Tar, Asphaltic Mastic, etc.		Tin.		Zinc.			
					Ore.		Ingot and Sheet.	
1892..	3,030	$76,390	2,954	$640,276	(*b*)	....	5,683	$543,366
1893..	52,593	322,717	4,007	868,763	(*b*)	....	8,146	909,670
1894..	3,021	74,075	3,386	688,333	(*b*)	....	8,289	879,164
1895..	2,721	77,668	3,417	705,597	(*b*)	....	9,194	962,106
1896..	3,011	89,810	4,481	841,436	(*b*)	....	7,516	766,164

(*a*) From the Russian official report "Sbornik Statisticheskekh Svedenie o Gornozavodskoi Promyshlennostye Rossie v Zavodskom Godu," St. Petersburg. (*b*) Not reported. (*c*) Included under copper and brass manufactures. (*d*) Includes bronze manufactures. (*e*) Represents clay used for furnaces.

MINERAL EXPORTS OF RUSSIA. (*a*) (IN METRIC TONS AND DOLLARS.)

(1 pood = ·01638 metric ton; 1 ruble = 40 cents.)

Year.	Bronze and Manufactures.		Clay, Bauxite and Talc.		Coal.		Copper and Alloys.		Copper and Brass Manufactures.		Gold Coin and Bullion—Kg.	
1891..	(*f*)	$18,469	(*f*)	....	11,188	$29,822	71	$18,003	255	$127,900	796·0	$246,395
1892..	(*f*)	7,042	409	$1,454	8,206	27,035	49	14,848	267	136,939	296·5	92,579
1893..	(*f*)	....	387	(*f*)	7,889	24,174	51	15,912	259	19,983	229·0	68,578
1894..	(*f*)	10,290	166	1,227	7,544	20,870	47	12,154	......	86,146	(*f*)	....
1895..	(*f*)	5,709	1,522	3,542	165,600	17,187	31	8,229	129	62,064	(*f*)	....

Year.	Iron.						Manganese Ore.		Metals, Other		Naphtha.			
	Castings.		Pig.		All Other.						Crude.		Residuum.	
1891..	562	$74,205	247	$6,830	4,783	$223,992	(*f*)	....	175	$13,564	9,563	$76,772	51,872	$409,730
1892..	481	52,258	234	6,340	5,189	252,647	129,016	$1,512,538	(*f*)	....	4,866	12,745	41,759	306,944
1893..	822	135,231	224	6,023	5,497	212,173	125,167	1,251,326	247	36,839	9,574	22,840	53,254	392,590
1894..	298	30,070	192	7,040	2,312	117,207	146,840	1,194,035	202	30,308	9,266	22,856	54,658	403,078
1895..	439	39,011	184	7,950	4,534	239,499	165,600	1,215,399	127	12,150	33,348	220,640	54,451	449,576

Year.	Oil, Kerosene, Benzine, etc.		Ores of Metals and Minerals, Except Graphite.		Paraffin.		Phosphorites.		Platinum, Crude.		Quicksilver.	
1891..	989,206	$9,676,068	63,206	$733,751	*b*1	$88	6,834	$91,269	3·0	$381,900	218	$330,597
1892..	790,007	8,416,610	66	1,189	(*f*)	....	7,629	109,553	6·0	447,685	266	363,317
1893..	819,485	6,778,882	1,183	19,553	(*f*)	....	8,892	28,800	2·0	270,190	132	153,111
1894..	709,406	5,642,097	482	10,260	*b*1	370	10,918	134,886	(*f*)	....	140	164,398
1895..	845,751	8,129,730	20,626	40,071	*b*93	3,746	8,840	107,898	4·7	777,900	430	363,689

Year.	Salt.		Silver—Kg.		Slags.		Steel.		Zinc.			
									Ore.		Ingot and Sheet.	
1891..	7,216	$32,116	125,694	$2,140,998	2,478	$4.273	212	$18,085	134	$1,866	45·0	$4,285
1892..	9,774	34,484	103,087	1.754,856	8,967	12.745	760	59.534	248	3,708	248·0	1,219
1893..	6,457	21,817	167,010	2,858,582	8,994	15,776	283	24,022	40	1.414	27·0	2,963
1894..	7,023	17,790	(*e*)	....	5,173	11,334	167	14,855	208	2,820	1·0	446
1895..	6,814	16,461	(*e*)	....	5,606	11,274	173	15,338	194	4,390	3·5	264

(*a*) From the Russian official report "Sbornik Statisticheskekh Svedenie o Gornozavodskoi Promyshlennosty Rossie v Zavodskom Godu," St. Petersburg. (*b*) Includes vaseline. (*c*) Included under paraffin. (*d*) Includes bronze manufactures. (*e*) Included under precious metals. (*f*) Not reported.

SPAIN.

The official statistics of mineral production, imports and exports, are summarized in the following tables:

MINERAL PRODUCTION OF SPAIN. (*a*) (IN METRIC TONS AND DOLLARS; $1 — 5 pesetas.)

Year.	Aluminous Earths.		Anthracite.		Antimony Ore.		Arsenical Pyrites.		Asphalt Rock.		Clay.		Coal.	
1893....	650	$3,250	(*b*)	......	88	$2,784	160	$324	820	$1,280	60	$30	1,484,794	$2,280,720
1894....	310	1,549	(*b*)	......	15	300	68	136	985	2,009	(*b*)	......	1,659,274	2,827,677
1895....	240	1,198	10	$10	44	700	(*b*)	......	790	1,581	(*b*)	......	1,739,075	2,648,367
1896....	320	1,600	14,895	26,788	54	1,385	(*b*)	......	1,117	2,234	(*b*)	......	1,852,947	2,806,654
1897....	409	2,046	8,758	15,745	354	6,718	(*b*)	......	1,656	3,312	(*b*)	......	2,010,960	3,409,636

Year.	Cobalt Ore.		Fluorspar.		Copper Ore.						Gold and Silver Ore.		Graphite.	
					Argentiferous.		Pyrites.		Copper and Cobalt.					
1893....	18	$194	56	$272	15,219	$27,901	2,144,908	$ 2,151,603	1,116	$26,784	(*b*)	....	*Nil.*	......
1894....	52	624	18	230	(*b*)	....	2,445,242	2,638,819	853	20,472	(*b*)	....	10	$30
1895....	7	84	27	405	(*b*)	....	2,701,661	2,829,422	410	6,560	918	$4,593	(*b*)	......
1896....	18	1,800	3	45	*c*157,365	146,596	2,200,919	2,205,843	992	23,808	854	3,056	(*b*)	......
1897....	13	3,100	2	36	*c*18,488	35,922	2,161,182	2,124,893	(*b*)	....	2,456	9,873	(*b*)	......

Year.	Iron Ore.				Iron Pyrites.		Kaolin. (China Clay.)		Lead Ore.			
	Non-Argentiferous.		Argentiferous.						Non-Argentiferous		Argentiferous.	
1893....	5,419,071	$4,056,546	873	$873	220,000	$110,000	1,502	$5,397	169,707	$3,203,648	179,458	$4,189,689
1894....	5,352,353	3,995,912	736	826	60,000	30,000	247	1,896	140,841	2,208,839	181,715	5,497,816
1895....	5,514,339	4,183,059	572	572	60,267	30,335	836	1,579	124,195	2,052,312	181,433	5,042,288
1896....	6,762,582	5,013,404	3,581	3,581	100,000	50,000	1,240	2,446	104,160	1,884,807	182,565	6,121,173
1897....	7,419,768	5,457,327	5,559	19,347	100,000	50,000	6,204	6,717	110,469	2,058,611	186,692	3,735,018

Year.	Lignite.		Manganese Ore.		Mineral Waters.		Nickel and Cobalt Ore.		Nickel Ore.		Ocher.	
1893....	35,315	$41,967	1,460	$7,666	7,162	$7,636	37.5	$801	31	$917	1,030	$710
1894....	48,460	63,266	340	522	169,107	65,882	(*b*)	....	7·2	144	120	240
1895....	44,708	58,338	10,162	17,043	15,198,865	75,538	(*b*)	....	(*b*)	....	208	787
1896....	55,413	60,261	38,265	53,732	15,738,142	92,485	(*b*)	....	100	830	212	850
1897....	54,232	54,027	100,566	136,250	16,180,585	115,874	(*b*)	....	(*b*)	....	200	800

Year.	Phosphorites.		Quicksilver Ore.		Salt.		Silver Ore.		Steatite.		Sulphate of Barium.	
1893......	211	$415	34,309	$1,618,187	151,464	85,052	4,825	$257,041	4,010	18,705	643	$2,943
1894......	40	80	30,031	1,200,069	186,121	228,374	7,411	257,041	3,108	13,603	541	2,699
1895......	1,040	2,081	33,792	1,248,815	326,320	952,098	16,299	75,836	2,347	6,219	494	2,255
1896......	770	3,080	34,959	1,377,396	521,751	1,153,880	1,230	129,943	756	1,166	345	1,720
1897......	2,084	16,672	32,378	1,328,443	508,606	1,159,294	982	101,797	3,601	21,341	429	2,449

Year.	Sulphate of Soda.		Sulphur, Crude Rock.		Tin Ore.		Topaz of Hinojosa. Kg.		Wolfram Ore		Zinc Ore.	
1893	180	$270	24,793	$59,830	34	$3,611	81	$1,874	19	$975	62,616	$387,101
1894	312	468	10,865	28,248	26	2,678	75	1,549	21	1,070	58,700	305,790
1895	460	690	8,481	22,049	17	1,500	67	1,385	14	710	54,109	370,007
1896	(b)	....	26,204	45,433	d2,348	5,908	80	12,860	31	1,560	64,828	[illegible]
1897	(b)	....	18,845	32,588	d2,378	5,795	44	751	10	309	73,848	362,846

(a) Figures for 1893 and 1897 are from the *Reports of the Comision Ejecutiva de Estadistica Minera*. The figures for 1894, 1895 and 1896 are from the official *Reports of the Junta Superior Facultativa de Minas*, Madrid. There was also produced in 1893: Bournonite, 55 metric tons, $275; refractory sand, 480 tons, $980; silica, 20 tons, $20. In 1896, lead and zinc ores, 400 metric tons, $400; in 1897, gold ore, 450 metric tons, $1,850; lead and zinc ores, 40 metric tons, $40. (b) Not reported. (c) Represents non-argentiferous copper ore. (d) Undressed ore.

METALLURGICAL PRODUCTION OF SPAIN. (*a*) (IN METRIC TONS AND DOLLARS; $1 = 5 pesetas.)

Year.	Arsenic Sulphide.		Asphalt, Refined.		Briquettes.		Cement, Hydraulic.		Coke.	
1893	129	$8,262	580	$7,540	273,118	$1,117,672	140,314	$356,071	116,992	$325,873
1894	184	13,224	905	11,761	209,776	852,231	132,645	376,204	149,905	748,028
1895	184	18,390	790	10,275	342,985	1,370,987	149,197	324,214	249,058	1,046,083
1896	271	27,100	1,285	16,705	343,432	1,373,061	130,738	281,144	288,523	1,202,716
1897	244	29,256	1,878	22,532	332,272	1,318,532	159,439	325,202	755,394	2,167,416

Year.	Copper.						Iron, Pig.		Iron, Forged.	
	Fine.		In Matte.		In Precipitate.					
1893	106	$21,134	18,899	$1,175,956	26,404	$3,696,628	134,563	$1,878,728	58,923	$2,441,290
1894	785	156,947	16,872	1,012,295	29,548	4,136,687	123,798	1,699,682	54,214	2,209,878
1895	7	1,520	5,756	345,385	31,725	4,439,889	179,752	2,575,065	48,462	2,053,822
1896	6	1,200	16,378	982,702	29,873	4,155,204	100,786	1,481,162	53,793	2,224,443
1897	7	1,400	16,120	967,210	29,652	4,138,725	146,940	2,267,403	80,894	3,190,139

Year.	Steel.		Lead.				Quicksilver.		Silver—Kg.	
			Non-Argentiferous.		Argentiferous.					
1893	76,583	$3,048,669	77,456	$3,886,669	91,832	$7,138,834	1,666	$1,394,630	63,605	$1,774,671
1894	92,854	3,584,400	64,189	2,910,453	88,433	7,385,797	1,610	1,286,961	192,745	4,195,184
1895	56,801	2,189,222	76,808	3,637,932	83,978	6,916,320	1,506	1,343,417	58,546	1,300,996
1896	68,126	2,666,843	82,215	4,386,544	84,802	5,808,313	1,524	1,287,144	64,554	1,507,441
1897	66,007	2,873,930	75,112	4,343,187	91,258	6,312,109	1,728	1,564,333	71,168	1,555,686

Year.	Sulphate of Soda.		Sulphur.		Zinc.					
					Slabs.		Sheet.		Calamine, Calcined.	
1893	247	$3,681	4,686	$82,600	3,290	$405,612	2,462	$433,418	21,548	$13,963
1894	174	3,059	3,417	58,086	3,052	360,276	2,421	421,354	17,768	11,514
1895	227	2,946	2,231	37,930	3,149	302,266	2,487	373,125	19,169	(b)
1896	(b)	....	1,800	36,004	3,485	390,286	2,648	405,175	23,108	(b)
1897	(b)	....	3,500	70,000	3,907	445,212	2,337	378,594	(b)	(b)

(a) The figures for 1893, except those for briquettes, copper, coke, iron, steel, and zinc, are from the *Reports of the Comision Ejecutiva de Estadistica Minera;* all others are from the official *Reports of the Junta Superior Facultativa de Minas.* (b) Not reported.

MINERAL IMPORTS OF SPAIN. (*a*) (IN METRIC TONS AND DOLLARS; $1 = 5 pesetas.)

Year.	Asphalt and Pitch.		Alkaline Carbonates		Coal.		Coke.		Gold Bars—Kg.	
1894	29,967	$599,335	28,372	$1,248,390	1,614,839	$8,720,129	225,902	$1,219,873	......	....
1895	33,766	675,316	27,591	1,219,755	1,515,506	8,190,270	255,043	1,377,232	......	....
1896	35,401	566,427	27,014	1,188,641	1,447,345	8,262,168	234,083	1,115,516	337	$235,900
1897	23,324	368,700	28,428	1,210,426	1,633,333	8,837,595	214,763	1,159,731	268	187,600
1898	28,352	526,058	26,099	1,148,371	1,215,554	6,482,757	231,487	1,057,171	2	1,440

Year.	Gold Coin	Iron, Pig.		Iron, Bar.		Iron and Steel, Forged.		Petroleum, Crude.		Silver Bars. Kg.	
1894........	$634,394	25,308	$354,240	9,648	$419,706	23,142	$969,009	44,052	$1,145,862	1,935	$54,181
1895........	119,945	12,385	173,889	7,768	354,289	18,288	753,267	46,592	1,211,897	4,286	120,008
1896........	16,231	8,577	120,087	18,861	607,539	26,468	1,398,827	34,427	1,233,345	5,576	150,812
1897........	12,149	1,855	25,972	13,558	571,043	24,087	1,472,690	34,972	1,259,011	243,540	6,841,894
1898........	(c)	1,575	25,206	(c)	....	2,583	113,640	32,468	1,168,843	152,364	3,961,484

Year.	Silver Coin.	Soda Nitrate.		Sulphur.		Tin Plate.		Tin Ingots.	
1894........	$4,345,859	24,110	$1,446,690	5,668	$1,248,390	3,149	$264.543	1,027	$472,695
1895........	4,776,929	26,385	1,583,129	6,818	1,219,755	1,241	104,223	825	379,625
1896........	20,190,400	26,841	1,610,495	7,744	185,870	1,071	80,977	1,080	432,001
1897........	22,325,312	34,862	2,091,729	5,310	128,883	565	46,098	923	369,359
1898	(c)	38,214	2,217,002	5,664	185,958	653	52,237	834	325,922

(a) The figures for 1894, 1895 and 1898 are from the *Anuario de la Mineria Metalurgia y Electricidad de Espana*, while those for the years 1896 and 1897 are from the *Revista Minera*, Madrid, Feb. 18, 1898. (c) Not reported.

MINERAL EXPORTS OF SPAIN (*a*) (IN METRIC TONS AND DOLLARS; $1 — 5 pesetas.)

Year.	Antimony Ore.		Coal.		Copper Ore.		Iron Ore.		Iron Pyrites.	
1894........	46	$3,728	16,738	$90,341	541,321	$4,871,887	4,975,727	$8,956,309	510,129	$1.020,257
1895........	26	1,566	8,820	44,927	530,075	4,706,835	5,175,260	9,315,468	490,709	981,419
1896........	119	7,190	4,295	23,203	629,546	5,665,918	6,280,257	11,320,670	464,882	920,977
1897........	20	1,176	2,539	13,300	822,570	5,613,482	6,884,244	13,180,064	217,545	569,287
1898........	50	3,002	2,772	14,881	912,238	6,202,821	6,478,488	14,252,674	260,016	624,039

Year.	Lead Ore.				Manganese Ore.		Salt.		Silver—Coin, Jewelry, etc.	
	Argentiferous.		Non-Argentiferous.							
1894........	11,457	$985,345	707	$35,327	7,321	$68,815	228,691	$686,071	22	$715,038
1895........	8,349	534,352	1,979	94,516	30,007	282,209	253,391	760,174	277	8,397,770
1896........	*c*6,270	392,668	(*b*)	....	87,835	827,620	254,752	764,257	(*b*)	....
1897........	*c*8,267	462,595	(*b*)	....	95,756	1,091,621	235,871	707,616	(*b*)	....
1898........	7,327	407,966	(*b*)	....	138,062	1,573,907	219,301	657,703	(*b*)	....

Year.	Steatite.		Sulphur.		Zinc Ore.		Cement, Hydraulic		Copper, in Matte and Precipitate.	
1894........	325	$13,000	244	$6,345	34,119	$337,721	2,409	$14.451	32,061	$3.496,283
1895........	1,415	45,272	83	2,152	29,360	267,957	2,807	16,841	36,210	4,213,024
1896........	(*b*)	....	(*b*)	....	36,656	308,272	(*b*)	....	46,417	4.385,213
1897........	(*b*)	....	(*b*)	....	41,040	353,121	(*b*)	....	51,487	7,373,928
1898........	(*b*)	....	(*b*)	....	4,551	500,577	(*b*)	....	48,834	6,846,854

Year.	Iron and Steel.		Lead.				Quicksilver.		Tin.		Zinc.	
			Argentiferous.		Non-Argentiferous.							
1894........	57,257	$1,590,853	91,511	$5,490,639	68,806	$3,467,557	870	$974,117	16	$7,756	2,680	$267,997
1895........	79,157	3,654,863	87,299	5,587,115	67,187	3,238,438	1,330	1,480,194	10	4,937	1,307	136,690
1896........	23,085	283,270	*d*166,124	9,500,985	(*b*)	....	1,559	1,746,154	(*b*)	....	3,560	356,060
1897........	43,612	610,776	*d*171,774	10,122,220	(*b*)	....	1,742	1,436,293	(*b*)	....	2,170	239,716
1898........	46,129	738,058	*d*179,885	11,015,702	(*b*)	....	1,742	1,881,352	(*b*)	....	4,551	500,577

(*a*) From the *Estadistica Minera de España*, except 1896, 1897 and 1898, which are from the *Anuario de la Mineria Metalurgia y Electricidad de España*, Madrid. (*b*) Not reported. (*c*) Includes non-argentiferous lead ore. (*d*) Includes non-argentiferous.

SWEDEN.

The most important article of mineral production in Sweden is iron ore, from which a metal of high reputation is made. Of other metalliferous deposits the most important are the copper mines of Falun and the silver-lead mines of Sala.

MINERAL PRODUCTION OF SWEDEN. (*a*) (IN METRIC TONS.)

Year.	Alum	Coal.	Cobalt Ore	Copper Ore.	Fire-clay.	Gold Ore.	Iron Ore.	Iron Ore (Bog)	Iron Pyrites	Manganese Ore.	Plumbago.	Red Ocher	Silver and Lead Ore.	Zinc Ore
1893	357	199,933	101	22,033	138,469	2,441	1,483,762	2,275	480	7,061	49	1,371	21,048	[illegible]
1894	261	195,950	(*c*)	25,710	129,617	(*c*)	1,927,212	689	656	3,359	107	1,564	14,825	[illegible]
1895	286	223,652	(*c*)	26,009	120,385	459	1,904,662	2,691	221	3,117	7	1,291	12,045	[illegible]
1896	334	225,848	(*c*)	24,351	120,426	736	2,039,019	925	1,009	2,056	14	(*c*)	15,381	[illegible]
1897	131	224,343	(*c*)	25,207	112,283	1,662	2,086,119	1,047	517	2,749	99	(*c*)	10,068	[illegible]

Year.	Cobalt Oxi. Kg.	Copper.	Copperas.	Copper Sulphate	Gold. Kg.	Iron.			Steel.			Lead. (*b*)	Silver. Kg.	Sulphur.
						Pig.	Bloom.	Bar.	Bessemer.	Martin	Crucible.			
1893	3,298	544	454	659	93·376	453,421	225,532	266,727	81,396	81,880	558	472	4,464	[illegible]
1894	1,580	350	362	723	93·003	462,800	204,517	267,049	83,322	84,003	510	390	2,870	[illegible]
1895	(*c*)	216	94	1,195	85·201	462,920	188,726	294,135	97,320	99,259	598	1,256	1,188	[illegible]
1896	(*c*)	249	191	1,506	114·529	494,418	188,396	298,788	114,120	142,301	604	1,590	2,042	[illegible]
1897	700	289	232	1,315	118·318	538,197	189,633	276,055	107,679	165,836	691	1,480	2,218	[illegible]

(*a*) From *Bidrag till Sveriges Officiela Statistik. Bergshandteringen.* In 1893 there was also an output of [illegible] ton of molybdenum ore; in 1894, 0·03 ton of antimony ore, and in 1895, 1·5 tons of antimony ore. (*b*) In [illegible], antimonial lead, 22 tons, and litharge, 10 tons, are included. (*c*) Not reported.

MINERAL IMPORTS OF SWEDEN. (*a*) (IN METRIC TONS AND DOLLARS; 1 krone = 27 cents.)

Year.	Alabaster (*b*)		Alum.		Aluminum Sulphate.		Ammonia, Caustic.		Ammonium.							
									Carbonate.		Chloride.		Nitrate.		Sulphate.	
1893..	0·348	$470	87·508	$2,306	288	$3,886	42·082	$3,409	49·197	$10,627	64·510	$13,984	0·103	$42	98·909	$7,968
1894..	0·555	749	89·572	2,660	266	3,590	63·515	5,145	64·875	14,018	56·620	12,230	0·158	64	263·239	21,484
1895..	0·567	765	92·916	2,760	348	4,704	75·891	6,147	74·282	16,045	84·171	18,181	10·514	4,258	98·673	3,153
1896..	0·178	58	74·908	2,223	629	8,494	81	7,270	79·234	17,114	87·884	18,963	11·379	4,609	88·058	7,152
1897	3·616	4,882	103·442	3,072	733	9,898	59	5,605	108·552	20,316	110·437	20,878	41·786	9,018	67·201	4,576

Year.	Antimony, Crude.		Arsenious Acid.		Asbestos. (*c*)		Asphalt. (*d*)		Barytes.		Boracic Acid.		Borax.	
1893....	40·996	$7,748	18·081	$1,953	51·784	$13,982	1,967	$21,241	201	$10,856	34·176	$ 6,367	106·496	$16,677
1894....	60·480	11,431	24·108	2,604	68·085	18,383	2,633	28,436	172	9,313	63·692	10,318	107·890	14,561
1895....	80·603	15,234	36·202	3,910	67·697	18,278	3,256	35,161	240	12,939	53·541	8,674	111·099	14,999
1896....	62·552	11,822	33·212	3,587	116·325	31,407	4,092	44,190	298	16,084	72·883	11,807	127·725	17,243
1897....	58·482	11,053	32·622	3,523	119·079	32,151	5,458	58,947	270	14,595	55·591	9,006	175·482	18,952

Year.	Bromine and Bromides of Potassium and Sodium.		Cement.		Chalk.				Chemico-Technical Preparations N.E.S. (*f*)	Chloride of Lime.		Clay.
					White—Unground. *Hektoliters.*		Other Kinds. (*e*)					
1893....	2·987	$2,419	852	$9,434	1,970	$798	124·227	$1,790	$63,204	1,072	$49,212	$48,434
1894....	3·977	3,221	2,425	22,919	2,254	244	100·357	862	64,261	1,102	47,603	68,844
1895....	3·999	3,239	4,270	40,354	4,466	482	108·838	1,579	69,962	1,250	53,982	69,178
1896....	4·334	3,511	2,901	27,411	6,148	664	145·995	1,143	87,835	1,600	69,116	79,655
1897....	5·549	4,495	1,826	17,259	14,368	1,552	142·398	1,069	100,021	1,676	72,388	80,010

Year.	Coal—*Hektoliters.* (*g*)		Coin.		Earths.		Emery.		Glass and Glassware. (*h*)		Glauber Salt. (*i*)		Gold Bars, and Gold M'f'res. Kg.	
			Gold.	Silver.										
1893..	19,396,808	$4,546,423	$617	$102,656	299·938	$8,098	75.100	$10,139	1,451	$376,570	3,774	$101,897	71	$47,350
1894..	24,148,786	8,802,232	530,923	14,803	114·081	3,080	70.688	9,543	1,427	411,182	4,642	125,340	1,633	1 089,048
1895..	23,895,370	8,387,275	11,606	114,340	161·572	4,362	124.623	16,824	1,619	452,276	7,420	200,332	89	59,354
1896..	24,896,994	8,402,736	608	204,601	446·669	12,060	103.731	14,004	2,477	447,352	8,486	229,121	1,161	107,001
1897..	28,003,094	9,829,086	941	136,823	310·645	8,388	128.119	17,296	1,618	507,491	11,384	153,683	4,267	2,086,848

Year.	Gypsum. (*j*)		Iron and Steel.			Lead.		Lime. *Hektoliters.* (*k*)		Litharge.		Nitric Acid.	
			Crude.		Manufactures.								
1893....	3,812	$23,479	27,005	$900,920	$3,252,817	1,493	$134,386	5,587	$2,768	80.681	$10,892	22·174	$2,095
1894....	3,578	27,655	34,201	734,304	2,616,178	1,750	163,197	6,000	2,155	93,412	12,611	26·038	2,109
1895....	4,112	30,347	40,582	878,005	2,549,183	1,624	149,729	4,436	1,628	116.737	15,760	36·208	2,933
1896....	4,940	37,274	34,549	878,383	3,703,338	1,911	180,749	7,768	2,619	149,643	20,202	31·777	2,574
1897....	7,260	75,270	89,606	1,420,175	3,785,043	2,098	206,076	20,050	7,389	199,443	26,925	40·552	3,285

Year.	Phosphorus.		Platinum. Kg.		Plumbago.		Porcelain.		Potash.		Potassium.			
											Chloride.		Cyanide.	
1893....	61·753	$75,030	6	$1,134	118·859	$2,246	218	$166,051	1,858	$175,563	90·970	$12,281	1·101	$892
1894....	56·658	68,839	14	2,646	132·834	7,172	235	175,379	1,744	164,835	300·701	40,595	1·315	1,065
1895....	71·407	86,760	42	7,938	134·154	7,244	277	211,033	1,979	187,019	561·477	113,699	1·457	1,180
1896....	52·482	63,766	34	6,426	135·397	7,311	327	247,878	1,933	182,688	240·784	45,508	2·122	1,719
1897....	57·972	70,436	63	11,907	157·752	8,519	362	273,784	1,432	135,370	363·069	58,817	2·922	2,367

Year.	Quicksilver.		Salt.				Saltpeter. (*l*)		Sand.	Silver and Man'factures Kg.		Soda, Caustic.	
			Common. *Hektoliters.*		Refined.								
1893....	3·069	$3,480	959,375	$446,256	2,144	$ 86,835	5,233	$285,789	$12,945	3,456	$139,968	144	$8,146
1894....	5·896	6,686	953,960	386,354	1,922	77,847	8,436	452,709	14,062	2,535	51,334	245	13,222
1895....	6·318	7,166	1,101,242	446,008	2,535	102,685	9,388	446,851	15,440	3,050	65,880	1,043	56,319
1896....	5·194	5,890	872,459	318,011	3,673	148,755	12,518	565,900	30,428	7,375	217,141	908	49,051
1897....	3·125	3,544	869,619	411,548	3,055	148,479	12,531	602,709	19,509	20,557	473,801	625	33,764

Year.	Stone, etc.	Sulphur.		Sulphuric Acid.		Tin.		Tin and Lead Ashes.		Tin Salts.		Tiles, All Kinds.	Vitriol, All Kinds.	Zinc.	
1893....	$30,700	8,876	$215,690	420	$10,209	355	$155,264	3·034	$3,277	3·944	$1,597	$59,013	$5,263	1,821	$181,517
1894....	36,601	8,142	197,850	418	8,472	413	161,532	3·088	3,335	3·982	1,075	105,233	12,212	1,783	165,582
1895....	35,363	7,140	173,491	772	15,638	462	162,085	6·690	7,235	4·091	1,105	117,188	4,554	2,216	198,289
1896....	122,525	11,369	276,265	615	12,455	551	170,307	11·233	12,132	4·437	1,198	196,482	11,028	2,275	230,865
1897....	227,476	9,723	236,269	1,418	28,709	541	196,478	17·599	28,709	3·823	1,032	263,379	5,805	2,551	261,249

(*a*) From *Bidrag till Sveriges Officiela Statistik.* (*b*) Wrought stone. (*c*) Includes raw product and manufactured articles. (*d*) Natural and artificial asphalts. (*e*) Ground chalk, pastels, etc. (*f*) N. E. S.—Not elsewhere specified. (*g*) Includes coal-dust. (*h*) Exclusive of powdered glass. (*i*) Includes also sodium bi-sulphate. (*j*) Raw, ground and calcined gypsum. (*k*) Slaked and unslaked lime. (*l*) Includes both soda and potash salts.

MINERAL EXPORTS OF SWEDEN. (*a*) (IN METRIC TONS AND DOLLARS; 1 krone = 27 cents.)

Year.	Alum.		Aluminum Sulphate. Kg.		Ammonium Sulphate.		Antimony, Crude.		Asbestos. Kg.		Cement.		Chalk.			
													Unground. *Hektoliters.*		All Other.	
1893....	82	$2,447	(*b*)	...	10	$810	2·3	$427	370	$100	41,399	$391,225	194	78	4,254	$34,461
1894....	77	2,294	10,000	$135	41	3,343	1·2	230	14,032	3,789	29,896	242,161	244	26	3,355	18,188
1895....	38	1,140	(*b*)	....	(*b*)	....	2·3	440	28	7	31,029	251,333	111	12	3,453	18,815
1896....	40	1,184	(*b*)	....	100	8,138	0·8	144	2,040	551	22,991	186,228	164	18	4,382	23,747
1897....	54	1,604	(*b*)	....	180	12,150	0·8	156	1,348	364	27,112	219,611	1,188	128	4,422	24,032

Year.	Chemico-Technical Preparations.	Clay.	Coal.		Copper Ore.		Copper and Alloys, and Metals Not Specified.		Earths Not Specified.		Glass, All Kinds.		Gypsum and Manufactures.	
1893....	$56,717	$12,837	36·9	$156	203·6	$3,848	437·2	$118,279	10,657·0	287,750	7,045	$606,016	11·9	$4,258
1894....	40,627	17,360	33·7	153	(*b*)	....	407·2	123,663	2,661·0	7,186	7,441	1,131,823	27·2	21,308
1895....	39,667	17,360	51·9	182	(*b*)	....	2,283·0	269,066	847·0	2,288	7,150	1,221,310	20·0	12,260
1896....	34,532	28,869	140·8	594	1,094·0	73,856	1,911·0	218,849	1,059·0	2,859	10,674	1,378,125	9·3	7,436
1897....	40,351	28,931	74·0	325	(*b*)	....	932·8	425,089	352·9	953	9,399	1,134,495	9·6	5,727

Year.	Iron Ore.		Iron and Steel.			Lead and Manufactures.		Lime. *Hektoliters.*		Minerals, Not Specified.	
			Unwrought.		Manufactures.						
1893....	484,055	1,306,949	267,052	$7,568,225	$ 956,785	1,005·2	$57,948	24,218	$11,933	52·5	$708
1894....	831,395	1,571,337	252,452	7,009,253	1,342,845	768·2	42,158	34,172	11,444	719·4	9,712
1895....	800,452	1,512,855	311,866	8,653,212	1,794,270	1,379·6	74,712	40,287	13,456	211·6	1,714
1896....	1,150,695	1,905,550	304,138	9,320,969	1,975,068	1,182·2	67,750	102,787	29,453	2,703·9	43,766
1897....	1,400,801	2,779,890	279,525	9,175,910	2,373,230	1,473·0	91,993	106,053	31,925	14·7	119

Year.	Other Ores.		Peat.		Phosphorus. Kg.		Plumbago. Kg.		Potassium Chlorate.		Salt, Refined. Kg.		Sand.
1893....	6,310	$102,218	1,698·6	$4,586	1,225	$1,489	2,043	$38	13·3	$5,014	145,183	$5,880	$514
1894....	5,104	80,741	1,411·1	3,810	1,585	1,926	4,336	234	72·3	19,522	1,781	72	619
1895....	5,419	86,077	1,143·2	3,087	885	1,075	9,749	526	436·4	88,377	1,661	67	511
1896....	2,704	43,766	1,451·8	3,920	1,510	6,795	3,500	189	253·7	47,958	830	34	2,232
1897....	327	5,304	1,815·7	4,902	1,627	1,977	7,215	390	462·5	74,927	1,424	69	969

Year.	Silver.					Soda, Caustic.		Sodium Sulphide.		Stone, Worked.			
	Coin.	Bullion—Kg.		Manufactures—Kg.						Polished.		Other Kinds	Not Specified
1893....	(*b*)	3,981	$161,230	25	$1,350	823	$15,546	9·3	251	38·3	$2,070	$1,094,885	$210,385
1894....	(*b*)	2,579	52,225	6	324	872	16,478	10·3	279	14·5	785	1,220,787	202,717
1895....	$194	751	15,918	10	540	(*b*)	....	12·9	348	8·3	446	1,208,196	211,806
1896....	(*b*)	819	17,359	14	756	(*b*)	....	6·8	184	(*b*)	....	(*b*)	249,516
1897....	14	329	6,396	119	6,426	(*b*)	....	3·4	46	112·5	6,076	1,609,134	446,801

Year.	Sulphur.		Sulphuric Acid.		Tin and Lead Ash.		Tin.				Vitriol.	Zinc.			
							Ingot.		Manufactures—Kg.			Ore.		Manufactures.	
1893....	11·6	$281	2·7	$88	9·9	$10,679	10·6	$4,632	572	$329	$43,201	26,772	$361,425	117	$10,308
1894....	16·1	391	2·6	77	1·8	1,982	18·4	7,214	800	803	44,876	24,822	335,096	117	7,341
1895....	11·2	271	3·1	85	7·6	8,245	13·3	4,683	786	459	90,501	33,075	312,562	26	3,130
1896....	8·8	214	6·6	178	7·6	8,197	18·9	6,367	2,996	1,636	121,524	41,401	458,306	184	11,789
1897....	10·6	258	7·6	205	1·7	1,836	25·6	8,305	7,113	4,328	97,449	44,425	527,769	135	9,009

(*a*) From *Bidrag til Sveriges Officiela Statistik.* (*b*) Not stated in the reports.

UNITED KINGDOM.

The statistics of the mineral production, imports and exports, are given in the subjoined tables. The statistics of the most important substances for 1898 will be found under the respective captions elsewhere in this volume.

MINERAL PRODUCTION OF THE UNITED KINGDOM. (*a*) (METRIC TONS AND DOLLARS; £1=$5.)

Year.	Alum Shale.		Arsenic.		Arsenical Pyrites.		Barytes.		Bauxite.		Bog Ore. (*b*)	
1893......	2,149	$1,320	6,072	$288,470	3,085	$14,740	26,635	$126,815	8,880	$20,750	10,919	$13,430
1894......	4,036	2,480	4,878	243,070	3,341	19,115	22,700	107,050	8,097	28,090	7,928	9,755
1895......	2,096	1,290	4,875	260,990	2,998	13,925	21,509	115,295	10,574	12,530	5,742	7,065
1896......	*Nil.*	....	3,674	227,415	8,949	40,035	24,117	127,950	7,365	9,590	6,758	8,315
1897......	621	380	4,232	373,975	13,347	53,670	23,087	120,585	13,540	14,115	7,238	8,905
1898......	....	....	3,958	....	13,423	....	21,514	....	12,600	....	....	....

Year.	Chalk.		Chert and Flint		Clay. (*d*)		Coal.		Copper Ore.	
1893......	(*c*)	....	(*c*)	....	3,114,251	$4,087,095	166,955,008	$279,049,040	5,432	$64,805
1894......	(*c*)	....	(*c*)	....	3,315,968	4,118,505	191,289,965	318,650,895	5,844	69,545
1895......	2,971,023	$769,820	96,304	$83,305	9,952,823	9,198,035	192,695,944	286,156,065	7,651	109,560
1896......	3,616,177	785,850	109,694	85,150	11,523,250	7,210,345	198,487,040	285,950,735	9,118	107,930
1897......	3,920 183	827,975	95,200	82,475	12,908,479	7,265,640	205,364,010	298,700,045	7,246	93,530
1898......	....	....	....	....	....	....	205,274,919	....	*h*6,848	....

Year.	Copper Precipitate.		Fluorspar.		Gold Ore.		Gravel and Sand.		Gypsum.		Iron Ore.	
1893......	234	$11,050	218	$805	4,561	$38,285	(*c*)	....	145,782	$296,845	11,382,732	$14,139,735
1894......	245	11,565	128	345	6,708	67,865	(*c*)	....	155,905	331,775	12,565,185	15,958,235
1895......	264	14,275	37	270	13,478	82,920	1,030,709	$405,535	180,738	359,175	12,817,261	14,328,545
1896......	201	10,620	400	2,390	2,809	21,285	1,208,310	450,100	196,404	372,690	13,919,976	15,752,120
1897......	224	11,600	302	1,085	4,589	31,410	1,378,496	556,660	184,287	334,890	14,008,484	16,088,975
1898......	....	....	57	......	715	....	....	....	....	....	....	....

Year.	Iron Pyrites.		Jet—Kg.		Lead Ore.		Lignite.		Manganese Ore.		Ocher, Umber, etc.		Oil Shale.	
1893......	16,090	$36,460	403	$885	41,461	$1,402,695	3,316	$1,580	1,357	$3,810	10,702	$69,400	1,987,824	$2,445,650
1894......	15,771	40,210	217	240	41,249	1,334,975	389	415	1,838	3,700	8,652	70,200	2,018,167	2,482,980
1895......	9,193	20,570	76	80	39,026	1,360,960	*Nil.*	......	1,293	3,405	7,747	84,945	2,282,815	2,808,580
1896......	10,177	23,015	133	50	41,726	1,516,990	*Nil.*	......	1,097	3,065	10,049	123,440	2,458,237	3,024,405
1897......	10,752	22,625	31	40	35,903	1,377,045	*Nil.*	......	609	1,755	14,653	64,985	2,259,325	2,779,680
1898......	......	....	......	......	33,202	....	......	......	......	....	......	....	2,172,201	....

Year.	Petroleum.		Phosphate of Lime.		Plumbago.		Quartz.		Salt.		Slag.	
1893......	264	$2,440	3,353	$28,855	*Nil.*	....	(*e*)	....	1,954,813	$3,676,110	(*e*)	....
1894......	50	460	711	6,385	*Nil.*	....	(*e*)	....	2,271,687	3,818,145	(*e*)	....
1895......	15	140	2,540	21,875	41	$500	736	$2,750	2,208,025	3,548,755	137,040	$29,440
1896......	12	145	3,048	26,250	*Nil.*	....	527	1,945	2,054,715	3,333,065	571,290	76,790
1897......	12	145	2,032	17,500	*Nil.*	....	459	1,695	1,933,949	3,104,490	(*e*)	....
1898......	6	....	....	....	....	....	......	....	....	....	....	....

Year.	Slates and Slabs.		Stone.							
			Granite.		Limestone.		Sandstone.		Whinstone, Basalt, etc.	
1893......	446,017	$5,538,130	(c)	....	(c)	....	(c)	....	(c)	....
1894......	469,060	5,856,830	(c)	....	(c)	....	(c)	....	(c)	....
1895......	591,068	6,370,730	1,694,450	$2,739,995	*f* 9,677,440	$6,026,305	4,299,714	$6,832,980	1,756,004	$1,761,910
1896......	596,324	6,691,280	1,784,925	2,490,370	*f* 11,187,532	6,078,020	4,579,869	7,089,925	2,323,591	2,127,935
1897......	618,941	8,247,880	1,876,880	2,763,020	*f* 11,179,580	5,779,965	5,043,535	7,623,500	2,398,243	2,206,955
1898......	....	....	....	....	....	....	....	....	....	....

Year.	Strontium Sulphate.		Tin Ore (Black Tin).		Uranium Ore.		Wolfram Ore.		Zinc Ore.	
1893......	5,905	$11,625	13,908	$3,185,265	25	2,500	22	$2,100	24,134	$406,380
1894......	6,932	9,810	13,117	2,437,615	19	4,075	*Nil.*	....	22,170	336,555
1895......	12,469	17,645	10,782	1,852,650	41	10,355	*Nil.*	....	17,758	247,150
1896......	18,331	25,940	7,786	1,299,640	36	7,500	44	6,775	19,588	332,255
1897......	15,227	21,550	7,234	1,271,090	30	6,835	127	10,040	19,586	345,770
1898......	....	....	8,078	....	26	....	329	....	23,929	....

METALS OBTAINABLE BY SMELTING FROM THE ORES IN THE ABOVE TABLE. (*a*) (IN METRIC TONS AND DOLLARS; £1 = $5.)

Year.	Copper, Fine.		Gold—Kg.		Iron.		Lead. (*g*)		Silver—Kg.		Tin.		Zinc.	
1893	433	$102,610	71·82	$43,455	4,042,353	$46,668,985	30,173	$1,462,010	8,525	$203,435	8,989	$3,928,705	9,432	$838,850
1894	454	74,055	131·72	74,055	4,417,032	49,995,930	30,162	1,423,120	7,575	166,565	8,461	3,022,500	8,260	655,145
1895	589	136,315	205·28	92,600	4,465,307	52,671,625	29,465	1,543,670	8,722	174,540	6,754	2,233,900	6,760	508,475
1896	565	140,900	42·07	25,175	4,835,597	56,877,370	31,311	1,754,700	8,828	181,825	4,915	1,538,390	7,224	616,200
1897	526	135,480	63·20	35,925	4,812,679	56,973,895	26,988	1,662,890	7,750	143,070	4,524	1,456,630	7,162	634,115

(*a*) From *Mineral Statistics of the United Kingdom*. (*b*) Bog ore, which is raised in Ireland, is an ore of iron, used principally for purifying gas. (*c*) Not reported separately, and value included in total value of stone which was as follows: 1893, $38,868,715; 1894, $38,478,580; 1895, $17,361,190; 1896, $17,786,250; 1897, $18,373,440. (*d*) Including china clay, potters' clay, and fuller's earth; and for 1895 and 1896 comprises a large quantity of ordinary brick clay not included in the returns of previous years. (*e*) Not reported. (*f*) Not including chalk. (*g*) Smelted from native ores. (*h*) Includes about 200 tons of copper precipitate.

Additional products: In 1897: Mica, 5,063 metric tons, $8,635; soapstone, 28 metric tons, $490; nickel ore, 305 metric tons, $1,500. Metals obtainable by smelting in 1897: Aluminum, 315 metric tons, $229,400; nickel, 7½ metric tons, $5,250; sodium, 86 metric tons, $63,750.

MINERAL IMPORTS OF THE UNITED KINGDOM. (*a*) (METRIC TONS AND DOLLARS; £1 — $5.)

Year.	Alkali.		Brass and Bronze Manufactures.		Brimstone.		Chemical Products.	China, Porcelain and Earthenware.		Copper Manuf'res Unenumerated.	Copper Ore.	
1894....	7,907	$519,120	1,133	$669,385	23,771	$513,140	$6,877,445	10,246	$3,097,860	$700,905	84,130	$2,468,125
1895....	9,353	572,930	1,637	854,905	25,318	496,835	6,383,350	12,082	3,432,875	910,500	101,353	2,873,015
1896....	8,949	480,650	2,075	1,096,020	22,990	470,060	6,922,975	16,444	4,223,745	1,243,050	88,732	2,833,955
1897....	11,557	545,010	(*b*)	....	22,811	511,045	6,781,900	(*b*)	....	(*b*)	83,916	2,383,130
1898....	12,179	673,180	(*b*)	....	19,642	450,155	6,939,150	(*b*)	....	(*b*)	91,141	3,291,150

Year.	Copper Regulus and Precipitate.		Copper, Wrought, Unwrought, and Old.		Diamonds—*Carats.*		Glass.				
							All Kinds.		Bottles—*Gross.*		Total.
1894...	80,107	$8,839,775	61,260	$12,325,000	2,485,352	$14,989,765	106,468	$11,295,535	821,795	$2,129,775	$13,425,310
1895...	92,727	11,164,775	46,493	9,588,485	3,607,750	23,770,425	101,400	10,565,200	750,856	1,810,680	12,375,880
1896...	92,252	11,477,545	66,405	15,013,905	3,527,810	22,995,830	114,339	10,565,200	853,693	1,972,685	12,537,885
1897...	90,008	11,378,900	62,055	15,200,790	(*b*)	....	122,009	13,022,535	901,188	2,012,020	15,034,555
1898...	76,201	9,779,790	70,018	17,988,325	(*b*)	....	134,933	14,192,300	1,010,615	2,231,615	16,423,915

Year.	Gold Leaf. *Number.*		Gold Ore.		Guano.		Iron, Bar, Angle, Bolt and Rod.		Iron, Girders, Beams & Pillars		Iron Manufactures.	
1894...	84,614,380	$817,855	483	$421,090	29,039	$731,805	64,259	$2,777,790	70,531	$2,141,150	157,416	$13,164,720
1895...	66,138,410	641,105	865	1,001,345	50,639	1,961,545	68,833	2,747,670	70,307	2,181,920	167,813	14,309,855
1896...	70,483,026	694,265	2,263	604,015	20,537	521,770	72,195	2,852,060	76,400	2,336,925	192,388	20,537,665
1897...	(*b*)	....	(*b*)	....	16,423	427,685	69,222	2,697,545	77,125	2,422,520	(*b*)	....
1898...	(*b*)	....	(*b*)	....	24,012	589,620	70,331	2,782,745	105,094	3,229,465	(*b*)	....

Year.	Iron Ore.		Iron, Pig and Puddled.		Iron and Steel Mfres	Steel, Unwrought.		Lead Ore.		Lead, Pig and Sheet.	
1894....	4,484,270	$14,892,985	62,967	$1,498,695	$14,230,955	8,724	$384,050	15,850	$366,625	164,451	$7,578,995
1895....	4,521,516	14,889,760	94,609	2,156,360	15,494,510	11,036	475,015	32,234	797,400	165,531	8,270,865
1896....	5,525,320	18,893,945	108,152	2,206,220	22,055,335	17,771	788,730	57,172	1,448,245	170,484	9,298,715
1897 ..	6,064,179	22,179,670	160,531	2,588,430	27,894,295	40,628	1,330,655	(b)	....	170,121	10,166,500
1898....	5,555,880	20,173,210	162,075	2,688,445	33,379,160	40,875	1,263,025	(b)	....	197,591	12,606,780

Year.	Manganese Ore.		Metals, Unclassified, Wrought and Unwrought.		Nitrate of Soda.		Ores, Unclassified.		Painters' Colors and Pigments.	Paraffin.	
1894....	130,029	$1,692,000	5,606	$2,162,300	127,305	$5,834,000	45,391	$1,822,915	$4,544,005	31,397	$3,166,045
1895....	133,623	1,484,740	4,620	1,945,575	124,650	4,994,485	46,104	1,570,045	4,369,555	36,612	3,778,950
1896....	162,542	1,844,740	5,886	2,294,375	108,148	4,182,760	45,782	1,289,665	4,950,635	30,850	3,555,250
1897....	(b)	....	(b)	....	107,525	4,059,325	(b)	....	(b)	39,284	3,375,055
1898....	(b)	....	(b)	....	132,412	4,864,005	(b)	....	(b)	48,104	4,045,160

Year.	Petroleum. *Liters.*		Phosphate Rock.		Plumbago.		Pyrites, Iron and Copper.		Quicksilver.		Saltpeter.	
1894....	740,519,277	$12,424,880	386,351	$3,616,025	14,361	$1,059,230	625,907	$5,246,555	1,743	$1,516,810	14,700	$1,327,880
1895....	804,777,131	16,844,520	365,414	3,166,570	10,526	887,100	591,782	4,935,825	1,689	1,672,790	11,607	1,068,365
1896....	862,960,772	18,660,280	295,975	2,329,655	13,861	1,121,195	598,480	4,998,965	1,604	1,638,645	17,310	1,392,420
1897....	842,920,307	16,676,355	330,335	2,466,060	(b)	....	633,009	5,104,480	1,862	1,954,405	16,744	1,283,495
1898....	995,397,447	18,668,150	334,884	2,499,360	(b)	....	665,544	5,404,415	1,856	1,940,880	13,323	1,016,215

Year.	Silver Ore.	Stone, Marble, Hewn or Manufactured.		Tin in Blocks, Ingots, Bars, or Slabs.		Tin Ore.		Zinc, Crude, in Cakes.		Zinc Manufactures.		Zinc Ore.	
1894....	$12,199,775	594,844	$3,622,230	39,774	$13,155,190	4,508	$742,760	53,743	$4,099,205	16,843	$1,875,075	46,341	$892,030
1895....	8,732,220	584,082	3,404,885	42,267	13,155,190	4,780	746,105	63,525	4,669,470	19,670	1,875,960	38,006	739,450
1896....	6,483,680	683,977	4,141,135	38,989	11,448,440	4,950	725,915	77,861	6,225,065	21,330	2,097,185	21,421	493,170
1897....	7,149,210	(b)	....	27,214	8,118,990	(b)	....	70,929	5,946,945	21,395	2,194,365	(b)	....
1898....	5,729,525	(b)	....	20,665	6,948,005	(b)	....	78,761	7,211,345	21,613	2,390,440	(b)	....

(a) *Accounts relating to Trade and Navigation of the United Kingdom,* (b) Not reported.

MINERAL EXPORTS OF THE UNITED KINGDOM. (*a*) IN METRIC TONS AND DOLLARS £1 — $5

EXPORTS OF DOMESTIC PRODUCE.

Year.	Alkali.		Brass and Manufactures.		Cement.		Coal, Coke, Cinders, and Fuel.		(b) Coal, etc., for Steam'rs
1894..........	303,896	$8,154,740	5,545	$2,034,155	432,391	$3,516,945	33,602,877	$86,856,655	9,448,172
1895..........	317,439	7,787,095	5,445	2,093,005	401,812	3,207,755	33,631,075	77,169,015	9,558,314
1896..........	245,938	6,210,865	6,070	2,463,115	359,429	2,902,085	34,810,249	75,781,565	10,096,302
1897..........	252,786	6,303,450	5,713	2,458,370	398,023	3,239,595	37,695,772	88,296,470	10,628,050
1898..........	191,578	5,038,025	5,418	2,346,205	331,648	3,053,615	37,130,890	90,073,450	11,444,431

Year.	Coal Products (c)	Copper Sulphate.		Copper, Unwrought, in Ingots.		Copper, Wrought or Manufactured.				Earthen and China Ware and Other Mfres. of Clay.
						Mixed or Yellow Metal.		Other Sorts.		
1894..........	$6,195,810	......	....	19,848	$4,268,775	16,011	$3,410,130	15,653	$4,146,235	$ 9,520,880
1895..........	7,868,655	......	....	30,019	6,724,565	15,150	3,149,530	15,931	4,220,155	10,851,610
1896..........	9,168,630	53,464	$4,303,315	23,698	5,697,555	11,428	2,576,510	15,598	4,445,120	10,847,690
1897..........	8,340,420	60,326	5,113,800	21,252	5,381,155	11,192	2,640,740	15,275	4,629,430	8,693,495
1898..........	7,624,740	35,150	4,224,085	27,102	7,154,225	10,452	2,542,915	13,765	4,298,590	9,037,105

Year.	Glass.					Hardware and Cutlery.	Implements, Tools and Parts.
	Plate—*Sq. Ft.*		All Other.		Total Value.		
1894	1,336,600	$363,170	45,562	$3,213,820	$3,576,990	$9,172,405	$5,972,970
1895	1,446,000	398,575	50,785	3,549,965	3,948,540	9,282,660	6,201,635
1896	1,804,000	512,690	57,222	3,949,755	4,462,445	10,612,020	7,068,345
1897	1,520,511	442,345	57,828	3,917,190	4,359,535	10,536,320	6,821,830
1898	1,815,958	482,040	56,179	3,928,165	4,410,205	9,945,005	6,547,455

Year.	Iron, Pig and Puddled.		Iron, Bar (Except Railroad), Angle, Bolt, and Rod.		Iron, Railroad, of All Sorts.		Iron and Steel Wire and Manufactures, Except Telegraph Wire.		Hoops and Hoop Iron, Sheets (Ungalvanized), Boiler & Armor Plates. (*d*)		Iron, Galvanized Sheets.	
1894	844,281[b]	$9,564,790	131,198	$4,116,825	432,046	$ 9,426,995	35,230	$3,102,680	129,230	$5,166,905	172,253	$ 9,743,260
1895	880,433	10,385,365	146,294	4,270,085	464,873	9,485,180	42,896	3,555,940	104,574	3,815,570	207,472	11,256,870
1896	1,077,128	12,669,415	180,973	5,471,445	759,625	17,802,050	57,008	4,519,975	122,660	4,226,140	248,348	14,218,110
1897	1,219,958	14,461,865	170,285	5,421,865	795,983	19,293,670	52,471	4,332,965	120,868	4,483,370	231,319	12,810,620
1898	1,058,973	13,684,020	152,911	4,978,125	619,976	15,081,290	44,954	3,864,220	102,638	3,916,335	230,219	12,653,330

Year.	Black Plates for Tinning (Iron and Steel)		Tinned Plates.		Iron, Cast and Wrought, and Manufactures.		Old Iron.		Steel, Unwrought. (*f*)		Manufactures of Steel and Iron.	
1894	(*e*)	....	359,591	$21,693,930	270,137	$17,159,950	84,588	$1,106,580	214,879	$9,870,905	18,966	$2,490,365
1895	34,918	$1,691,730	371,978	21,195,965	293,486	18,638,035	98,654	1,262,700	211,616	9,743,975	23,718	3,163,290
1896	49,179	2,389,995	271,234	15,180,075	372,090	23,598,635	129,463	1,694,535	302,198	12,587,775	37,299	4,650,350
1897	59,663	2,844,645	276,260	15,186,395	382,179	24,426,375	99,259	1,195,925	304,249	13,033,895	47,626	5,706,825
1898	59,289	2,706,435	255,797	13,768,540	361,927	23,630,065	86,602	1,201,050	290,182	13,203,560	35,857	4,513,470

Year.	Total of Iron and Steel.		Lead, Pig, Sheet, Pipe, and Other M'f'res.		Machinery	Salt, Rock and White.		Telegraph Wire and Apparatus.	Tin, Unwrought.		Zinc or Spelter, Wrought and Unwrought	
1894	2,692,398	$93,443,815	47,813	$2,589,055	$71,026,075	781,303	$3,020,525	$6,983,070	5,943	$2,160,085	9,300	$622,525
1895	2,880,910	98,404,615	42,372	2,465,980	75,752,610	752,397	2,731,095	3,911,875	5,761	1,918,020	10,036	651,925
1896	3,607,204	119,008,500	41,873	2,616,835	85,071,250	671,510	2,351,650	4,285,870	6,299	1,987,390	9,640	672,645
1897	3,750,122	123,198,215	41,151	2,710,440	81,410,425	680,477	2,332,620	5,005,510	5,050	1,616,000	8,013	619,165
1898	3,299,326	113,200,460	38,639	2,725,170	91,900,380	698,882	2,300,075	5,089,040	5,557	1,974,430	8,846	790,645

EXPORTS OF FOREIGN AND COLONIAL PRODUCE. (*a*) IN METRIC TONS AND DOLLARS· £1 = $5.

Year.	Chemical Products.	Copper, Unwrought, Partly Wrought, and Old.		Glass.					Guano.		Iron, Bar, Angle, Bolt, and Rod.	
				Bottles, *Gross.*		All Other Kinds		Total.				
1894....	$1,081,810	6,594	$1,394,975	10,952	$26,020	4,179	$347,475	$373,495	1,549	$53,730	18,664	$700,860
1895 ...	1,050,315	8,093	1,788,920	10,813	24,935	4,334	380,280	405,215	2,001	63,085	23,184	874,945
1896....	1,093,585	9,809	2,338,295	12,335	24,895	3,324	292,575	317,470	1,727	69,465	16,882	638,705
1897....	1,295,400	10,207	2,526,150	11,666	26,670	4,320	376,305	402,975	2,415	86,370	21,842	962,905
1898....	1,509,195	13,454	3,578,710	12,694	29,875	3,961	353,905	383,780	3,125	94,510	26,891	1,135,450

Year.	Steel, Unwrought.		Iron and Steel, N. E. S., Wrought and Manufactured. (*h*)		Petroleum. *Liters.*		Quicksilver.		Saltpeter.		Tin, in Blocks, Ingots, Bars, or Slabs.	
1894....	3,792	$146,495	34,574	$2,086,325	8,650,635	$288,035	1,483	$1,296,975	2,716	$237,195	21,886	$7,615,225
1895....	5,291	234,030	34,790	2,499,670	8,001,341	268,690	1,214	1,241,465	1,459	130,020	21,042	6,593,555
1896. ..	4,000	167,345	37,292	3,712,450	11,709,283	354,420	1,198	1,150,020	1,913	152,765	19,007	5,662,945
1897....	4,244	194,980	29,305	4,324,980	6,753,750	270,735	1,047	1,050,330	2,949	218,610	14,994	4,545,810
1898....	2,619	154,480	29,509	5,658,860	6,479,451	217,200	1,157	1,199,500	2,775	203,030	15,885	5,310,330

(*a*) From *Accounts Relating to Trade and Navigation of the United Kingdom.* (*b*) Number of tons shipped for the use of steamers engaged in the foreign trade. This not being an export in the ordinary acceptation of the term, the value thereof is not given in the trade returns. (*c*) Including naphtha, paraffin, paraffin oil and petroleum. (*d*) Prior to 1895 iron black plates for tinning also were included under this heading. (*e*) Iron black plates for tinning included these years under "Hoops, Hoop Iron, Sheets, Boiler and Armor Plates;" steel black plates for tinning included under "Steel Unwrought." (*f*) Prior to 1895 includes steel black plates for tinning. (*g*) Included under "Glass—all other kinds." (*h*) N. E. S. signifies "not elsewhere ed."

UNITED STATES.

Statistics and full particulars of the mineral production of the United States will be found in the introduction and the articles on the different substances. We give below the mineral imports and exports for five years:

MINERAL IMPORTS OF THE UNITED STATES. (*a*)

Year.	Aluminum.							Antimony.			
	Crude.				Leaf.		Mfd.				
	Lb.	Kg.	Value.	Value per Kgm.	Pkgs.	Value.	Value.	Lb.	Metric Tons.	Value.	Value per Met. Ton.
1894..	5,303	2,406	$2,524	$1·04	10,780	$1,210	$386	2,653,487	1,204	$193,968	$161·12
1895..	25,294	11,475	7,814	·68	6,610	646	1,841	3,399,903	1,587	223,968	141·12
1896..	698	316	591	1·87	4,657	523	2,305	2,402,502	1,090	148,700	136·42
1897..	1,822	834	1,082	1·26	4,260	870	3,279	1,146,696	520	46,955	90·30
1898..	60	27	30	1·11	2,000	174	4,675	2,025,133	919	148,909	156·59

Year.	Antimony Ore.				Asbestos.			Asphaltum.			
	Lb.	Metric Tons.	Value.	Value per Met. Ton.	Crude. Value.	Manu-fact'ed. Value.	Total. Value.	Long Tons.	Metric Tons.	Value.	Value per Met. Ton.
1894..	375,468	170	$18,068	$106·28	$240,029	$15,989	$256,018	106,014	107,710	$313,685	$2·91
1895..	668,610	303	14,718	48·57	225,147	19,731	244,878	79,194	80,461	212,407	2·63
1896..	1,180,828	535	21,392	39·98	229,884	15,654	244,738	95,233	96,757	286,099	2·95
1897..	5,502,132	2,496	167,373	67·06	264,220	10,570	274,790	127,325	129,362	427,390	3·30
1898..	3,725,222	1,690	50,256	29·74	287,636	12,899	300,535	77,711	78,954	202,452	2·56

Year.	Barium Sulphate.								Bauxite.			
	Manufactured.				Unmanufactured.							
	Long Tons.	Metric Tons.	Value.	Value per Met. Ton.	Long Tons.	Metric Tons.	Value.	Value per Met. Ton.	Lb.	Metric Tons.	Value.	Value per Met. Ton.
1894...	836	849	$10,556	$12·43	1,884	1,914	$5,270	$2·75	2,305,783	1,046	$6,661	$6·36
1895...	1,629	1,655	17,112	10·34	2,551	2,592	7,561	2·92	12,985,525	5,890	29,194	4·95
1896...	2,467	2,506	23,345	9·32	509	517	1,274	2·46	4,765,974	2,161	10,477	4·84
1897...	1,300	1,321	13,822	10·46	502	510	579	1·14	5,924,639	2,695	10,515	3·90
1898...	687	698	8,678	12·43	1,022	1,038	2,078	2·58	2,690,240	1,220	4,238	3·47

Year.	Brass and Manufactures of. Value.	Chloride of Lime, or Bleach.				Cement. (b)			
		Lb.	Metric Tons.	Value.	Value per Met. Ton.	Barrels.	Metric Tons.	Value.	Value per Met. Ton.
1894	$134,356	96,256,251	43,673	$1,697,038	$38·85	2,638,107	478,785	$3,396,729	$7·09
1895	158,822	103,317,968	46,855	1,628,877	34·76	2,975,082	543,844	3,873,123	7·12
1896	157,798	99,491,335	45,129	1,441,600	31·94	2,668,624	484,192	3,394,426	7·01
1897	85,341	109,176,451	49,522	1,460,799	29·50	2,090,824	379,357	2,688,122	7·09
1898	24,611	108,462,828	49,198	1,229,978	25·00	2,015,605	365,709	2,625,950	7·18

Year.	Chrome Ore.				Chromic Acid.				Clays or Earths, Including Kaolin.			
	Long Tons.	Metric Tons.	Value.	Value per Met. Ton.	Lb.	Kg.	Value.	Value per Kg.	Long Tons.	Metric Tons.	Value.	Value per Met. Ton.
1894	3,470	3,525	$38,364	$10·88	7,680	3,030	$1,045	$0·34	78,698	79,983	$616,506	$7·79
1895	5,230	5,314	82,845	15·59	2,083	945	414	·43	98,862	100,450	716,590	7·13
1896	8,669	8,808	187,400	21·27	2,429	1,002	387	·39	98,243	99,815	707,826	7·09
1897	11,566	11,751	186,313	15·85	71,220	32,305	5,477	·17	102,391	104,029	696,935	6·70
1898	16,304	16,565	272,234	16·43	5,329	2,417	1,758	·73	113,277	115,089	779,491	6·77

Year.	Coal.										
	Anthracite.				Bituminous.				Total.		
	Long Tons.	Metric Tons.	Value.	Value Per Met. Ton.	Long Tons.	Metric Tons.	Value.	Value Per Met. Ton.	Metric Tons.	Value.	Value Per Met. Ton.
1894	90,068	91,509	$233,773	$2·55	1,244,330	1,264,239	$3,829,807	$3·02	1,355,748	$4,063,580	$3·00
1895	141,337	143,598	328,706	2·29	1,229,109	1,248,775	3,642,055	2·92	1,392,373	3,970,761	2·85
1896	101,689	103,316	237,717	2·30	1,246,991	1,266,943	3,488,115	2·75	1,370,259	3,725,832	2·72
1897	3,281	3,333	8,720	2·62	1,276,963	1,297,394	3,423,434	2·64	1,300,728	3,432,154	2·64
1898	3,149	3,199	8,609	2·69	1,269,775	1,290,091	3,562,991	2·76	1,293,291	3,571,600	2·76

Year.	Coke.				Cobalt Oxide.				Copper, Ore and Regulus.			
	Long Tons.	Metric Tons.	Value.	Value per Met. Ton.	Lb.	Kg.	Value.	Value per Kg.	Lb.	Metric Tons.	Value.	Value per Met. Ton.
1894	27,591	28,032	$66,463	$2·37	24,020	10,900	$29,857	$2·73	4,898,880	2,222	$71,283	$32·08
1895	26,448	26,871	71,366	2·66	36,155	16,400	39,839	2·42	21,580,160	9,789	404,439	41·32
1896	43,372	44,066	114,713	2·60	27,189	12,333	36,212	2·93	6,117,440	2,775	348,629	125·63
1897	35,193	35,756	98,558	2·76	24,771	11,236	34,773	3·09	11,587,520	5,256	816,590	155·36
1898	41,185	41,844	142,334	3·40	33,731	15,300	49,245	3·22	6,921,600	3,140	666,079	212·13

Year.	Copper, Ingots, Old, etc.				Copper, Manufactures. Value.	Copper. Total Value.	Cryolite.			
	Lb.	Metric Tons.	Value.	Value per Met. Ton.			Long Tons.	Metric Tons.	Value.	Value per Met. Ton.
1894	3,446,724	1,563	$251,737	$161·06	$66,256	$389,276	10,684	10,855	$142,494	$13.12
1895	9,381,800	4,255	839,603	197·32	60,344	1,304,386	9,500	9,652	127,165	13.17
1896	11,397,273	5,170	961,920	186·05	85,123	1,395,672	3,009	3,057	40,056	13.10
1897	16,578,420	7,520	1,454,016	193·35	58,897	2,329,503	10,115	10,277	135,114	13.24
1898	54,166,467	24,570	4,120,680	167.71	39,467	4,826,226	6,201	6,300	88,501	14·05

Year.	Earthen, Stone, and China Ware Value.	Emery Grains.				Emery Rock.			
		Lb.	Metric Tons.	Value.	Value per Met. Ton	Long Tons.	Metric Tons.	Value.	Value per Met. Ton.
1894	$6,978,572	597,713	271	$18,645	$68·80	1,642	1,668	$51,487	$30·86
1895	10,523,925	678,761	308	25,066	81·38	6,803	6,912	80,386	11·62
1896	9,690,142	751,464	341	26,520	77·77	6,289	6,390	119,667	18·72
1897	8,409,678	520,095	238	20,022	84·84	5,209	5,292	107,649	20·35
1898	7,271,511	577,655	262	23,320	89·01	5,547	5,636	106,269	18·86

Year.	Emery Manufactures. Value.	Emery. Total Value.	Gold and Silver in Coin and Bullion.		Gold and Silver in Ores.	
			Gold.	Silver.	Gold.	Silver.
1894	$1,830	$71,962	$20,607,561	$9,824,408	$743,046	$7,809,186
1895	27,586	133,038	32,538,736	11,286,007	1,857,656	13,087,840
1896	1,971	148,158	102,761,282	12,504,577	1,969,977	17,775,163
1897	2,211	129,882	29,081,760	12,146,750	4,941,052	20,935,552
1898	3,810	133,399	153,145,259	9,571,810	5,006,593	19,554,446

Year.	Guano.				Iron Ore.				Pig Iron.			
	Long Tons.	Metric Tons	Value.	Value per Met. Ton.	Long Tons.	Metric Tons.	Value.	Value per Met. Ton.	Long Tons.	Metric Tons.	Value.	Value per Met. Ton.
1894	5,646	5,736	$105,879	$18·45	168,541	171,238	$267,241	$1·56	15,582	15,831	$407,638	$25·74
1895	4,066	4,131	48,917	11·84	524,153	532,571	786,207	1·47	53,232	54,084	1,337,978	24·73
1896	6,838	6,947	88,231	12·70	682,806	692,999	1,036,917	1·49	56,272	57,172	1,207,890	22·87
1897	4,931	5,019	55 709	11·12	489,970	497,809	678,912	1·39	19,212	19,519	484,655	24·83
1898	5,259	5,343	56,983	10·66	187,219	190,214	255,802	1·34	25,137	25,539	703,829	27·56

Year.	Scrap Iron and Steel.				Bar Iron, Rolled and Hammered.				Bars, Railway, of Iron or Steel.			
	Long Tons.	Metric Tons.	Value.	Value per Met. Ton.	Lb.	Metric Tons.	Value.	Value per Met. Ton.	Long Tons.	Metric Tons.	Value.	Value per Met. Ton.
1894	2,380	2,418	$43,710	$18·07	20,669,825	9,576	$377,397	$39·41	300	304	$4,292	$14·11
1895	6,066	6,163	103,523	16·79	44,910,023	20,381	772,822	37·91	1,447	1,470	27,076	18·41
1896	8,250	8,382	130,012	15·51	36,458,326	16,538	686,733	41·52	7,796	7,921	207,648	26·21
1897	1,549	1,573	12,433	7·26	31,349,707	14,220	614,318	43·20	415	422	15,939	37·77
1898	1,783	1,812	33,330	18·39	42,826,982	19,429	844,351	43·46	200	203	5,181	25·52

Year.	Hoop, Band, or Scroll.				Ingots, Blooms, Slabs, Billets, etc.				Sheet, Plate, and Taggers Iron or Steel.			
	Lb.	Met. Tons.	Value.	Value per Met. Ton.	Lb.	Met. Tons.	Value.	Value per Met. Ton.	Lb.	Metric Tons.	Value.	Value per Met. Ton.
1894	1,803,104	818	$51,111	$62·48	21,266,348	9,646	$809,184	$83·89	64,500,887	29,258	$1,197,671	$40·93
1895	33,168	15	2,925	195·00	58,811,651	26,677	1,610,889	60·38	32,549,127	14,764	727,638	49·28
1896	59,105	27	5,664	209·78	45,598,270	20,683	1,657,805	80·15	15,201,124	6,896	328,421	47·62
1897	61,916	28	2,873	102·61	38,600,934	17,509	1,523,410	87·00	5,854,990	2,656	170,385	64·15
1898	7,527	3	224	74·66	23,868,684	10,827	1,008,360	93·13	5,085,287	2,307	181,021	78·47

Year.	Ties for Baling Cotton.				Tin Plates, Terne Plates, and Taggers Tin.			
	Lb.	Metric Tons.	Value.	Value per Metric Ton.	Lb.	Metric Tons.	Value.	Value per Metric Ton.
1894	123,413	46	$3,392	$73·74	481,751,601	218,521	$12,053,167	$55·15
1895	7,806,304	3,541	102,309	28·89	491,880,513	223,100	11,482,380	51·46
1896	16,031,468	7,272	235,830	32·43	266,943,267	120,803	6,140,161	50·82
1897	235,154	107	3,686	34·45	187,825,880	85,198	4,366,828	51·25
1898	....	....	....	....	149,576,525	67,847	3,311,658	48·81

Year.	Wire Rods.				Wire, and Articles made from.				Manufactures. Value.	Total Value of Iron Imports. (e)
	Lb.	Metric Tons.	Value.	Value per Met. Ton.	Lb.	Metric Tons.	Value.	Value per Met. Ton.		
1894	50,640,787	22,971	$923,675	$40·21	9,604,368	4,357	$514,484	$118·08	$4,457,855	$20,843,576
1895	60,107,521	27,264	1,090,707	40·00	12,931,695	5,866	716,901	122·21	7,796,988	25,772,136
1896	42,509,008	19,281	811,224	42·07	8,230,248	3,733	492,961	132·05	7,602,238	19,506,587
1897	36,768,538	16,679	772,950	46·34	5,720,322	2,596	344,855	132·84	5,523,678	13,886,010
1898	35,307,369	16,015	767,909	47·95	4,517,761	2,049	318,553	155·46	5,299,221	12,473,887

Year.	Lead, Pig, Bars, Scrap, and in Ore.				Lead, Sheet, Pipe, and Shot.	
	Lb.	Metric Tons.	Value.	Value per Metric Ton.	Lb.	Metric Tons.
1894	139,481,911	63,269	$4,228,796	$66·84	44,080	20
1895	201,617,855	91,453	2,910,547	31·82	128,008	58
1896	160,318,517	72,720	1,780,265	24·48	216,820	98
1897	185,318,412	84,060	2,480,471	29 51	95,891	43
1898	179,291,290	81,326	2,535,362	31·18	242,759	110

Year.	Lead, Sheet, Pipe, and Shot—*Continued.*		Lead, Other Manufactures. Value.	Lead, Total Value.	Manganese Ore.			
	Value.	Value per Met. Ton.			Long Tons.	Metric Tons.	Value.	Value per Met. Ton.
1894	$2,050	$102·50	$1,552	$4,232,398	44,655	45,384	$432,561	$9·53
1895	5,030	86·72	12,961	2,928,538	86,111	87,489	747,910	8·54
1896	8,513	86·87	5,587	1,794,365	31,489	31,993	250,468	7·82
1897	4,042	94·00	4,856	2,489,369	39,574	40,207	340,945	8·48
1898	9,389	85·35	8,329	2,553,080	114,885	116,723	831,967	7·13

Year.	Marble and Stone, and M'f'res of.			Metals, Metal Compositions, and Manufactures of. N.E.S.			Mica, Mineral Substances, and Nickel.		
	Marble and Manufactures of.	Stone and Manufactures of.	Total.	Bronze Manufactures.	All Other.	Total.	Mica.	Mineral Substances N. E. S.	Nickel.
	Value.	Value. (*d*)	Value.	Value.	Value.	Value.	Value.	Value.	Value.
1894	$787,657	$864,733	$1,652,390	$473,672	$3,293,477	$3,767,149	$126,184	$139,688	$45,927
1895	865,243	411,122	1,276,365	521,505	4,180,892	4,702,397	174,886	111,447	56,890
1896	808,030	387,979	1,196,009	468,884	3,692,937	4,161,821	169,086	74,395	32,532
1897	860,659	263,381	1,124,040	526,741	3,543,827	4,070,568	159,616	73,373	53,013
1898	726,288	229,908	956,196	508,814	3,479,643	3,988,457	150,082	19,160	(*f*)

Year.	Oil, Mineral.				Paints, Mineral. Zinc Oxide in Oil.				Paints and Colors.	Platinum, Man'f'ur'd.
	Gallons.	Liters.	Value.	Value per Liter.	Lb.	Metric Tons.	Value.	Value pr. Met. Ton.	Value.	Value.
1894	152,870	578,659	$27,667	$0·047	59,291	27	....	....	$1,045,251	....
1895	1,917,823	7,259,804	54,963	·007	129,343	59	$7,272	$123·25	1,363,657	$171
1896	68,803	260,419	14,062	·054	311,023	141	14,793	104·90	1,189,781	....
1897	949,178	3,592,638	77,789	·021	502,357	228	19,608	86·00	1,319,894	121
1898	2,024,090	7,661,808	146,713	·019	27,050	12	2,279	189·92	1,140,967	271

Year.	Platinum, Unmanufactured.			Plat'm Vases, Retorts	Potash, Chromate & Bichromate.				Potash, Muriate of.			
	Kg.	Value.	Val. per Kg.	Value.	Lb.	Met. Tons.	Value.	Value per Met. Ton.	Lb.	Metric Tons.	Value.	Val. per Metric Ton.
1894	1,712·78	$485,415	$283·41	$88,195	1,483,762	673	$125,796	$186·91	101,597,074	46,084	$1,540,081	$33·42
1895	2,320·15	690,584	297·65	27,354	2,045,910	928	181,242	195·30	81,833,531	37,119	1,296,184	34·92
1896	2,584·14	926,678	358·60	106,286	952,794	432	80,538	186·43	88,525,983	40,155	1,392,504	34·68
1897	2,584·14	960,299	371·61	43,800	1,329,473	603	108,497	179·93	108,839,049	49,369	1,683,472	34·10
1898	3,142·07	1,178,142	374·96	52,012	1,160,710	526	86,134	163·75	104,358,601	47,337	1,620,720	34·45

Year.	Potash, Nitrate of.				Precious Stones.		Pyrites.				
					Rough & Uncut.	Cut.					
	Lb.	Metric Tons.	Value.	Value per Met. Ton.	Value.	Value.	Long Tons.	Metric Tons.	Value.	Value per Met. Ton.	Average Sulphur Contents.
1894..	9,375,950	4,254	$249,842	$58·73	$339,806	$6,710,472	163,546	166,163	$590,905	$3·55	25%
1895..	11,419,090	5,180	305,207	58·91	111,033	6,623,669	190,436	193,483	673,812	3·48	....
1896..	20,085,327	9,111	479,899	52·67	78,815	4,854,358	199,678	202,873	1,140,571	5·62	....
1897..	16,276,352	7,383	306,696	41·54	1,425,614	5,090,125	259,546	263,699	874,419	3·32	25%
1898..	11,461,823	5,199	281,504	54·15	2,550,076	7,388,086	171,870	174,620	544,165	3·12	....

Year.	Quicksilver.	Salt.				Soda, Nitrate of.			
	Value.	Lb.	Metric Tons.	Value.	Value per Met. Ton.	Long Tons.	Metric Tons.	Value.	Value Per Met. Ton.
1894............	$6	420,226,039	191,118	$643,167	$3·36	98,026	99,594	$3,185,356	$31·98
1895............	2,503	566,869,046	257,130	760,811	2·95	118,477	120,378	3,778,360	31·38
1896............	2,037	527,635,090	239,334	696,197	2·91	115,504	117,352	3,566,744	30·39
1897............	20,147	461,813,843	209,479	611,166	2·44	94,965	96,484	2,810,187	29·12
1898............	51	374,308,045	169,785	587,825	3·46	147,494	149,854	2,298,240	15·34

Year.	Soda, Bicarbonate of.				Soda, Caustic.			
	Lb.	Metric Tons.	Value.	Value per Metric Ton.	Lb.	Metric Tons.	Value.	Value per Metric Ton.
1894..........	4,479,524	2,032	$63,625	$31·31	44,772,512	20,309	$911,942	$44·90
1895..........	8,862,028	4,020	123,425	30·70	72,019,114	32,668	1,211,090	37·18
1896..........	4,343,786	1,970	60,878	30·90	47,827,247	21,694	859,809	39·63
1897..........	965,669	438	13,982	31·92	57,742,392	26,192	988,318	37·73
1898..........	280,988	127	5,794	45·62	24,981,873	11,332	354,270	31·26

Year.	Soda Ash and Sal Soda.				Other Soda Salts.			
	Lb.	Metric Tons.	Value.	Value per Metric Ton.	Lb.	Metric Tons.	Value.	Value per Metric Ton.
1894..........	321,300,874	144,971	$2,665,836	$18·39	13,238,190	6,005	$68,760	$11·45
1895..........	307,026,094	139,266	2,321,612	16·67	1,081,887	491	31,581	64·32
1896..........	191,780,537	86,991	1,444,417	16·60	1,279,285	580	40,039	69·03
1897..........	154,656,060	70,152	1,122,867	16·00	7,927,145	3,596	100,989	28·08
1898..........	73,064,707	33,142	447,119	13·49	23,354,295	10,593	256,958	24·26

Year.	Sulphur, Crude.				Sulphur, Flowers.				Sulphur, Refined.			
	Long Tons.	Metric Tons.	Value.	Value per Met. Ton.	Long Tons.	Metric Tons.	Value.	Value per Met. Ton.	Long Tons.	Metric Tons.	Value.	Value per Met. Ton.
1894	124,467	126,500	$1,734,648	$13·71	165	168	$4,145	$24·67	41	42	$1,207	$28·74
1895	115,959	127,974	1,593,148	12·45	581	590	12,888	21·84	229	233	4,378	18·79
1896	145,318	147,648	2,085,070	14·12	665	675	13,266	19·65	447	454	8,226	18·11
1897	138,846	141,067	2,442,420	17·31	819	824	7,050	24·54	148	150	3,387	22·58
1898	159,790	162,347	3,081,974	18·98	507	515	14,548	28·25	163	166	4,391	26·45

Year.	Talc.				Tin.			
	Lb.	Metric Tons.	Value.	Value per Metric Ton.	Lb.	Metric Tons.	Value.	Value per Met. Ton.
1894..........	1,242,171	563	$6,835	$12·14	39,268,628	17,812	$5,944,065	$333·71
1895..........	6,329,268	2,871	25,843	9·00	54,252,045	24,609	7,405,619	300·93
1896..........	3,901,314	1,770	18,693	10·56	44,639,324	20,248	5,848,933	288·86
1897..........	1,507,849	725	8,423	11·61	55,172,571	25,026	7,415,933	296·33
1898..........	889,645	404	5,526	13·68	62,748,399	28,462	8,770,221	308·14

Year.	Zinc.					
	Block, Pig, and Old.				Manufactures of. Net Value.	Total Value.
	Lb.	Metric Tons.	Value.	Value per Met. Ton.		
1894..........	512,932	233	$17,271	$74·12	$12,342	$29,613
1895..........	864,113	392	29,352	74·87	12,183	41,535
1896..........	856,044	388	25,904	66·76	15,728	41,632
1897..........	2,557,341	1,160	95,883	82·66	19,431	115,314
1898..........	2,741,361	1,243	109,626	88·19	13,447	123,073

(a) From *Summary of Commerce and Finance of the United States.* (b) Custom-house returns for these years are given in pounds, which are reduced to barrels of 400 lb. for convenience of comparison. (d) Including slate. (e) Not including ore. (f) Not reported.

MINERAL EXPORTS OF DOMESTIC PRODUCTION OF THE UNITED STATES. (*a*)

Year.	Asbestos.	Brass & Manufactures of.	Cement.				Chemicals, Drugs, & Medicines.	Coal.			
								Anthracite.			
	Value.	Value.	Bbls.	Metric Tons(*m*)	Value.	Value per Met. Ton.	Value.	Long Tons.	Metric Tons.	Value.	Val. per Met. Ton.
1894	$5,762	$779,875	(*b*) 108,000	19,596	$165,809	$8·46	$7,722,532	1,440,625	1,463,675	$6,359,021	$4·34
1895	(*n*)	730,424	(*b*) 83,632	15,174	117,646	7·75	8,749,090	1,470,710	1,494,241	5,937,130	3·96
1896	(*n*)	1,026,191	54,839	9,950	86,757	8·72	9,261,354	1,350,000	1,371,600	5,925,506	4·32
1897	73,736	1,346,802	53,466	9,701	93,684	9·66	9,633,381	1,298,768	1,319,548	5,836,730	4·42
1898	54,660	1,237,017	36,732	6,665	73,838	11·08	9,732,734	1,350,948	1,372,563	5,712,985	4·16

Year.	Coal—*Continued.*							Copper Ore. (*c*)			
	Bituminous.				Total Long Tons.	Total Metric Tons.	Total Value.				
	Long Tons.	Metric Tons.	Value.	Value per Met. Ton.				Lb.	Metric Tons.	Value.	Val. per Metric Ton.
1894	2,195,716	2,230,847	$4,970,270	$2·22	3,636,341	3,694,522	$11,329,291	9,748,480	4,422	$440,129	$ 99·53
1895	2,211,983	2,247,375	4,816,847	2·14	3,682,693	3,741,616	10,753,977	30,965,760	14,046	1,631,251	116·13
1896	2,276,202	2,312,621	5,072,818	2·19	3,626,202	3,684,221	10,998,324	41,426,560	18,790	2,393,914	127·40
1897	2,399,263	2,437,651	5,326,761	2·19	3,698,031	3,757,199	11,163,491	20,303,360	9,209	1,199,029	130·20
1898	3,152,457	3,202,896	6,699,248	2·09	4,503,405	4,575,459	12,412,233	20,928,320	9,492	755,443	79·59

Year.	Copper, Pig, Sheet, and Old.				Copper. Manufactures. Value.	Copper. Total Value.	Earthen, and China Ware.	Glassware.	Gold and Silver in Coin and Bullion. (*e*)	
	Lb.	Metric Tons.	Value.	Val. per Met. Ton.			Value.	Value.	Gold.	Silver.
1894	*d*162,393,000	73,681	$15,324,925	$207·99	$378,040	$16,143,094	$138,289	$917,519	$101,819,924	$47,044,205
1895	*d*121,328,390	55,034	12,222,769	222·09	1,084,289	14,938,309	139,297	1,002,328	104,605,023	53,833,153
1896	259,223,924	111,758	27,822,280	248·95	819,017	31,035,211	169,343	1,115,252	58,047,269	63,063,336
1897	277,255,742	125,763	30,627,945	243·54	958,379	32,785,353	207,603	1,234,558	34,174,182	58,352,274
1898	291,955,905	132,430	33,598,869	253·71	1,190,939	35,545,251	251,821	1,289,919	16,113,258	53,673,605

Year.	Gold and Silver in Ores. (*f*)		Iron Ore.				Iron, Pig.			
	Gold.	Silver.	Long Tons.	Metric Tons.	Value.	Value per Met. Ton.	Long Tons.	Metric Tons.	Value.	Value per Met. Ton.
1894......	$158,765	$72,648	....	....	....	....	24,480	24,872	$309,222	$12·43
1895......	362,379	377,933	1,549	1,573	$7,654	$4·86	26,164	26,584	371,297	13·96
1896......	209,621	993,405	11,016	11,192	38,916	3·48	62,071	63,064	943,022	14·95
1897......	102,219	309,018	7,583	7,704	24,612	3·19	262,686	266,889	3,269,010	12·25
1898......	81,696	123,499	31,579	32,084	67,548	2·11	253,077	257,126	2,702,551	10·51

Year.	Iron, Bar.				Iron, Band, Hoop and Scroll.				Iron, Nails and Spikes, Cut.			
	Long Tons.	Metric Tons.	Value.	Value per Met. Ton.	Long Tons.	Metric Tons.	Value.	Value per Met. Ton.	Lb.	Metric Tons.	Value.	Val. per Metric Ton.
1894	3,195	3,248	$130,374	$43·22	99	101	$5,380	$53·27	18,321,904	8,311	$331,133	$39·84
1895	3,329	3,383	147,798	43·68	198	201	8,169	40·64	17,639,414	8,001	339,825	42·47
1896	3,505	3,561	168,680	47·37	268	272	12,303	45·23	23,708,847	10,754	468,758	43·59
1897	4,493	4,566	150,897	33·05	1,424	1,446	44,754	30·95	33,773,216	15,319	670,709	43·78
1898	7,074	7,188	241,499	33·60	1,593	1,619	58,731	36·28	35,247,266	15,988	641,779	40·20

Year.	Iron Nails, Spikes, Wire, Wrought, Horseshoe, and all other, inc. Tacks.				Plates and Sheets of Iron.				Plates and Sheets of Steel.			
	Lb.	Metric Tons.	Value.	Val. per Met. Ton.	Lb.	Metric Tons.	Value.	Value per Met. Ton.	Lb.	Metric Tons.	Value.	Val. per Metric Ton.
1894	3,892,015	1,765	$196,175	$111·15	4,987,830	2,262	$111,586	$49·33	1,856,832	842	$53,641	$63·17
1895	5,301,208	2,304	239,610	104·00	946,470	429	32,170	74·98	1,818,319	825	60,889	73·80
1896	9,563,812	4,338	362,201	83·51	1,725,779	784	48,905	62·37	4,144,127	1,880	96,568	51·37
1897	20,070,024	9,104	550,654	60·49	9,061,447	4,110	175,799	42·77	11,364,821	5,155	173,567	33·66
1898	35,410,339	16,062	839,299	52·25	10,203,903	4,628	204,170	44·12	60,647,662	27,510	787,245	28·62

Year.	Rails of Iron.				Rails of Steel.				Wire.			
	Long Tons.	Metric Tons.	Value.	Value per Met. Ton.	Long Tons.	Metric Tons.	Value.	Value per Met. Ton.	Lb.	Met. Tons.	Value.	Value per Met. Ton.
1894	1,327	1,348	$32,306	$24·34	12,229	12,425	$323,880	$26·07	59,386,012	26,937	$1,304,794	$48·44
1895	6,792	6,900	139,470	20·21	8,807	8,948	222,661	24·88	66,024,358	29,948	1,375,195	45·90
1896	628	638	18,581	28·18	72,503	73,663	1,712,716	23·30	85,216,157	38,655	1,788,095	46·23
1897	5,418	5,499	95,520	17·37	142,808	145,093	2,949,901	20·33	118,887,872	53,927	2,353,829	43·65
1898	10,865	11,039	152,189	13·79	291,038	295,695	5,787,384	19.57	167,306,882	75,890	3,040,226	40·06

Year.	Lead Manufactures.	Marble, Stone, & Manufactures of.	Mica.	Nickel. (*g*)	Petroleum (1 — 1,000 in Quantities and total Values).							
					Crude.				Naphtha.			
	Value.	Value. (*k*)	Value.	Value.	Gallons.	Liters.	Value.	Value per Liter.	Gallons.	Liters.	Value.	Value per Liter.
1894	$497,993	$1,009,704	$659	$495,218	114,825	434,612	$4,661	$0·010	14,915	56,453	$912	$0·016
1895	214,856	959,871	*Nil.*	239,897	116,108	439,469	6,286	·014	12,922	48,910	1,000	·024
1896	601,873	1,156,051	*Nil.*	606,833	118,133	447,133	6,082	·013	13,641	51,631	1,128	·021
1897	433,319	1,682,044	*Nil.*	997,391	121,864	461,255	5,044	·011	13,704	51,869	1,020	·020
1898	215,239	1,842,220	6,272	1,359,609	120,436	455,888	5,016	·011	17,258	65,326	1,071	·016

Year.	Petroleum—*Continued.*											
	Illuminating.				Lubricating.				Residue, etc.			
	Gallons	Liters.	Value.	Value per Liter.	Gallons.	Liters.	Value.	Value per Liter.	Gallons (*j*)	Liters.	Value.	Value per Liter.
1894	734,057	2,778,406	$30,299	$0·019	39,946	151,195	$5,369	$0·035	119	450	$10	$0·022
1895	686,006	2,596,583	43,540	·017	47,877	180,214	6,289	·035	170	643	14	·022
1896	758,076	3,869,317	49,704	·013	51,705	195,719	6,771	·035	521	1,972	28	·014
1897	804,446	3,044,828	46,876	·015	52,659	199,332	6,782	·034	12,247	46,355	335	·007
1898	764,823	2,895,093	38,895	*·013*	65,526	248,038	7,626	·031	30,436	115,209	815	·007

Year.	Petroleum—*Continued.*				Quicksilver.				Tin Manufactures. Value.
	Paraffine.								
	Lb.	Metric Tons.	Value.	Value per Met. Ton.	Lb.	Metric Tons.	Value.	Value per Met. Ton.	
1894	85,125	38·6	$3,277	$84·35	1,102,187	500	$397,528	$795·05	$301,508
1895	114,234	51·9	4,505	86·79	1,188,955	504	482,085	956·91	252,223
1896	112,517	51·0	4,563	89·41	1,525,726	692	618,437	895·02	268,581
1897	136,069	61·7	5,284	85·61	1,007,770	457	394,549	863·35	284,020
1898	166,317	75·4	6,363	84·34	981,497	445	440,587	990·08	281,794

Year.	Zinc Ore and Oxide.				Zinc Sheets, Pigs, Bars.				Manufactures. Value.	Total Value.
	Lb.	Metric Tons.	Value.	Value per Met. Ton.	Lb.	Metric Tons.	Value.	Value per Met. Ton.		
1894	....	....	$5	....	3,607,050	1,610	$144,074	$89·49	$99,406	$243,485
1895	53,760	24	1,008	$42·00	3,060,805	1,388	153,175	110·35	50,051	204,234
1896	4,648,000	2,108	47,408	22·49	20,300,169	9,208	1,013,620	110·08	51,001	1,112,029
1897	18,502,400	8,393	211,350	25·18	28,490,662	12,923	1,356,538	104·97	71,021	1,638,909
1898	23,564,800	10,689	299,870	28·05	20,998,413	9,525	1,033,959	108·55	141,932	1,475,761

MINERAL EXPORTS OF FOREIGN PRODUCE FROM THE UNITED STATES. (*a*)

Year.	Asphaltum or Bitumen (Crude.)				Brass and Manufactures of. Value.	Cement.			
	Long Tons.	Metric Tons.	Value.	Value per Met. Ton.		Lb.	Metric Tons.	Value.	Value per Met. Ton.
1894	502	510	$9,211	$18·06	$6,909	3,889,928	1,764	$15,072	$8·54
1895	72	73	1,906	26·11	8,711	3,578,230	1,623	13,895	8·56
1896	113	115	2,615	22·74	3,488	4,437,261	2,013	16,558	8·22
1897	40	41	1,042	25·41	3,210	2,488,350	1,129	9,705	8·60
1898	1,514	1,538	28,666	18·64	1,080	6,581,590	2,985	24,312	8·14

Year.	Chemicals.											
	Salts of Potash. (*h*)				Chloride of Lime.				Nitrate of Soda.			
	Lb.	Kg.	Value.	Value per Kg.	Lb.	Kg.	Value.	Value per Kg.	Lb.	Kg.	Value.	Value per Kg.
1894	719,361	326,529	$14,436	$0·04	19,943	9,046	$550	$0·06	1,229,760	557,819	$19,819	$0·04
1895	379,767	172,260	12,555	·07	10,600	4,808	305	·06	2,428,160	1,101,413	44,847	·04
1896	774,452	351,291	22,333	·06	5,500	2,495	105	·04	1,462,720	663,490	24,254	·04
1897	124,277	56,372	4,763	·08	40,916	18,559	718	·04	2,022,720	917,506	33,979	·04
1898	183,804	82,466	5,444	·07	28,605	12,975	381	·03	1,646,400	746,802	24,825	·03

Year.	Chemicals—*Continued.*											
	Caustic Soda.				Sal Soda and Soda Ash.				All Other Salts of Soda.			
	Lb.	Kg.	Value.	Value per Kg.	Lb.	Kg.	Value.	Value per Kg.	Lb.	Kg.	Value.	Value per Kg.
1894	2,929,473	1,328,809	$66,017	$0·05	101,886	46,216	$1,305	$0·03	131,118	59,475	$5,648	$0·09
1895	1,592,150	722,199	31,691	·04	269,765	122,365	2,423	·02	68,127	30,902	1,174	·04
1896	1,703,173	772,559	33,433	·04	470,102	213,238	4,098	·02	36,839	16,710	380	·02
1897	1,824,565	827,623	34,332	·04	2,246,181	1,018,868	9,128	·01	20,797	9,433	262	·03
1898	1,237,257	561,216	22,202	·04	4,247,945	1,926,855	14,255	·01	120,542	54,677	1,400	·03

Year.	Clays or Earths of All Kinds, including China Clay.				Coal, Bituminous.				Copper.			
									Copper Ore and Regulus.			
	Long Tons	Metric Tons.	Value	Value per Met. Ton.	Long Tons.	Metric Tons.	Value	Value per Met. Ton.	Lb.	Metric Tons.	Value.	Val. per Met. Ton.
1894	92	94	$616	$8·67	877	891	$4,984	$5·54	566,720	257	$36,988	$143·92
1895	89	90	740	8·22	3,440	3,495	7,699	2·20	2,457,280	1,115	111,007	99·56
1896	32	33	109	3·30	5,204	5,287	7,990	1·51	1,075,200	488	32,715	67·04
1897	16	16	63	3·93	5,207	5,382	6,337	1·18	1,328,320	608	85,170	141·24
1898	166	169	657	3·89	2,890	2,936	2,675	0·91	7,622,720	3,458	547,960	158·46

Year.	Copper—*Continued.*					Earthen, Stone, and China-ware. Value.	Fertilizers.			
	Pigs, Bars, Ingots, Old, and Other Unmanufactured.				Manufactures. Value.		Guano.			
	Lb.	Metric Tons.	Value.	Value per Met. Ton.			Long Tons.	Metric Tons.	Value.	Value per Met. Ton.
1894.....	1,150,160	521	$78,544	$ 150·75	$13,056	$19,541	106	107	$1,620	$15·18
1895.....	475,588	215	43,193	200·89	11,626	25,482	....	....	....	....
1896.....	*Nil.*	....	....	....	7,307	28,267	26	26	657	25.27
1897.....	406,598	184	30,167	163·93	4,225	23,438	*Nil.*	....	....	....
1898.....	23,647,968	10,727	1,437,464	134·00	4,637	29,305	*Nil.*	....	....	....

Year.	Fertilizers—*Continued.*					Glass and Glassware.	Graphite.			
	Phosphates, Crude or Native.				Other Fert'z's					
	Long Tons.	Metric Tons.	Value.	Value per Met. Ton.	Value.	Value.	Long Tons.	Metric Tons.	Value.	Value per Met. Ton.
1894............	....	....	....	....	$8	$26,509	43	44	$4,371	$99·33
1895............	743	755	$12,775	$16·92	2,154	15,308	5	5	305	61·00
1896............	*Nil.*	....	....	....	1,111	13,761	·6	6	296	46·00
1897............	65	66	1,782	27·00	3,482	15,579	128	130	7,898	56·87
1898............	10	10	306	30·12	13,499	13,034	165	168	11,645	69·32

Year.	Iron and Steel, and Tin Plate.											
	Pig Iron.				Scrap Iron and Steel, fit only to be remanufactured.				Bar Iron, Rolled or Hammered.			
	Long Tons.	Metric Tons.	Value.	Value per Met. Ton.	Long Tons.	Metric Tons.	Value.	Value per Met. Ton.	Lb.	Metric Tons.	Value.	Value per Met. Ton.
1894	83	84	$962	$11·45	1,526	1,550	$18,422	$11·88	72,160	33	$1,606	$48·66
1895	210	213	2,536	11·89	92	94	1,024	10·89	8,207	3	288	96·00
1896	599	609	12,220	20·06	240	244	2,160	8·85	19,107	9	610	67·77
1897	114	116	1,394	12·00	57	58	313	5·40	66,345	30	1,699	56·60
1898	581	590	7,098	12·03	63	64	270	4·22	48,151	22	1,648	74·91

Year.	Iron and Steel, and Tin Plate—*Continued.*											
	Railway Bars of Iron or Steel, or in Part of Steel.				Ingots, Blooms, Slabs, Billets and Bars of Steel, and Steel in Forms, N. E. S.				Sheet, Plate and Taggers, Wire Rod, Wire, and Wire Rope and Strand, Iron or Steel.			
	Long Tons.	Metric Tons.	Value.	Value per Met. Ton.	Lb.	Metric Tons.	Value.	Value per Met. Ton.	Lb.	Metric Tons.	Value.	Value per Met. Ton.
1894	546	555	$10,449	$18·82	64,050	29	$8,094	$286·00	234,857	107	$5,141	$48·04
1895	204	207	3,755	18·14	43,148	19	2,499	131·52	516,067	235	12,026	51·17
1896	*Nil.*	....	....	....	*Nil.*	....	....	....	3,102,098	1,407	46,381	32·96
1897	262	266	11,921	44·82	15,515	7	1,728	246.86	2,446,012	1,109	68,878	62·11
1898	140	142	3,479	24·50	100,417	46	3,829	83·24	824,788	374	21,231	56·77

Year.	Iron and Steel, and Tin Plate—*Continued.*					Lead and Manufactures of. Value.	Marble and Stone and Manufactures of. Value. (*k*)	Metal Compositions and Manufactures. Value.	Mineral Substances, N. E. S. Value.
	Tin Plates, Terne Plates, and Taggers Tin.				Manufactures. Value.				
	Lb.	Metric Tons.	Value.	Value per Met. Ton.					
1894.........	920,426	418	$29,402	$70·34	$175,727	$3,316,481	$18,832	$270,557	$7,196
1895.........	666,458	302	18,446	61·08	126,576	713,271	6,998	66,389	4,655
1896.........	1,674,187	760	40,920	53·84	114,472	1,504,937	14,227	43,757	....
1897.........	2,951,473	1,339	79,179	59·21	165,710	2,088,923	15,156	32,791	....
1898.........	899,757	408	22,271	54·59	279,466	2,553,289	5,452	58,814	....

Year.	Oil, Mineral.				Paints and Colors. Value.	Precious Stones. Value.	Salt.			
	Gallons.	Liters.	Value.	Value per Liter.			Lb.	Met. Tons.	Value.	Value per M. Ton.
1894.........	40,350	152,725	$12,291	$0·08	$9,180	$7,071	4,693,678	2,129	$5,618	$2·63
1895.........	23	87	43	.49	24,147	22,967	3,116,595	1,414	6,500	4·59
1896.........	*Nil.*	....	....	....	23,703	18,979	7,038,134	3,192	14,947	4·68
1897.........	*Nil.*	....	....	....	15,473	26,698	5,074,995	2,302	10,189	4·43
1898.........	3,219	12,185	486	·04	15,854	39,974	4,827,288	2,189	4,751	2·17

Year.	Sulphur or Brimstone (Crude).				Tin in Bars, Blocks, Pigs, or Grain, or Granulated.				Zinc and M'f'res.
	Long Tons.	Met. Tons.	Value.	Value per M. Ton.	Lb.	Met. Tons.	Value.	Value per Met. Ton.	Value.
1894.......	......	....	....	....	168,484	76	$26,537	$349·17	$216
1895.......	220	224	$3,685	$16·45	395,456	134	63,584	474·50	2,875
1896.......	484	492	8,535	17·34	289,266	131	41,874	319·64	215
1897.......	194	197	3,982	20·20	869,929	395	118,882	300·97	2,291
1898.......	1,414	1,437	31,822	22·14	740,326	336	113,334	337·30	1,536

(*a*) From *Summary of Commerce and Finance of the United States.*.

(*b*) Includes lime.

(*c*) Ore, so called, consisting chiefly of matte.

(*d*) Sheets are not included these years, but are reported with manufactures.

(*e*) Total exports of coin and bullion; that is, includes both domestic and foreign not elsewhere specified.

(*f*) Only approximately correct. The Bureau of Statistics reports only the value of silver ores exported, but a much larger amount of silver leaves the country in copper matte which is classified as copper ore and no record is kept of its silver contents. The gold in copper matte exported is not included in the exports of gold given in the above table. These figures include ore of both domestic and foreign origin.

(*g*) Including nickel oxide and matte.

(*h*) Includes chlorate, muriate, and nitrate of potash, and all other salts of potash.

(*j*) Reported in bbls. and reduced to gals. at 42 gals. = 1 bbl.

(*k*) Including slate.

(*m*) Calculated at 1 bbl. = 400 lb.

(*n*) Not enumerated.

INDEX.

The Buyers' Manual

Preface.

THE advertising pages of the annual volumes of this work will well repay the careful perusal and study of every reader who wishes to be well informed upon the present condition of the mineral industry. They give an admirable and practical insight into the present state of the mining and metallurgic arts, for in them nearly every manufacturer or dealer of note in this country advertises the machines, appliances, and processes which are now in vogue, or which it is sought to introduce; while the names and specialties of the most eminent members of the engineering professions, as indicated in their cards, show the direction of modern mining and metallurgical progress.

These advertising pages are no less important to those who desire a clear knowledge of the means by which this country has come to be far the most important producer of minerals and metals than to those who wish to know where to get that full and reliable information concerning the values of properties, machinery, processes, and products which should precede the investment of capital.

Every country in the world is wisely striving to develop its mineral resources, and to build up its mineral industry, and in all of them this volume, which gives the latest and best practice in every department of the industry, has become indispensable. It is constantly consulted for the best technical skill and the most advantageous machinery and appliances in use. All enterprising manufacturers of such wares appreciate this fact, and know that in no other way can they so effectively bring their goods before those who may need them as through the advertising pages of THE MINERAL INDUSTRY. These pages have become a veritable directory of the best in everything relating to the industry, and American, German and English manufacturers there compete for the orders of the whole world. $1,000,000,000 a year is certainly not an overestimate of the annual expenditure for technical skill, machinery and supplies used in the industry of which THE MINERAL INDUSTRY and *The Engineering and Mining Journal* are not only the chief but the only universal representatives published in any language.

The Publishers.

Buyers' Manual of The Mineral Industry.

Alphabetical Index.

Buyers' Manual of The Mineral Industry.

Classified Index.

Air and Gas Compressors.

Allis, Edw. P., Co.
Bullock, M. C., Mfg. Co.
Krupp, Fried., Grusonwerk.
Mine and Smelter Supply Co.
Norwalk Iron Works.
Rand Drill Co.
(See Machinery—Ventilating Machinery.)

Air Lift Pumps.

Allis, Edw. P., Co.
Mine and Smelter Supply Co.
Rand Drill Co.

Aluminum.

(See Metals.)

Amalgamators.

Allis, Edw. P., Co.
Krupp, Fried., Grusonwerk.
Mine and Smelter Supply Co.
(See Machinery.)

Anti-Friction Metals.

Mine and Smelter Supply Co.
(See Metals.)

Antimony.

(See Metals.)

Artists' Materials.

Devoe, F. W., & C. T. Raynolds Co.

Artesian Wells.

(See Machinery.)

Assayers.

Ricketts & Banks.
(See Chemists.)

Assayers' and Chemists' Supplies.

Baker & Co.
Hoskins, Wm.
Mine and Smelter Supply Co.

Assay Furnaces.

Allis, Edw. P., Co.
Hoskins, Wm.
Fraser & Chalmers, Ltd.
Krupp, Fried., Grusonwerk.
Mine and Smelter Supply Co.

Babbitt Metal.

Mine and Smelter Supply Co.
(See Metals.)

Ballast Unloaders.

Lidgerwood Mfg. Co.

Barytes.

(See Paints.)

Batteries (Stamp).

Allis, Edw. P., Co.
Fraser & Chalmers, Ltd.
Krupp, Fried., Grusonwerk.
Mine and Smelter Supply Co.
(See Machinery.)

Battery Plates.

Allis, Edw. P., Co.
Fraser & Chalmers, Ltd.
Krupp, Fried., Grusonwerk.
Matthiessen & Hegeler Zinc Co.
Mine and Smelter Supply Co.
(See Amalgamators.)

Battery Screens.

(See Perforated Metals—Screens.)

Battery Zincs.

Matthiessen & Hegeler Zinc Co.

Belting (Link Belts, Chain Belts).

Allis, Edw. P., Co.
Krupp, Fried., Grusonwerk.
Jeffrey Mfg. Co.
Mine and Smelter Supply Co.

Blast Furnaces.

(See Furnaces.)

Blasting Supplies.

Allis, Edw. P., Co.
Fraser & Chalmers, Ltd.
Mine and Smelter Supply Co.

Blowers and Exhausters.

Allis, Edw. P., Co.
Fraser & Chalmers, Ltd.
Bacon, Earle C.
Krupp, Fried., Grusonwerk.
Mine and Smelter Supply Co.
(See Air Compressors, Fans, Machinery, Ventilators)

Blowpipes—Hydrocarbon.

Hoskins, Wm.

Boilers.

Allis, Edw. P., Co.
Fraser & Chalmers, Ltd.
Bacon, Earle C.
Krupp, Fried., Grusonwerk
Bullock, M. C., Mfg. Co.
Lidgerwood Mfg. Co.
Mine and Smelter Supply Co.
(See Machinery.)

Books.

(See Publications.)

Boots.

Jeffrey Mfg. Co.

Brick Machinery.

Allis, Edw. P., Co.
Fraser & Chalmers, Ltd.
Mine and Smelter Supply Co.

Brushes, Paint.

Devoe, F. W., & C. T. Raynolds Co.

Buckets (Ore and Water).

Allis, Edw. P., Co.
Fraser & Chalmers, Ltd.
Jeffrey Mfg. Co.
Mine and Smelter Supply Co.

Cable Conveyors, Cableways, Cable Tramways, Ropeways.

Allis, Edw. P., Co.
Fraser & Chalmers, Ltd.
Jeffrey Mfg. Co.
Mine and Smelter Supply Co.

Cables.

Allis, Edw. P., Co.
Jeffrey Mfg. Co.
Fraser & Chalmers, Ltd.
Roeblings, J. A., Sons Co.

Cages and Skips.

Allis, Edw. P., Co.
Bacon, Earle C.
Fraser & Chalmers, Ltd.
Jeffrey Mfg. Co.
Krupp, Fried., Grusonwerk.
Mine and Smelter Supply Co.
(See also Machinery.)

Cams.

Allis, Edw. P., Co.
Fraser & Chalmers, Ltd.
Krupp, Fried., Grusonwerk.
Mine and Smelter Supply Co.
(See Machinery.)

Cars.

Allis, Edw. P., Co.
Bacon, Earle C.
Fraser & Chalmers, Ltd.
Jeffrey Mfg. Co.
Krupp, Fried., Grusonwerk.
Mine and Smelter Supply Co.
(See Machinery.)

Car Unloaders.

Lidgerwood Mfg. Co.

Car Wheels.

Mine and Smelter Supply Co.
(See Cars.)

Castings.

Allis, Edw. P., Co.
Bacon, Earle C.
Fraser & Chalmers, Ltd.
Krupp, Fried., Grusonwerk.
(See Machinery—Metals, Iron, Steel, and Brass.)

Chain Belting.

Jeffrey Mfg. Co.
(See Belting.)

Chains.

Allis, Edw. P., Co,
Bacon, Earle C.
Fraser & Chalmers, Ltd.
Jeffrey Mfg. Co.
Krupp, Fried., Grusonwerk.
Mine and Smelter Supply Co.

Chemical Works' Machinery.

(See Machinery.)

Chemists.

Ricketts & Banks.
(See Assayers.)

Chemists' Apparatus and Supplies.

Baker & Co.
Hoskins, Wm.
Mine and Smelter Supply Co.

Chlorination Plants.

Allis, Edw. P., Co.
Fraser & Chalmers, Ltd.
Krupp, Fried., Grusonwerk.
Mine and Smelter Supply Co.
(See Machinery.)

Chrome Steel.

Allis, Edw. P., Co.
Fraser & Chalmers, Ltd.
Krupp, Fried., Grusonwerk.
Mine and Smelter Supply Co.
(See Metals.)

Classifiers (Hydraulic).

Allis, Edw., P. Co.
Fraser & Chalmers, Ltd.
Krupp, Fried., Grusonwerk.
Mine and Smelter Supply Co.

Clay-Working Machinery.

Allis, Edw. P., Co.
Bacon, Earle, C.
Fraser & Chalmers, Ltd.
Krupp, Fried., Grusonwerk.

Coal.

Berwind-White Coal Mining Co.
Castner, Curran & Bullitt.

Coal-Handling Machinery.

Allis, Edw. P., Co.
Bacon, Earle C.
Fraser & Chalmers, Ltd.
Jeffrey Mfg. Co.
Krupp, Fried., Grusonwerk.
(See Machinery.)

Coal-Mining Machinery.

Allis, Edw. P., Co.
Bacon, Earle C.
Bullock, M. C., Mfg. Co.
Fraser & Chalmers, Ltd.
Jeffrey Mfg. Co.
Krupp, Fried., Grusonwerk.
Mine and Smelter Supply Co.
Rand Drill Co.
(See Machinery.)

Coal-Washing Machinery.

Allis, Edw. P., Co.
Jeffrey Mfg. Co.
Fraser & Chalmers, Ltd.
Krupp, Fried., Grusonwerk.
(See Machinery.)

Coke Ovens.

Allis, Edw. P., Co.
Fraser & Chalmers, Ltd.
Krupp, Fried., Grusonwerk.
(See Machinery.)

Coke-Works Equipments.

Allis, Edw. P., Co.
Fraser & Chalmers, Ltd.
Krupp, Fried., Grusonwerk.
(See Machinery.)

Compressors.

Allis, Edw. P., Co.
Bullock, M. C., Mfg. Co.
Fraser & Chalmers, Ltd.
Krupp, Fried., Grusonwerk.
Mine and Smelter Supply Co.
Norwalk Iron Works.
Rand Drill Co.
(See Machinery.)

Concentrators.

Allis, Edw. P., Co.
Fraser & Chalmers, Ltd.
Krupp, Fried., Grusonwerk.
Mine and Smelter Supply Co.
(See Machinery.)

Condensers.

(See Machinery.)

Contractors' Supplies.

Allis, Edw. P., Co.
Bacon, Earle C.
Bullock, M. C., Mfg. Co.
Jeffrey Mfg. Co.
Krupp, Fried., Grusonwerk.
Mine and Smelter Supply Co.
Rand Drill Co.

Converters.

Allis, Edw. P., Co.
Fraser & Chalmers, Ltd.
Krupp, Fried., Grusonwerk.
Mine and Smelter Supply Co.
(See Machinery.)

Conveying Belts.

Allis, Edw. P., Co.
Jeffrey Mfg. Co.
Mine and Smelter Supply Co.
(See Elevating and Conveying Machinery.)

Conveying Machinery.

(See Elevating and Conveying Machinery.)

Copper.

(See Metals.)

Copper Bottoms.

Lewisohn Bros.
(See Metals.)

Copper Dealers and Producers.

(See Metals—Copper.)

Copper Sulphate.

American Smeltiug and Refining Co.

Cranes.

(See Elevating and Conveying Machinery.)

Crucibles.

Hoskins, Wm.
Mine and Smelter Supply Co
(See Assayers' and Chemists' Supplies.)

Crusher Plants.

Allis, Edw. P., Co.
Bacon, Earle C.
Fraser & Chalmers, Ltd.
Krupp, Fried., Grusonwerk.
Mine and Smelter Supply Co.
(See MACHINERY.)

Crusher Rolls.

Allis, Edw. P., Co.
Bacon, Earle C.
Fraser & Chalmers, Ltd.
Krupp, Fried., Grusonwerk.
Mine and Smelter Supply Co.
(See MACHINERY.)

Crushers.

Allis, Edw. P., Co.
Bacon, Earle C.
Fraser & Chalmers, Ltd.
Krupp, Fried., Grusonwerk.
Mine and Smelter Supply Co.
(See MACHINERY.)

Cyanide Mills.

Allis, Edw. P., Co.
Fraser & Chalmers, Ltd.
Krupp, Fried., Grusonwerk.
Mine and Smelter Supply Co.
(See MACHINERY.)

Cyanide Tanks.

Allis, Edw. P., Co.
Mine and Smelter Supply Co.

Diamond Drills.

Allis, Edw. P., Co.
Bullock, M. C., Mfg. Co.
Fraser & Chalmers, Ltd.
Mine and Smelter Supply Co.
(See DRILLS.)

Disintegrators.

Allis, Edw. P., Co.
Bacon, Earle C.
Fraser & Chalmers, Ltd.
Jeffrey Mfg. Co.
Krupp, Fried., Grusonwerk.
Mine and Smelter Supply Co.
(See PULVERIZERS.)

Drawing Materials.

Devoe, F. W., & C. T. Raynolds Co.
(See MATHEMATICAL INSTRUMENTS.)

Dredging Machinery.

Allis, Edw. P., Co.
Jeffrey Mfg. Co.
Fraser & Chalmers, Ltd.
Mine and Smelter Supply Co.
(See MACHINERY.)

Drills.

Allis, Edw. P., Co.
Bullock, M. C., Mfg. Co.
Fraser & Chalmers, Ltd.
Jeffrey Mfg. Co.
Krupp, Fried., Grusonwerk.
Mine and Smelter Supply Co.
Rand Drill Co.
(See MACHINERY.)

Drill Steel.

(See METALS—MINE SUPPLIES.)

Drying and Calcining Machinery.

Allis, Edw. P., Co.
Fraser & Chalmers, Ltd.
Jeffrey Mfg. Co.
Krupp, Fried., Grusonwerk.
Mine and Smelter Supply Co.
(See MACHINERY.)

Dumping Cars.

(See CARS.)

Dynamite.

(See EXPLOSIVES.)

Dynamos.

Allis, Edw. P., Co.
Fraser & Chalmers, Ltd.
Jeffrey Mfg. Co.
Mine and Smelter Supply Co.
(See MACHINERY.)

Electric Coal Mining Machines.

Allis, Edw. P., Co.
Bacon, Earle C.
Bullock, M. C., Mfg. Co.
Fraser & Chalmers, Ltd.
Jeffrey Mfg. Co.
Krupp, Fried., Grusonwerk.
Mill and Smelter Supply Co.
(See COAL MINING MACHINERY.)

Electric Machinery and Supplies.

(See MACHINERY.)

Elevating and Conveying Machinery.

Allis, Edw. P., Co.
Bacon, Earle C.
Bullock, M. C., Mfg. Co.
Fraser & Chalmers, Ltd.
Jeffrey Mfg. Co.
Krupp, Fried., Grusonwerk.
Lidgerwood Mfg. Co.
Mine and Smelter Supply Co.
(See MACHINERY.)

Engineering Instruments.

Devoe, F. W., & C. T. Raynolds Co.

Engines.

Allis, Edw. P., Co.
Bacon, Earle C.
Bullock Mfg. Co.
Fraser & Chalmers, Ltd.
Jeffrey Mfg. Co.
Krupp, Fried., Grusonwerk.
Lidgerwood Mfg. Co.
Mine and Smelter Supply Co.
Rand Drill Co.
Weber Gas and Gasoline Engine Co.
(See MACHINERY.)

Engravers.

Poates, L. L.

Etching Plates.

Matthiessen & Hegeler Zinc Co.

Excavating Machinery.

(See STEAM SHOVELS—MACHINERY.)

Explosives.

Allis, Edw. P., Co.
Fraser & Chalmers, Ltd.
Mine and Smelter Supply Co.

Fans.

(See MACHINERY AND VENTILATING MACHINERY.)

Feeders.

Allis, Edw. P., Co.
Bacon, Earle C.
Fraser & Chalmers, Ltd.
Jeffrey Mfg. Co.
Krupp, Fried., Grusonwerk.
Mine and Smelter Supply Co.
(See ORE FEEDERS—QUICKSILVER FEEDERS—MACHINERY.)

Ferro-Nickel.

Orford Copper Co.
(See METALS.)

Filters.

(See MACHINERY.)

Fire Brick.

Hoskins, Wm.
Mine and Smelter Supply Co.

Forgings.

(See METALS—IRON AND STEEL.)

Franklinite Ore.

New Jersey Zinc Co.
(See Ore and Zinc Ore.)

Furnaces.

Allis, Edw. P., Co.
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Hoskins, Wm.
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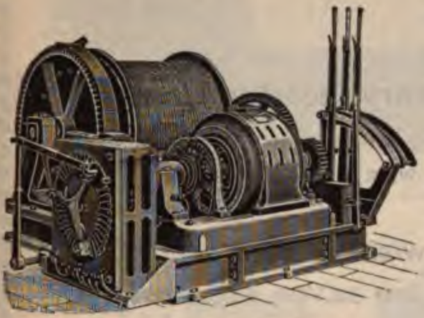

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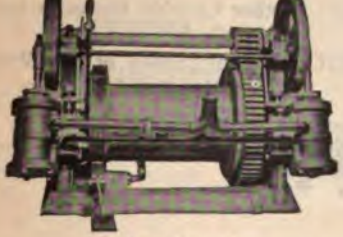

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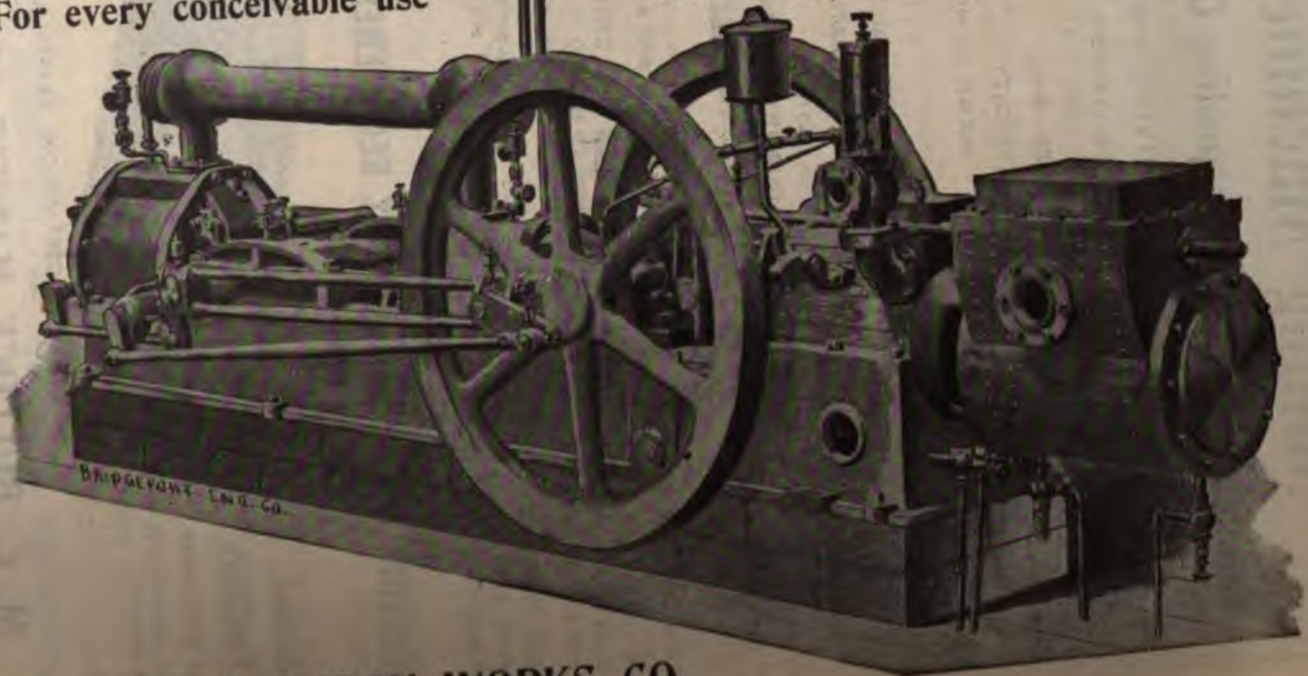

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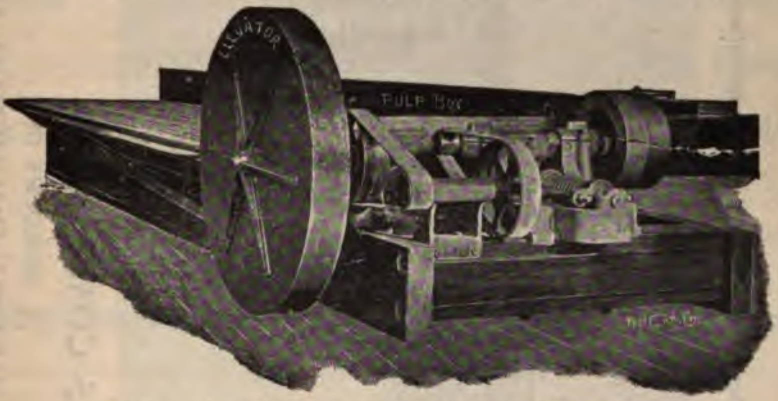

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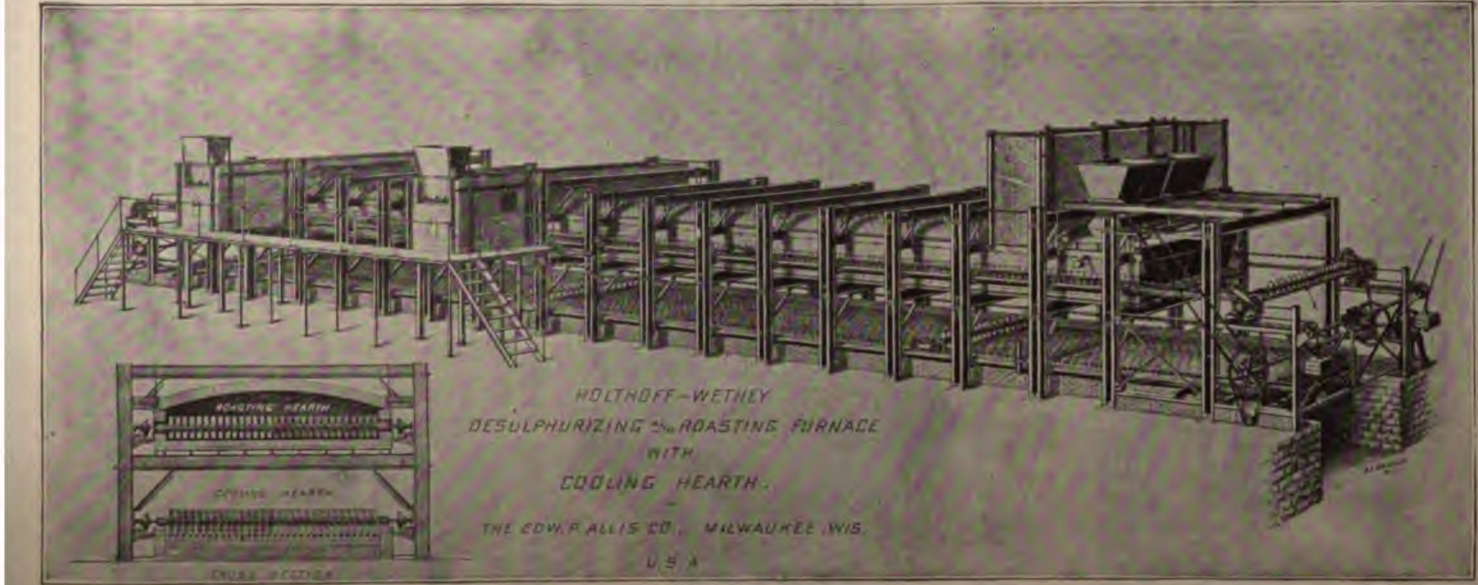

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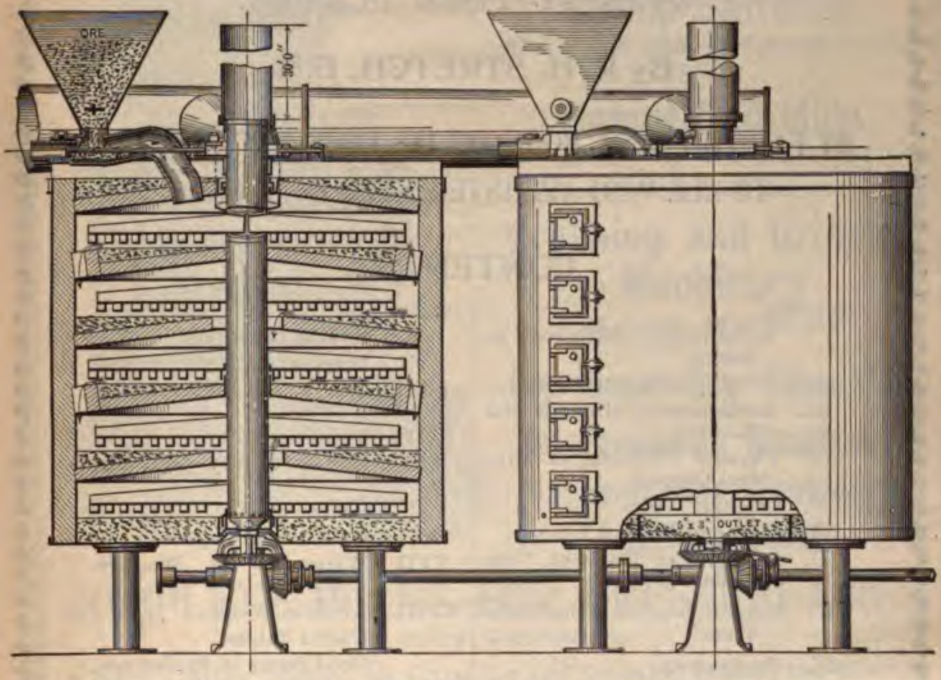

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VIII. Mineral Deposits Other than Veins.
IX. Prospecting.

CHAPTER
X. Making Locations.
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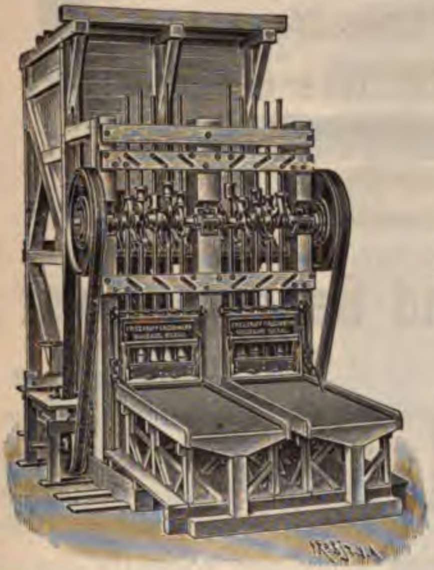

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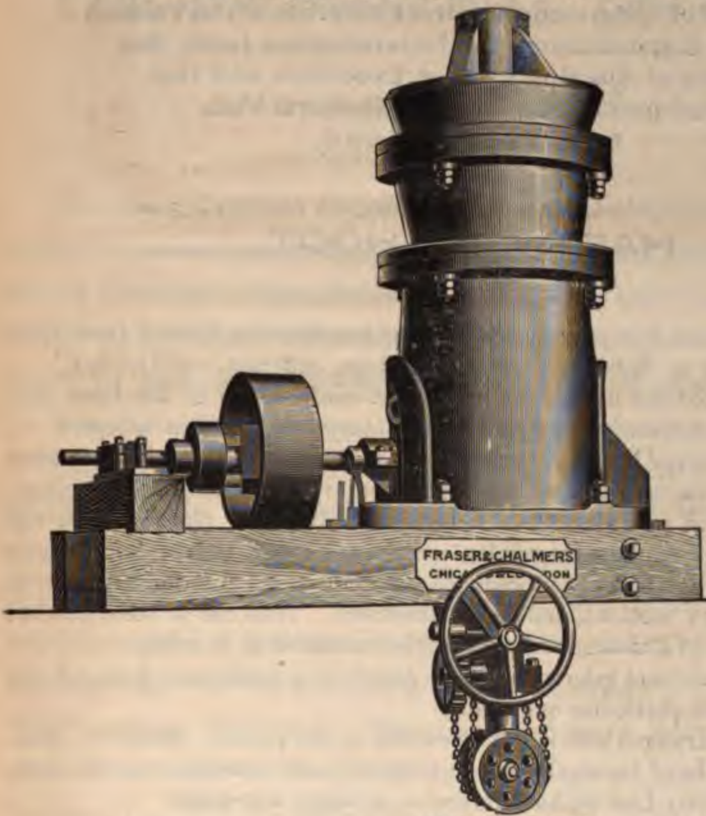

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